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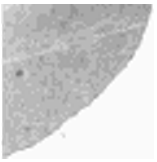

MINING ENGINEERS HANDBOOK

WRITTEN BY A STAFF OF SPECIALISTS
UNDER THE EDITORSHIP OF

ROBERT PEELE

PROFESSOR OF MINING ENGINEERING IN THE SCHOOL
OF MINES, COLUMBIA UNIVERSITY

FIRST EDITION

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PREFACE

There is a considerable literature of mining, comprising
graphs, papers published in the transactions of engi
tments of the mining periodicals. The treatises and
ive, and are intended chiefly for students. Among
of Foster, Hughes, Haton de la Goupillière, Köhler, C
son, Bulman and Redmayne, Bailes, Boulton, and Pam
books are antiquated in their engineering features, s
those of Callon and Haton) contain much that is still
neers would do well to have acquaintance with them.
Also there are the more recent monographs of Truscott,
Denny, on the Witwatersrand goldfields, Charleton's
ed." Hoover's "Principles of Mining," Finlay's "Cost
ber of useful books on specific subjects relating to mi
nical engineering of mines.

A valid reason for bringing out a new Mining Engineer's
ed in the fact that the two already in existence either
ver, many subjects which constitute important parts
ipment of the present day mining engineer. It will be
ry examination of the following pages, that a handbo
s a greater variety of subject matter than books o
neering, and that the field to be covered is too wide to
fully by a single writer within any reasonable period of
In February and March, 1913, the Editor of this book c
ents, and invited a number of Associate Editors to co
r respective specialties. Besides those sections dealin
deposits, methods of prospecting, exploration and minin
minerals, there are others on certain branches of civil, electr
neering. It may be thought by some that this collater
much space in a book on mining. But, in view of the im
the allied branches of engineering in equipping and opera
Editor believes the allotment of space is reasonable. H
et the demands not only of engineers concerned with th
agement of mines, but also of the large number of thos
with, and greater interest in, the construction details invo
of plant. Therefore, the aim has been to supply such
wer plant, electric transmission and structural design, as
and when in the field and out of reach of his personal
ary. For office use, there is at the end of each section a
these important books and papers on the subjects dealt with
practice, no well-defined boundary exists between the f
ning engineer and the metallurgist. While, under some
ver regions, the mining engineer's functions end with the
ct to delivery to a custom reduction works (mill or smelter
ucing company's plant includes a concentrating mill, amalg
ng works (as at many gold and silver mines), or even a smel
to planning the book, the question arose as to how much sp
e given to the processes of ore treatment. To cover any
e great field of modern metallurgy would be impracticable

the work far beyond the limits of a single volume. Realizing that the urgent need of a companion Handbook of Metallurgy must soon be supplied, it was decided, as a compromise, to furnish condensed summaries of those processes of treatment which are frequently carried on by mining companies themselves. The book, therefore, contains sections on ore-dressing, ore-testing, gold amalgamation, an outline of the cyanide process, the preparation of anthracite, bituminous coal and coke, and a brief résumé of certain facts respecting the selling, purchasing, and metallurgical treatment of ores, that are of immediate interest to the engineer in control of mining operations.

The relatively small space allotted to coal mining is due chiefly to three considerations: first, a Coal-mining Pocketbook is already in existence; second, metal-mining methods are more varied than those for coal, due to the greater diversity in form and occurrence of metalliferous deposits; third, having discussed in Section 10, under Metal-mining Methods, the operations common to nearly all mining, the articles on coal mining are properly confined to the methods and data peculiar to that branch of the industry.

The question of supplying cost data is difficult. A large number of itemized tables are included in the sections on Cost of Mining, Exploitation of Mineral Deposits, Boring, and other subjects, but costs of machines and apparatus are given sparingly throughout the book. This has been judged best, because of frequent price changes, and the great diversity of types of mechanical plant. In any case, to make close estimates, the engineer must apply to the makers for current prices. In some parts of the book, the names of machinery builders have been used freely, but without intention to indicate a preference for the product of any particular maker.

While the Editor has aimed to make the style and arrangement uniform, he has had good reason to realize the difficulty of securing consistency in these matters, considering the heterogeneous nature of the subject matter, and the fact that it has been written or compiled by so large a corps of Associate Editors. In these circumstances, unity and evenness of treatment can hardly be expected, but an endeavor has been made to observe a reasonable proportion between the length of each section and its relative importance. To save space, abbreviations are employed for a few words in common use by engineers, and chemical elements and compounds are generally represented by their symbols.

The thanks of the Editor are due to members of the staff for their painstaking work, in many cases carried on in the intervals between pressing professional engagements in the field, and to the Publishers for their liberal spirit of coöperation in facilitating the preparation of the book. The Editor desires to express his especial appreciation of the valuable suggestions and assistance in revising manuscript and correcting proof, of Professor Edward K. Judd, of the Columbia School of Mines. It was planned to publish this book in 1916. The breaking out of the Great War, about one year after the work was begun, is responsible in large measure for the delay.

ROBERT PEELE

SCHOOL OF MINES,
1917

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SECTION 1

MINERALOGY

BY

ALFRED J. MOSES

PROFESSOR OF MINERALOGY, COLUMBIA UNIVERSITY

IDENTIFICATION OF MINERALS

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IDENTIFICATION OF MINERALS

1. Definitions

On the basis of several thousand analyses the crust of the earth for a of about ten miles is estimated by Clarke, "*Data of Geochemistry*," p. 3 composed almost entirely of compounds of fourteen elements:

	Per cent		Per cent		
Oxygen	49.78	Sodium	2.33	Chlorine	
Silicon	26.00	Potassium	2.28	Carbon	
Aluminium	7.51	Magnesium	2.24	Phosphorus	
Iron	4.21	Hydrogen	0.95	Sulphur	
Calcium	3.39	Titanium	0.37	Total	

These great elements, and the sixty or so others which form the ren fraction of one per cent, occur in approximately one thousand different cal combinations, known as minerals; that is, as *homogeneous substa definite chemical composition, found ready-made in nature, and not dir product of the life or decay of an organism.*

The two conditions in which minerals may occur. A miner other chemical substances, will usually occur either in crystals of charac shapes or in masses made up of many crystals so crowded together tl shapes are not evident, although in each grain of the aggregation the cry structure will be shown by the constancy of the properties in parallel dir and their variation in directions not parallel.

Any mineral may in solidifying fail to assume a crystalline structure, b of too great viscosity, or too rapid cooling, or other cause. If this cond invariable, the mineral is said to be *amorphous*. Opal is the best ex Amorphous minerals are few in number.

2. Identification by Aid of Crystals

The crystalline structure is much used in the identification of minerals, by a consideration of the symmetry and angles and habit of the external or by the study of properties dependent on the structure, particularly cle and the so-called optical characters and etch figures.

Symmetry. In every complete crystal there is some repetition of and similarly grouped faces. By considering this so-called "symmetry" tals may be grouped in divisions, and as all crystals of any one mineral the same grade of "symmetry," they belong to the same symmetry divis

In identifying an axis of symmetry imagine or actually cause the crys revolve about some prominent line through its centre. Note the groupi faces at the initial position. Note whether at any stage of the revoluti crystal faces appear to be all coincident (rarely), or all parallel to the positions of other faces. Or, in other words, note whether groups of fac replaced during the revolution by other groups containing just as many at exactly the angles of the first set. If so, a probable axis of symmetr been determined. If by measurement the angles of one set correspond in and order with those of the other sets, then the existence of the symmetry is confirmed. According to the number of times corresponding groups or recur during a complete revolution about a symmetry axis the axis is kno two-fold, three-fold, four-fold, or six-fold. No other varieties exist.

If a plane so divides the crystal that on each side of that plane ther

...ed the same number of faces at the same angles to it and to each other, ...ane is called a Plane of Symmetry.

...visions or "systems" based on symmetry. The following seven ...es result readily from this partial determination of symmetry, the state- ...s not implying the absence of other symmetry elements:

1.	Isometric.....	More than one axis of three-fold symmetry. (Often also more than one of four-fold.)
2.	Tetragonal.....	One axis of four-fold symmetry and one only.
3.	Hexagonal.....	Rhombohedral division — one axis of three-fold symmetry and one only.
4.	Hexagonal.....	Hexagonal division—one axis of six-fold symmetry.
5.	Orthorhombic.....	Three axes of two-fold symmetry, but nothing higher than two-fold; or one axis of two-fold symmetry at the intersection of two planes of symmetry.
6.	Monoclinic.....	One axis of two-fold symmetry and one only, or one plane of symmetry, or both.
7.	Triclinic.....	Without axes or planes of symmetry.

...tinguishing species by angles. Although different crystals of the ... substance may differ in shape, angles, and number of faces, the angles be- ...*corresponding* faces are constant and characteristic.

...responding faces on the same crystal, or on different crystals of the same substance, ...corresponding or analogous positions with reference to the symmetry axes and usu- ...correspond in lustre and markings. They frequently do not correspond in shape.

...e measuring of a few selected angles will, therefore, usually serve to differ- ...ate the crystal from others in the same symmetry division.

...les may be determined within one or two degrees by a very simple apparatus, ...as the Penfield No. 2 goniometer, consisting of a cardboard on which is printed a ...ded semicircle, with an arm of celluloid swiveled by an eyelet in the centre of the ...ircle, or better a similar apparatus of metal with removable and adjustable arms. ...the crystal is placed so that the card edge and the swinging arm, or the two ...arms, are each in contact with a face and perpendicular to the edge of intersection ...two faces, and the mean of at least three readings is used.

...e "cleavage" directions, obtained as described later, are of great service in orien- ...ing the crystal. These and the angles between them are used in the lists which follow ...them.

...nes are composed of faces all parallel to the same line. Their intersections are ...e parallel to this line and to each other.

...sometric crystals. If a crystal shows more than one axis of three-fold ...metry it is an isometric crystal, and not otherwise. There will always be ...also, axes of two-fold or four-fold symmetry. The faces are often ...s and equilateral triangles, or these modified by cutting off corners. The ...ns are usually approximately equal in several directions, the forms ...ing sometimes to the sphere. Repetitions in any crystal of equal angles ..."corresponding" faces are more frequent than in other crystal systems.

...gles. These are of the same series whatever the species. The important species ...ed by their "habit"; that is, the dominant forms of the crystals, as follows:

...tahedral. (Tetrahedron angles, 70° 31′) boracite, sphalerite, tetrahedrite.

...: *With easy cubic cleavage:* cobaltite, galenite, halite; *with octahedral cleavage:* flu- ...rite; *without marked cleavage:* argentite, boracite, cerargyrite, cuprite, pyrite.

...tahedral. (Octahedron angles, 109° 29′) chromite, cobaltite, cuprite, fluorite, ...galenite, gold, linnæite, magnetite, pyrite, spinel. Cleavages: galenite, ...rite, octahedral. Partings: franklinite and magnetite, octahedral.

Dodecahedral. (Dodecahedron angles, 120°) boracite, cuprite, garnet, m sphalerite.

Trapezohedral. (24-faced trapezohedra, approximating spheres; common 131° 19′, 146° 27′) analcite, garnet, leucite.

Pyritohedral. (12-faced pyritohedra; most common angles, 126° 53′ and cobaltite, pyrite, smaltite.

Tetragonal crystals. If the crystal shows one axis of four-fold sym and only one, it is a tetragonal crystal, and not otherwise. A section t right angles to the four-fold axis is usually square or octagonal, or more the angles are again truncated. The dimension in direction of the fo axis is usually notably greater or less than in directions at right angles t

Angles. In the zone of faces parallel to the four-fold axis there are no varia angle dependent on the species. Between prominent *corresponding* faces the an almost always 90°, and between prominent *adjacent* faces either 90° or 135°. T acterizing angles lie in other zones.

The principal tetragonal minerals may be classified by angles and cleavage as Angles between corresponding faces *oblique to the four-fold axis:* chalcopyrite, wulfenite, 99° 38′; scheelite, 100° 5′; apophyllite, 105°; braunite, 109° 53′; cas 121° 41′; rutile, 123° 8′; zircon, 123° 19′; vesuvianite, 129° 21′; wernerite, 136° Braunite, scheelite, and wulfenite cleave at the angles mentioned. Werneri rutile cleave parallel to the four-fold axis, giving angles of 90° and 135°. Apo cleaves at right angles to the four-fold axis.

Hexagonal crystals. If the crystal shows one and only one axis of fold symmetry it is a hexagonal crystal, rhombohedral division. crystal shows one and only one axis of six-fold symmetry it is a hex crystal, hexagonal division. A section taken at right angles to the three-fold or six-fold symmetry is usually a hexagon, or its most pro edges form an equiangular triangle or a hexagon. Not infrequently each is replaced by one or two smaller edges. The dimension parallel to this usually notably greater or less than the dimensions at right angles thereto.

Angles. In the zone of faces parallel to the three-fold (or six-fold) axis there variations in angle dependent on the species. The angles between prominent sponding faces are chiefly 120° or 60°. Other angles in this zone are usually large an occurrence leads to an apparently rounded, often nearly circular, cross-section. characterizing angles lie in other zones.

The crystals of important hexagonal minerals may be classified by angles betwe responding faces and by cleavage as follows:

I. With evident axis of three-fold symmetry and usually rhombohedral h

Angles which are both interfacial and between cleavage directions. Soda 73° 30′; chabazite, 85° 14′; hematite, 86°; calcite, 105° 5′; dolomite, 106° 15′; chrosite, 107°; siderite, 107°; magnesite, 107° 24′; smithsonite, 107° 40′; proustite, 10

Angles which are interfacial only. Ilmenite, 85° 31′; alunite, 90° 50′; cin 92° 37′; willemite, 115° 30′; phenacite, 116° 36′; tourmaline, 133° 8′ or 103°.

II. With real or apparent axis of six-fold symmetry, and usually prismatic

Prisms capped by faces oblique to axis and at angles, for example, corun 86° 4′ or 128° 2′; quartz, 94° 14′ or 133° 44′; apatite, 142° 15′.

Prisms usually capped by single face at right angles to axis. Beryl, iod mimetite, nephelite, pyrargyrite, pyromorphite, vanadinite.

Tabular. Graphite, molybdenite, iridosmine.

Orthorhombic crystals. If a crystal shows either three axes of two symmetry or one axis with two planes of symmetry, and nothing of hi symmetry, it belongs to the orthorhombic system. Cross-sections take right angles to the axes of symmetry are unlike in angles, and tend to angles and rhombs or to these combined.

Angles. There is no zone of faces which has a constant series of angles for all species. The interfacial angles in the zones parallel to the axes of symmetry are unlike (except rarely 30°, and vary with the species. The orientation is best obtained by reference to the angles, and on this basis the important species may be tabulated as follows:

1. **With one direction of cleavage which bisects prominent angles, for example:** [illegible] 30° 26′; sillimanite, 91° 45′; goethite, 94° 52′; manganite, 99° 40′; brochantite, 103° 32′; atacamite, 113° 03′; staurolite, 129° 20′. Topaz, with one direction of cleavage, has prominent angles 124° 17′ and 90° 11′, not bisected by the cleavage.

2. **Crystals with two directions of cleavage or more than two in one zone, and known angles between faces parallel to two such directions:** columbite and chrysoberyl, 90°; andalusite, 90° 48′; natrolite, 91° 15′; enargite, 97° 53′; calamine, 103° 51′; [illegible]pyrite, 112° 27′; cerussite, 117° 14′; strontianite, 117° 19′; aragonite, 118° 12′; [illegible]ite, 119° 35′.

3. **Crystals with three or more directions of cleavage not in one zone, and known angles between faces parallel to such directions:** anhydrite, 90°; barite and 101° 38′; anglesite, 90° and 103° 44′; celestite, 90° and 104° 10′: stephanite, [illegible] and 107° 44′.

Monoclinic crystals. If a crystal shows one and only one axis of two-fold symmetry, or one and only one plane of symmetry, or both, it is a monoclinic crystal. Any face in the zone of the symmetry axis makes a 90° angle with the symmetry plane (or a face parallel to it). No other 90° angles occur. The cross-section of the zone of the symmetry axis is never rectangular, rarely hexagonal and usually markedly unsymmetrical.

Angles. No zone has a constant series of angles for each species. In this system the symmetry plane, the one symmetry axis and the cleavages, all assist in the orientation according to the following tabulation:

Easiest cleavage	Species	Angles in zone of symmetry axis	Angles bisected by symmetry plane
⊥ to symmetry plane	Colemanite	110° 9′, 111° 36′	107° 56′, 140° 12′, 126° 9′
	Gypsum		131° 30′, 143° 48′, 138° 40′
	Realgar		74° 26′, 132° 3′, 108° 2′
	Vivianite		
	Wolframite	118° 6′, 124° 18′, 117° 6′	100° 37′, 98° 6′, 117° 49′
∥ to symmetry plane	Azurite	135° 14′, 137° 10′, 132° 45′	99° 19′, 119° 13′, 90° 53′
	Borax	106° 35′	87°, 122° 33′, 96° 32′
	Cryolite	124° 58′, 124° 43′	91° 58′, 71° 32′
	Epidote	115° 23′, 128° 19′, 155° 11′	70° 4′, 70° 29′, 63° 5′
	Monazite	140° 48′, 87° 17′, 126° 29′	93° 26′
	Orthoclase	99° 42′, 129° 44′	118° 47′, 90° 7′
	Polybasite	90°	119° 58′
∠ between easiest cleavages bisected by plane of symmetry	Amphibole	130° 6′	124° 11′, 148° 28′
	Crocoite		93° 41′, 119° 10′
	Pyroxene	105° 50′, 148° 40′	87° 10′, 120° 49′, 131° 31′
	Spodumene	110° 20′	87°, 91° 26′
	Titanite	140° 43′, 159°	113° 31′, 136° 11′, 67° 57′
Cleavage	Datolite	90° 9′, 135°	115° 13′, 120° 56′, 115° 21′

micas and chlorites are usually pseudo-hexagonal.

Triclinic crystals. If the crystal shows no axes nor planes of symme is a triclinic crystal. There will be no right angles either between fa edges. The only corresponding faces will be opposite (parallel) faces. crystals of some of the most prominent triclinic minerals, however, approx in angles monoclinic crystals but are usually distinguishable by the occu of faces which have no symmetrically placed associates.

Angles. No angle will occur more than twice in any crystal. There are co tively few common triclinic species. The following table records a few of thei important angles:

Angles between the two easiest cleavages or the faces parallel to the cleavages		Other angles between com adjacent faces
The Plagioclases:		
Albite	93° 36′	127° 44′, 120° 46′
Anorthite	94° 10′	116° 3′, 98° 46′, 120° 31′
Labradorite	93° 56′	.....
Oligoclase	93° 28′	128° 3′, 98° 8′, 120° 54′
Amblygonite	104° 30′	120° 54′
Chalcanthite	123° 10′	110° 10′, 70° 22′, 103° 27
Cyanite	101° 30′	74° 16′, 131° 42′, 78° 58′
Rhodonite	87° 32′	107° 24′

Cleavage and its value as a test. In any crystal, whether with acteristic external form or not, the cohesion varies in different directions. U strain there is frequently a tendency to split or cleave perpendicular to directions of weakest cohesion *in definite* planes, which are always parall faces of simple crystals characteristic of the substance. In all crystals o same substance the same cleavages can be obtained. The number of direc of cleavage and the angles between the cleavage planes are, therefore, extre characteristic; moreover the cleavages serve to orientate the crystals in n cases. If the individual crystals are large enough, cleavage is obtained placing the edge of a knife or chisel upon the crystal and striking it a sh quick blow. If the individual crystals are very small the cleavage direct can usually be developed by crushing with pressure or a blow, and examining fragments with a hand glass.

In pyroxene, spodumene, corundum, magnetite, and some other species, some imens break easily in definite planes, while others do not. This is not true clea but a secondary phenomenon due to pressure, and is called "parting."

Cleavage and parting shapes may even be microscopically determined. To do sieve the crushed material through a 100-mesh screen upon a 120-mesh screen, particles remaining on the latter being collected on a clean paper.

A few fragments are placed in a drop of water or other liquid upon an object gla cover slip placed in position, and the preparation is examined.

Other tests dependent on crystalline structure. If the microscope is a able other tests may be applied to the crushed fragments, the best of which are: Ind of refraction (by the Becke or van der Kolk methods), absorption or pleochroism, ext tion angles, birefringence and elongation. A scheme for utilizing these tests was lished by A. J. Moses, in the *School of Mines Quarterly*, Vol. 34, July, 1913.

2. Important Physical Tests not Directly Dependent on Crystalli Structure

The most important of these tests or characters are Lustre, Color and Stre Hardness, and Specific Gravity.

Lustre. The lustre of a mineral is dependent upon its refractive power, its transparency, and its structure. It may be called the *kind* of brilliancy or shine of the mineral.

In determinative work minerals are broadly divided into Metallic and Non-metallic. Metallic lustre is the lustre of metals. It is exhibited only by opaque minerals, and these, *with the exception of the native metals, have a black or nearly black streak*. Opaque dark-colored minerals not distinctly non-metallic are said to be sub-metallic. Non-metallic lustre is exhibited by all transparent or translucent minerals. It may be vitreous or glassy, adamantine like unset diamond, resinous like sphalerite, pearly like mother of pearl, silky like fibrous serpentine, greasy like elaeolite, or waxy like chalcedony.

Hardness. The resistance of a *smooth* plane surface to abrasion is called hardness and is recorded in terms of the following scale:

1.0 Talc	6.0 Orthoclase	8.5 Chrysoberyl
2.0 Gypsum	7.0 Quartz	9.0 Sapphire
3.0 Calcite	7.5 Zircon	9.5 Carborundum
4.0 Fluorite	8.0 Topaz	10.0 Diamond
5.0 Apatite		

Approximations may be reached by use of finger nail (2½), copper coin (3) and knife (5½). Some smooth surface of the mineral to be tested is selected, and with a point of the standard is pressed and moved back and forth several times one-eighth of an inch or less. If the mineral is scratched it is softer than the standard. Two minerals of equal hardness will scratch each other. Pulverulent or splintery minerals are "broken down" by the test and yield an apparent hardness often much lower than the true hardness. Rough surfaces also yield doubtful results.

Color and streak. The color of minerals of metallic lustre and the color of the powder, or streak, when not white, are very much used in sight recognition. Minerals with non-metallic lustre often vary greatly in color. The color must safely obtained on a fresh surface. The streak is usually obtained by rubbing the mineral on a smooth but not glazed white or black surface, such as that of porcelain "streak plate" or a piece of touchstone (black quartz). The excess of powder should be brushed away and the thin adhering layer considered.

Specific gravity. The specific gravity of a substance is equal to its weight compared by the weight of an equal volume of distilled water at 4° C. Ordinary room temperature is used. Pure compact material is needed. The most accurate results are obtained by a delicate chemical balance, but for determinative purposes the following are more rapid and sufficiently accurate.

The **Jolly balance.** Two scale pans are attached, one below the other, to a spiral spring parallel to which is a mirror with a graduated scale. The lower scale pan is kept immersed in distilled water. The coincidence of a bead on the wire and its image in the mirror give:

A = Reading with nothing in either scale pan.
B = " " mineral in upper scale pan.
C = " " same fragment in lower scale pan.

$$\text{Sp. Gr.} = (B - A) \div (B - C)$$

The **Westphal balance.** More accurate results are obtained by substituting for the thermometer float of a Westphal balance a double scale pan, the lower pan of which is immersed in distilled water.

A = Weight needed to balance apparatus.
B = " " " " " with mineral in upper scale pan.
C = " " " " " " " lower scale pan.

$$\text{Sp. Gr.} = (A - B) \div (C - B)$$

Heavy liquids. If a fragment of a mineral is floating in a liquid of higher gravity and a diluent is stirred in, drop by drop, until the fragment if pushed do neither sink nor rise but stay where pushed, the specific gravity of the liquid a mined by a Westphal balance will be the specific gravity of the mineral. The liquids most used are: Methylene iodide (3.32); diluent, benzol. Bromoform diluent, xylol or benzol. Solution of mercuric iodide and potassic iodide (3.2); water.

4. Testing with the Blowpipe

Apparatus. The *essential* pieces of apparatus for all the tests given a

1. Either a gas blowpipe, or some form of burner for gas or heavy oil plain blowpipe.
2. Platinum wire about 0.25 mm diameter. Six inches of it will mal wires. A holder is needed.
3. Platinum-pointed forceps.
4. Charcoal in convenient sizes and with smooth surfaces (say 4 by 1 by
5. Tubes of hard glass about 3 by 5/16 in, closed at one end.
6. Pocket lens of good quality.
7. Simple goniometer.
8. Merwin Color Screen (G. M. Flint, Cambridge, Mass.).

For the other apparatus considerable latitude is possible and substitut be improvised for the regular stock article. The needed list would be: glasses, bottles (1 oz) for reagents, hammer, anvil, and magnet.

About ten reagents are used, the principal being borax, salt of phosp sodic carbonate, potassic bisulphate, cobaltic nitrate, and hydrochloric Two others are needed in preparing the bismuth flux and there will be n occasionally metallic tin and nitric or sulphuric acid.

A continuous blowpipe blast is obtained by distending the cheeks and the mouth as an air reservoir, breathing regularly through the nose and time to time admitting more air from the lungs through the throat mouth.

Any luminous flame may be used and, by regulating the relative amou air and flame, may be "blown" as a clear blue flame or a yellow flame, b which owe their color to incomplete combustion (CO or C) and therefore t *reduce*, that is, to take oxygen from substances placed therein. Hereafte flame is designated by the letters R.F. A practically non-luminous co envelope surrounds the blue flame and less distinctly the yellow flame. I there is an excess of oxygen and it therefore tends to *oxidize* substances therein. Hereafter this flame is designated by the letters O.F.

Fusion or fusibility. The ease of fusibility and the phenomena c fusion are convenient tests. The hottest portion of the flame is just b the tip of the blue flame. Some substances, noticeably certain iron ores, are infusible in the oxidizing flame are fusible in the reducing flame.

The test is most safely made by first heating on charcoal a fragment substance the size of a pin's head, to prove presence or absence of volat easily-reducible elements, which are likely to alloy with platinum. If the present the fusion test must be limited to the test on charcoal. If red metals or volatile constituents are absent, a small sharp-edged fragme heated in the platinum forceps, at the tip of the blue flame, directing the upon the point. Fragments long enough to project beyond the platinum sl be used and it is always well to examine the splinter with a magnifying before and after heating. *Fragments for comparison must be approximat same size and shape.*

The degree of fusibility is stated either in terms of a scale of fusibility, suggested by Kerl, or more simply as easily fusible, fusible, fusible with difficulty, or infusible:

Easily fusible:	1.	*Stibnite*, coarse splinters fuse in a candle flame.
	2.	*Chalcopyrite*, Small fragments fuse in the Bunsen burner flame.
Fusible:	3.	*Garnet* (*almandite*), coarse splinters easily fuse before the blowpipe. Not fusible in Bunsen burner.
	4.	*Actinolite*, fine splinters fuse easily before the blowpipe.
Fusible with difficulty:	5.	*Orthoclase*, fused only in fine splinters or on thin edges before the blowpipe.
	6.	*Calamine*, finest edge only rounded in hottest part of flame.
Infusible:	7.	*Quartz*, infusible, retaining the edge in all its sharpness.

The result of the fusion may be a glass or slag, which is clear and transparent, or white opaque, or of some color, or filled with bubbles; during the fusion there may be a slight intumescence, or a swelling and splitting (exfoliation). In certain instances the original form may change without fusion, or the substance may take fire and burn, or a gas may follow the loss of some volatile constituent.

Solubility. Acids, especially dilute (1 : 1) hydrochloric acid, are used not only to determine composition but also to determine the ease or degree of solubility. This test fails only from carelessness. The substance must be powdered as nearly pure as possible, finely ground and added to the acid in successive small quantities. A clear solution should be aimed at, acid being added only as it is needed until everything has dissolved. If complete solution cannot be obtained, the liquid must be filtered and the clear filtrate slowly and partially evaporated until separation commences. If doubt exists as to solubility the liquid *must be evaporated to dryness*, a residue proving solution to have taken place. Solubility may be accompanied by effervescence with or without odor of the acid, or only on heating. The evaporation may be difficult and incomplete, there may be separation of a perfect jelly, or of separate lumps of jelly, of powder, or of crystals. The solution may be of a characteristic color.

Testing for chemical components. The tests used are described in place in the determinative tables following Art 10. The manipulations and precautions are briefly as follows:

1. **Testing in closed tubes.** A narrow tube of hard glass, about 3 in by 5/16 in and closed at one end, is best. Enough of the substance is slid down a narrow strip of paper, which is inserted in the tube, to fill it to the height of about ¼ in; the paper is withdrawn and the inclined tube heated gradually at the lower end to a red heat. Soda or other reagents are sometimes mixed with the substance. The results may be: evolution of odorous or non-odorous vapors, sublimates of various colors, decrepitation, incandescence, fusion, charring, change of color, and magnetization.

2. **Testing on charcoal.** A shallow cavity, to prevent the substance from slipping, near one end of the charcoal, and a small fragment of the mineral is placed in it. The charcoal is held in the left hand, the surface tipped at about 120° to the direction in which the flame is blown, and a gentle O.F. is blown on the substance. If no sublimate forms the heat is increased, still keeping the flame oxidizing. Another fragment is treated in the R.F., the substance being kept covered for several minutes with the yellow flame.

The sublimates, their color, position on the charcoal, ease of removal by heating in the O.F. or R.F., and the colors imparted to the flame are all noted. Chemical changes may be indicated by reduced metal, magnetic residues, alkalinity, etc.

3. **Testing with soda on charcoal.** Sodic carbonate ("Soda"), heated on charcoal acts as a flux; it also exerts a reducing action, attributed to the formation of sodic oxide, nascent sodium, and carbon monoxide. It combines with many substances, forming both fusible and infusible compounds. The most satisfactory general method

is to mix one part of the substance to be tested with three parts of moistened soda little borax, and treat with a good R.F. on charcoal until everything that can be ab has disappeared.

IV. Testing with bismuth flux on charcoal and on plaster tablets. limates of brilliantly colored iodides and sulpho-iodides are obtained if bismut (two parts sulphur, one part potassium iodide, and one part acid potassium sulph mixed with certain powdered minerals, placed on charcoal, or a plaster tablet, and l gently. The larger series of tests are obtained on plaster, the sublimates differ position and to some extent in color from those obtained on charcoal. Plaster t are prepared by spreading a thick paste of plaster of Paris and water upon a sh oiled glass, and smoothing to a uniform thickness (¼ in to ½ in). While still so paste is cut with a knife into uniform slabs, 4 in by 1½ in. It is then dried, after the tablets are easily detached from the glass.

V. Flame coloration. A number of minerals when heated color the flame, so a gentle heat, some only at the highest heat attainable. Repeated dipping of the m in hydrochloric acid usually assists by forming volatile chlorides. A good meth cover all cases is as follows: Arrange a black background, such as a piece of cha powder the substance finely, flatten the end of a clean platinum wire and dip it in acid, then in the powder, and hold it first just touching the flame near the blowpip then at the tip of the blue flame; again dip in the acid and again heat as before.

Concentrated sulphuric acid, and also a paste made of water, 4½ parts acid potas sulphate and 1 part of calcium fluoride, are also used to release certain flame-col constituents, especially boron, phosphorus and lithium.

Red flames of calcium, strontium, lithium, and the violet flames of potassium i presence of sodium, are most conveniently studied by Merwin's Color Scale (*Sc* Vol. 30, p. 571), consisting of three colored strips of celluloid; No 1, blue, No 2, lapping blue and violet, No 3, violet. These absorb different portions of the spec as follows:

	No 1	No 2	No 3
Sodium	Absorbed	Absorbed	Absorbed
Potassium	Blue-violet	{ Violet and { Violet-red	{ Violet and { Violet-red
Calcium	Greenish yellow	Absorbed	Faint crim
Strontium or lithium	Absorbed	Absorbed	Crimson

These elements are still more exactly distinguished by use of a small pocket spe scope. The mineral is moistened with hydrochloric acid and brought on a plati wire into the non-luminous flame of the Bunsen burner. This is viewed through spectroscope and bright lines are seen. The yellow sodium line is almost invari present and the position of the other lines is best fixed by their situation relative to bright yellow line.

VI. Bead tests with borax and with salt of phosphorus. The oxides of cer elements dissolve in borax and salt of phosphorus and impart characteristic colors to mass, which may differ when hot and cold and according to the degree of oxidatio reduction. Preliminary to bead tests, sulphides, arsenides, arsenates, etc., may be verted into oxides by treating in a shallow cavity on charcoal at a dull red heat; first a feeble oxidizing flame, then a feeble reducing flame, then again an oxidizing flame, so on as long as odors or fumes are noticeable.

To make a bead: Make a loop in platinum wire by bending it around a pencil p so that the end meets but does not cross the straight part. Heat the loop, dip it into flux, and fuse to a clear bead the portion that adheres. Add more flux until the bea of full rounded shape. With salt of phosphorus the bead should be held a little ab the flame so that the ascending hot gases will help to retain the flux upon the w Touch the warm bead to the substance, place it in the O.F., and treat until clear. N the colors, hot and cold. Then treat in the R.F. and note colors as before.

Fusing. Some substances heated with a strong flame will give clear glasses until ...ed; but, if heated slowly and gently or intermittently, will yield opaque or enamel... beads before saturation.

Testing with cobalt solution. Certain substances become colored, when ...ated with a solution of cobalt nitrate in ten parts of water and then heated to a ... test. The test is usually made on charcoal. Certain other substances yield colors ... only heated, cooled, and then moistened with the cobalt solution without reheating. ... minerals boiled with cobalt solution are colored thereby.

OCCURRENCE AND ASSOCIATION OF MINERALS

5. Minerals of Rocks and Veins

Associates. Most minerals occur in more than one class of rock or mineral ...t, and therefore with different groups of associates. The most probable ...ates of any mineral in any particular occurrence are: 1. The great minerals ... form of occurrence. 2. Minerals containing some prominent element ...ments of the given mineral. In the following lists, which include the rock-...ing minerals and the minerals of economic importance, the species in italics ... relatively rare.

Minerals of the igneous rocks. These minerals in general have either sep...ted from a fusion solution or "magma" (each separating whenever for the ...ing temperature and pressure the magma is supersaturated with it), or ... have formed later, as secondary minerals, by the decomposition or alter... of the primary minerals. It is estimated that they constitute 95% of ... known crust of the earth.

Principal primary minerals of igneous rocks. Albite, anorthite, amphi...e (hornblende), biotite, chrysolite (olivine), enstatite, hypersthene, labra...ite, leucite, muscovite, nephelite (elaeolite) oligoclase, orthoclase, pyroxene ...ite), quartz, sodalite.

Minor primary minerals of igneous rocks. *Analcite*, apatite, *chalco...ite, chrysoberyl*, chromite, *cinnabar*, corundum, *epidote, garnet* (almandite, ...andite, pyrope), *goethite, gold, graphite*, hematite, ilmenite, *lepidolite*, mag...ite, *millerite, molybdenite*, monazite, pyrite, pyroxene (diopside), pyrrhotite, ...ile.

Secondary minerals in igneous rocks. *Albite*, alunite, analcite, apophyllite, ...ite, *azurite*, *barite*, calcite, chabazite, chalcedony, *chalcanthite, chalcopyrite*, chlo...ite *chrysocolla*, copper, datolite, epidote, kaolin, lepidolite, limonite, magnetite, *mala...ite*, muscovite, natrolite, opal, *pyrargyrite*, quartz, serpentine, siderite, *sphalerite*, ...ite, talc, *tetrahedrite, turquois*, wernerite.

Minerals of pegmatite veins. Vein-like portions of granites or other igneous rocks ... which the minerals of the rock are found in much larger crystals and in which many other minerals occur not noticed in the adjoining rocks.

Albite, *amblygonite*, apatite, beryl, biotite, *cassiterite, chabasite, chlorite, chrysoberyl*, ...ite, cryolite, *diamond, galenite, garnet* (almandite and spessartite), graphite, lepi...ite, magnetite, microcline, *molybdenite, monazite*, muscovite, nephelite, orthoclase, ...ite, *pyrrhotite*, quartz, spodumene, topaz, tourmaline, uraninite, zircon.

Minerals of ore veins. For convenience these have been listed under two headings: Minerals in zone of weathering or oxidation, and minerals of unoxidized zone. **In zone of oxidation.** Anglesite, azurite, brochantite, cala...ite, celestite, cerargyrite, cerussite, *chalcanthite*, chrysocolla, copper, crocoite, ...rite, embolite, erythrite, goethite, gold, iodyrite, limonite, malachite, manga-

nite, mimetite, pyromorphite, rhodochrosite, siderite, silver, **smithsonite**, *tianite*, *sulphur*, vanadinite, *vivianite*, wulfenite. **In unoxidized zone.** mony, argentite, *arsenic*, arsenopyrite, barite, bornite, braunite, **calcite**, c erite, chalcocite, chalcopyrite, cobaltite, copper, dolomite, fluorite, **gal** gold, graphite, jamesonite, linnæite, marcasite, millerite, niccolite, **orpi** orthoclase, pentlandite, proustite, pyrargyrite, pyrite, pyrrhotite, **quartz**, gar, smaltite, sphalerite, stannite, stephanite, stibnite, sylvanite, **tetrahe** uraninite.

Minerals of tin veins. Albite, amblygonite, apatite, arsenopyrite, muth, calcite, cassiterite, chlorite, columbite, fluorite, *galenite*, kaolin, **lepid** *molybdenite*, pyrite, pyroxene, quartz, scheelite, wernerite, wolframite.

Minerals of apatite veins. Albite, amphibole, apatite, biotite, ca *enstatite*, hematite, ilmenite, magnetite, oligoclase, pyrite, quartz, rutile, tita tourmaline, wernerite.

Minerals due to volcanic exhalations. Alunite, sassolite, sulphur, and *relat* small quantities of other species, as amphibole, hematite, sal-ammoniac, etc., occur a result of gases given off during volcanic action.

6. Minerals Found in Saline Residues

These exist as sediments precipitated from solution in natural waters, spr rivers, marshes, lakes, seas, and oceans.

From springs. Alunogen, aragonite, barite, bauxite (?), calcite, celestite, chalced cinnabar, fluorite, hydrozincite, kalinite, limonite, pyrite, sassolite, siderite, sulphur

From soda and borax lakes and lagoons. Anhydrite, calcite, borax, cele *cerargyrite*, colemanite, dolomite, *embolite*, *gold*, gypsum, halite, mirabilite, sass *soda nitre*, *sulphur*, *trona*, ulexite.

From oceans, seas, lakes, and marshes. Apatite, anhydrite, bauxite, bora *calcite*, carnallite, *celestite*, *cerargyrite*, *dolomite*, *epsomite*, gypsum, halite, kainite, kies limonite, siderite, wad.

Local saline residues (often incrustations or efflorescences). Alunite, aluno chalcanthite, copiapite, epsomite, kalinite, mirabilite.

7. Minerals in Gravels, Sands, Clays, and Marls

Minerals common to all. Biotite, calcite, chlorite, *garnet*, hematite, kaoli limonite, *magnetite*, muscovite, orthoclase, plagioclase, pyrite, *pyrophyllite*, *pyrox* rutile, siderite, titanite, *tourmaline*.

Gem minerals and ores in gravels and sands. Cassiterite, chrysoberyl, chryso corundum, diamond, gold, ilmenite, monazite, platinum, spinel, tourmaline, **topaz**, zir

Minor minerals in gravels and sands. *Amphibole*, *andalusite*, apatite, cyan *dolomite*, enstatite, epidote, *hypersthene*, *microcline*, sepiolite, serpentine, *sillimanite*.

Ores in clays. Galenite, limonite, manganite, psilomelane, pyrolusite, wad.

Minor minerals in clays and marls. Amphibole, aragonite, barite, *celestite*, g sum, halloysite, *orpiment*, realgar, strontianite, *vivianite*.

Minerals in sandstones. Chiefly quartz, orthoclase, plagioclase, limonite, mus vite. *Minor Minerals* are *carnotite*, *galenite*, *gold*, *marcasite*, *manganite*, *pyrite*, *py lusite*, *siderite*, *sphalerite*.

Minerals in sedimentary limestone. Aragonite, calcite, dolomite, fluorite, galen limonite (bog ore), *nitre*, opal (diatomaceous earth), *siderite*, *soda nitre*, sulphur, sphaleri

In serpentine and soapstones. *Amphibole*, *aragonite*, *arsenopyrite*, calcite, ch rite, chromite, chrysolite, cinnabar, diamond, dolomite, enstatite, epidote, garn (pyrope), garnierite, ilmenite, magnesite, magnetite, phlogopite, *platinum*, *pyroxe* pyrophyllite, quartz, *sepiolite*, serpentine, talc.

8. Contact Minerals

...e an igneous rock penetrates a preëxisting rock the heat, pressure, and ...d vapors frequently produce new minerals at and near the surface of ...

...ntacts with limestone. Amphibole (tremolite), anorthite, biotite, bor... ...rodite, corundum, *danburite*, *enstatite*, epidote, *fluorite*, garnet ...ar and andradite), graphite, *lazurite*, molybdenite, phlogopite, pyrite, ... (diopside), *scheelite*, spinel, tourmaline, *vesuvianite*, wernerite, wollas...

...ntacts with silicate rocks (clay, shale, slate, or crystalline schists). ... (hornblende), andalusite (chiastolite), biotite, chlorite, corundum, ... epidote, garnet, ilmenite, magnetite, pyroxene (augite), quartz, rutile, ...te, spinel, staurolite, titanite, tourmaline, topaz, wernerite, zircon.

Minerals of Metamorphic Rocks

...e minerals of the metamorphic rocks include many species of the ... rocks, and many species already listed under contact minerals. A ... list follows: In *Crystalline Limestones, and Dolomites:* amphibole ..., apatite, aragonite, calcite, *chondrodite*, corundum, dolomite, frank... *molybdenite*, phlogopite, pyroxene, pyrrhotite, rhodonite, serpentine, ...nite, spinel, *talc*, willemite, zincite, zircon. In *Gneisses and Schists:* ...tact minerals of the second list (contacts with silicate rocks). Also ...te, apatite, beryl, biotite, calcite, chalcopyrite, chrysoberyl, *datolite*, ..., *fibrolite*, graphite, hematite, molybdenite, monazite, muscovite, ortho... ...oclase, pyrite, pyrophyllite, pyrrhotite, talc, *tetrahedrite*, vesuvianite, ...

THE USES OF MINERALS

...is list includes only the principal uses of the minerals as such, and their ... as the material from which other substances are directly extracted or ...ufactured. The secondary products derived from these primary products ... not mentioned.

9. Uses of Minerals in Their Natural State

Abrasives. Quartz, garnet, opal (tripolite and diatomaceous earth), corundum and ... diamond (bort), orthoclase. Leucite and alunite rocks have been used as mill...

Building stones. Quartz, orthoclase, plagioclase, muscovite, biotite, pyroxene and ... in varying proportions, forming igneous rocks commercially known as granite ...; talc and pyrophyllite (soapstones), serpentines, calcite and dolomite (lime... and marbles), quartz (sandstone).

Electrical insulators. Muscovite, phlogopite, calcite (marble).

Fertilizers. Carnallite and kainite for potash; soda nitre for nitrogen; gypsum and ... lime; apatite (phosphate rock) for phosphoric acid. Muscovite and biotite as ...ers of moisture.

Fluxes. Calcite, fluorite, borax, pyrolusite.

Glass. Chiefly quartz (sand and sandstone) and calcite (limestone); to a less extent ...clase, plagioclase, cryolite, and pyrolusite.

Lubricants. Graphite, talc, muscovite.

Paints and pigments. Hematite and limonite as "metallic paint;" the same min... associated with clay, "ocher." Calcite (chalk) as "whiting;" wad, barite, gypsum, ...a, muscovite, talc, kaolin, quartz, magnesite, azurite, graphite, asphaltum.

Paper manufacture. Talc (fibrous), gypsum (selenite), as constituents of Barite, calcite, kaolin, magnesite, bauxite, muscovite, for weight and glaze.

Porcelain, pottery, etc. Kaolin and other clays, quartz, orthoclase, albite gypsum.

Precious stones. Diamond, beryl, emerald, corundum (sapphire and ruby), beryl (alexandrite), garnet (demantoid), spinel (ruby spinel). **Semi-precious** Other varieties of beryl, corundum, chrysoberyl, spinel, and garnet. **Also opal** olite (peridot), quartz (amethyst and yellow), topaz, tourmaline, turquoise, spodumene (kunzite, hiddenite), orthoclase (moonstone). **Ornamental stones.** chalcedony (onyx, carnelian, sard, agate, etc.), quartz (rose, cat's eye, ave smoky, etc.), orthoclase (amazon stone), plagioclase (labradorite and sunstone phibole (jade), lazurite (lapis lazuli), malachite, azurite, calamine, smithsonite, chr fluorite, gypsum (satin spar), serpentine, hematite, pyrite, rhodonite, talc. Oc faceted stones are cut from apatite, andalusite, cassiterite, chondrodite, cyanite, p (diopside), enstatite, epidote, prehnite, staurolite, titanite and vesuvianite.

Refractory materials and heat insulators. Asbestos, bauxite, chromite, d graphite, ilmenite, kaolin, magnesite, muscovite, opal (diatomaceous earth), ser (chrysotile), quartz, pyrophyllite and talc (soapstone).

Rubber manufacture. Sulphur, stibnite, barite, calcite.

Soap and washing powders, toilet articles. Borax, opal (diatomaceous talc, quartz, magnesite, orthoclase.

Sundries. *Coloring or decolorising:* pyrolusite, psilomelane, rutile. *Cond* halite. *Explosives:* nitre, sulphur. *Filters:* opal (tripolite). *Enamels:* fluorite, *Matches:* stibnite, sulphur. *Optical:* quartz, calcite, fluorite, gypsum, mus *Pencils:* graphite, talc, pyrophyllite. *Pipes:* sepiolite (meerschaum), succinite (a

10. Products Extracted or Manufactured Directly from Mine

Aluminum from bauxite, possibly gibbsite, with cryolite as flux.
Alundum (Al_2O_3) from bauxite.
Aluminium sulphate and alum from alunite, cryolite, bauxite, kaolin.
Antimony from stibnite and its alteration products and lead ores carrying *antim*
Arsenic from arsenopyrite and sometimes from smaltite, cobaltite, enargite, etc.
Barium hydroxide and barium sulphide from barite.
Bismuth from native bismuth, bismutite, and bismite.
Borax and boric acid, from colemanite, ulexite, borax, and sassolite.
Bromine from halite (salt brine).
Cadmium from sphalerite and smithsonite containing greenockite.
Calcium oxide (lime) from calcite (limestone).
Calcium sulphate (hemi-hydrate) or plaster from gypsum.
Calcium superphosphate from apatite.
Cements from calcite and clays.
Carbonic acid from magnesite and calcite.
Chlorine from hydrochloric acid and pyrolusite, the former being derived from ha
Chromium alloys, especially ferrochrome from chromite.
Cobalt oxide and cobalt arsenate (saffre) from smaltite, cobaltite, and cobaltif limonite.
Copper principally from chalcopyrite and bornite, native copper, cuprite, mala and azurite, though chalcocite, enargite, tetrahedrite, atacamite, brochantite, cha thite, and chrysocolla are all sources of copper in certain districts. In addition to the iron sulphides often carry copper which is extracted after burning for sulphuric
Copper sulphate from chalcopyrite.
Gold from gold and the gold tellurides (sylvanite, calaverite, petzite), from silver copper ores and from the minerals pyrite, arsenopyrite and pyrrhotite, and spha and other sulphides or tellurides.
Hydrochloric acid from halite.
Hydrofluoric acid from fluorite and cryolite.
Iodine from sodium iodate obtained from soda nitre.
Iridium from iridosmine.

... hematite, limonite, magnetite, and siderite, goethite, and turgite (commer-
... with limonite), some ilmenite, and rarely residues from the roasting of
...

... (ferrous) or "copperas" from pyrite and chalcopyrite.
... *alloy* from franklinite and certain manganiferous hematites and sider-
... pyrolusite, psilomelane, manganite and other manganese oxides.
... from galenite and cerussite. Anglesite and pyromorphite sometimes occur
...

... (sublimed white lead and blue lead) from galenite.
... *carbonate* from spodumene, lepidolite, amblygonite.
... from carnallite.
... *carbonate* from dolomite. Basic carbonate from kieserite.
... *oxide* from magnesite, and indirectly kieserite.
... *chloride* from carnallite.
... *sulphate* (epsom salts) from kieserite and less extensively from magnesite
...

... *alloys* from pyrolusite, psilomelane and braunite, or with intermixed rhodo-
... and rhodonite.
... salts from pyrolusite.
... from cinnabar.
... *and ammonic molybdate* from molybdenite.
... from pentlandite, garnierite, nickeliferous pyrrhotite, and to a less extent
... niccolite and the cobalt minerals, cobaltite and linnæite.
... acid from soda-nitre and nitre.
... from copper ores and platinum.
... from an impure calcium phosphate (sombrerite), or from bone ash.
... from native platinum and sperrylite, and from some gold and copper ores.
... from carnallite.
... *dichromate* from chromite.
... *sulphate* from kainite.
... *nitrate* from soda nitre and carnallite.
... *chloride* from uraninite, carnotite, and autunite.
... from platinum.
... from sulphur, chalcopyrite, and pyrite.
... *carbide* (carborundum) from quartz and coke.
... *alloys* (ferro-silicon) from quartz.
... from native silver, argentite, cerargyrite, embolite, proustite, pyrargyrite, and less
... , hessite, polybasite, and iodyrite. Included in other minerals, notably, galen-
... cerussite, but also in copper ores, manganese ores and with gold in pyrite and arsen-
...

... *borate* (borax) from colemanite, ulexite, sassolite, and native borax.
... *stannate* from cassiterite.
... *sulphate* (salt-cake) from halite, and from this, caustic soda, carbonate, bicar-
... etc.
... *nitrate and chloride* from strontianite.
... *acid, sulphurous acid*, from native sulphur, pyrite, marcasite, chalcopyrite,
... pyrrhotite, and other sulphide ores.
... from columbite.
... *nitrate and thorium oxide* from monazite, thorite, thorianite.
... *and sodium stannate* from cassiterite.
... *and ferro-titanium* from ilmenite.
... *carbide* from rutile.
... *ferro-tungsten*, from wolframite and scheelite.
... *of soda* from wolframite.
... *yellow or sodium diuranate* from uraninite, carnotite.
... *and ferro-vanadium* from carnotite, patronite, roscoelite, vanadinite, descloi-
...

... *oxide* from mottramite.
... *"zinc dust," and zinc oxide* from sphalerite, smithsonite, and calamine; and, in
... Jersey, willemite and zincite.
... *sulphate* from sphalerite.
... *oxide* from zircon.

DESCRIPTIVE AND DETERMINATIV TABLES

Rare species without economic value are omitted from these tal found and tested, the result would be a failure to identify. Their i would enormously increase the complexity of the tables and somewhat the difficulty of any determination.

In view of the limited space the species are described only in the tab the accompanying index will enable the user to find a brief description species.

(For example, Scheelite. A reference to VII, 4, will give compositio talline system, hardness, specific gravity, colors, solubility, flame colc behavior with fluxes and general appearance.)

The uses and occurrence of minerals are summarized in separate tabl using the tables the customary precautions are understood to be taken:

1. Tests must be made upon homogeneous materials, and lustres and observed on fresh fractures.

2. Classifying tests must be decided; not weak, nor indefinite. If cided, the species on both sides of the dividing line must be considered.

3. Hardness tests should be assumed to be within say one half; tha determination H = 5 should for safety be taken as 4.5 to 5.5.

As shown by the accompanying key, the principal subdivision is be metallic and non-metallic lustre. The blowpipe test is made subordina minerals of metallic lustre and minerals of non-metallic lustre with c streaks; but, for minerals of non-metallic lustre with white streaks, expe has proved that either the blowpipe or the microscope lead to a determi with less repetition than such qualities as color and hardness.

A novel feature of the tables is the "scheme within a scheme," by whi order of testing may be varied. For instance, in Table VI the arrangem by blowpipe tests in order of hardness, but the parallel columns permit to be used as the classifying test; that is, the order of testing may be colo hardness or blowpipe test and hardness.

Similarly in Table III, the arrangement of the metallic white and gray erals is by streak and hardness, but the parallel columns permit the behavi charcoal in oxidizing and reducing flame to be used as the classifying test; th the order of testing may be color, streak and hardness, or color and beh on charcoal.

Chemical symbols are used only for the formulas of the species and fo common solvents, HCl, H_2SO_4, HNO_3, KOH, etc. Aside from these a abbreviations are used, the principal being:

Systems of crystallization are indicated by the letters: I (Isometric), T (ragonal), O (Orthorhombic), M (Monoclinic), Tri (Triclinic), H (Hexagon

Terms in blowpipe tests. Soda for sodic carbonate, S. Ph. for sa phosphorus, O. F. and R. F. for oxidizing and reducing flame, Co. Sol. for co solution, coal for charcoal.

The + sign in any column opposite any mineral indicates that the qu indicated is a character of that mineral.

TABLE I. *Minerals of Metallic or Sub-Metallic Lustre, Black or Nearly Blac Color* (including arbitrarily some dark-colored minerals of doubtful lust

1. With black or nearly black streak.
2. With streak not black.

TABLE II. *Minerals of Metallic Lustre, Tin White, Silver White, Lead Gray or Steel Grey in Color.*

With black streak.
With streak not black.

TABLE III. *Minerals of Metallic Lustre, Metallic Yellow, Bronze or Red in Color.*

With black streak.
With streak not black.

TABLE IV. *Minerals of Non-Metallic Lustre, with Decided Taste* (soluble in water).

When fused, powdered, moistened with concentrated H_2SO_4 and ignited on platinum wire, yield:

Flame yellow. No violet through blue glass.
Flame violet through blue glass.
Flame green.
Flame not colored.

TABLE V. *Minerals of Non-Metallic Lustre, Tasteless, with Colored Streak.*

Blue streak.
Green streak.
Yellow to yellowish brown streak.
Red to reddish-brown streak.
Orange streak.

TABLE VI. *Minerals of Non-Metallic Lustre, Tasteless, with White Streak, Yielding Reactions on Charcoal with Sodic Carbonate:*

Volatile fumes or sublimates.
No fumes or sublimates, but are reduced to metallic or magnetic particles.
No fumes, sublimates or reduced particles, but if moistened will stain silver.

TABLE VII. *Minerals of Non-Metallic Lustre, Tasteless, with White Streak, Yielding no Tests with Sodic Carbonate.*

A thin fragment heated in platinum forceps at tip of the blue flame is:

Fused easily (1 to 4) to a white, opaque glass, or enamel.
Fused easily (1 to 4) to a colorless glass.
Fused easily (1 to 4) to a colored glass.
Fused with difficulty (5 to 6), sometimes barely rounding the edges.
Infusible, but in powder is made deep blue by ignition with cobalt solution.
Infusible and is *not* made deep blue by ignition with cobalt solution.

Table I. Minerals of Metallic or Sub-M
(Including arbitrarily some

	Crystal system: name, composition, hardness and specific gravity	On coal in O. F. and R. F.					
		Odor:		Sublimates	Residue:		
		As	SO_2		None	Metallic	Magnetic
I. WITH BLACK OR NEARLY-BLACK STREAK	H. Graphite ..... C H=1 to 2 G=2.1 to 2.2	.....	.....	.....	....	....	....
	Pyrolusite ..... MnO_2 H=1 to 2.5 G=4.7 to 4.8	.....	.....	.....	....	....	....
	O. Stibnite ..... Sb_2S_3 H=2 G=4.5 to 4.6	.....	+	Dense white	+	....	....
	I. Argentite ..... Ag_2S H=2 to 2.5 G=7.2 to 7.6	.....	+	.....	....	+	....
	O. Stephanite ..... Ag_5SbS_4 H=2 to 2.5 G=6.2 to 6.3	.....	+	White, later reddens	....	+	....
	I. Galenite ..... PbS H=2.5 G=7.4 to 7.6	.....	+	White O. F. Yellow R. F.	+	+	....
	O. Jamesonite ..... $Pb_2Sb_2S_5$ H=2.5 G=5.5 to 6	.....	+	Dense white. Some yellow	+	+	....
	O. Polybasite ..... $(Ag\cdot Cu)_9SbS_4$ H=2 to 3 G=6 to 6.2	.....	+	Dense white	....	+	....
	O. Chalcocite ..... Cu_2S H=2.5 to 3 G=5.5 to 5.8	.....	+	.....	....	+	....

...re, Black or Nearly Black in Color
...ed minerals of doubtful lustre)

...ity ...ral	Solubility	Heated in closed tube	Other tests	Appearance
... ...ly ...ded	Insoluble	.....	In $CuSO_4$ solution in contact with zinc *quickly* copper-plated. (Molybdenite is slowly plated)	Shining flakes and masses or dull, impure masses. Soft, greasy and cold to the touch. Shining mark on paper.
... ...wa	Sol. HCl (evolution of Cl)	Yields oxygen and often water	Colors borax amethystine in O. F.	Bright, easily bruised needles or fibres or dull masses. Dull mark on paper.
1	Sol. (with odor H_2S)	Fuses. Sublimate black hot, red cold.	Made yellow by KOH and partially dissolved. HCl gives orange ppt.	Bright columnar, bladed or fine-grained masses, less frequently in prismatic crystals or interlaced bunches of needle crystals.
1 5	Sol. HNO_3. (Residue S. Ppt. with HCl)	Fuses. No sublimate	With soda, metallic Ag and S reaction	Coatings and disseminated plates. Rarely crystals. Cuts like metallic lead. Streak is shining.
1	Sol. HNO_3 (white residue)	Fuses. Slight red sublimate	Decomposed by KOH. HCl gives orange ppt.	Fine-grained, often disseminated. Sometimes crystals. Soft but brittle.
2	Sol. excess. hot HCl	Decrepitates. A little white sublimate	"Bismuth Flux" on coal gives greenish yellow sublimate	Granular and cleavable masses and cubic crystals which cleave into cubes.
1	Sol. (with odor H_2S)	Brownish red sublimate	"Bi. Flux" on coal gives greenish yellow sublimate	Needle crystals, or hair-like or felted; also compact and fibrous massive.
1	Sol. HNO_3. (Ppt. with HCl)	Fuses. No sublimate	Metallic residue ignited with HCl gives azure blue flame	Best known in six-sided plates. In thin splinters is cherry red by transmitted light.
...: 2.5	Sol. HNO_3 (residue S)	Fuses. No sublimate	Emerald-green flame made azure blue by HCl	Compact masses, nodules and disseminated. Often coated with the green carbonate. Rarely crystals.

Table I (Continued). — Minerals of M

Crystal system: name, composition, hardness and specific gravity	On coal in O. F. and R. F.			
	Odor:		Sublimates	Residu
	As	SO_2		None / Metal-lic
H. Pyrargyrite Ag_3SbS_3 H=2.5 G=5.7 to 5.8		+	Dense white	.... / +
I. Sphalerite ZnS H=3.5 to 4 G=3.9 to 4.1		+	Yellow hot, White cold	+ / ....
O. Manganite MnO OH H=4 G=3.7 to 4.7				
Limonite $Fe_2(OH)_6 \cdot Fe_2O_3$ H=5 to 5.5 G=3.6 to 4				
J. Goethite FeO OH H=5 to 5.5 G=4 to 4.4				
T. Hausmannite Mn_3O_4 H=5 to 5.5 G=4.7 to 4.8				
W. Wolframite $(Fe,Mn)WO_4$ H=5 to 5.5 G=7 to 7.5				
Ilmenite $FeTiO_3$ H=5 to 6 G=4.5 to 5				

A. WITH STREAK NOT BLACK

Sub-Metallic Lustre, Black or Nearly Black in Color

	Solubility	Heated in closed tube	Other tests	Appearance
	Sol. HNO_3. White residue	Subl. black hot, red cold	Decomposed by KOH. HCl produces orange ppt.	Veins or crusts with a brilliant adamantine lustre showing red tint in thin layers. Rare crystals. Streak purplish-red.
	Effervesces. Gives odor H_2S	No sublimate	Sublimate on coal made bright green by ignition with cobalt solution	Black and gray crystals and cleavable to fine-grained masses. Streak pale brown.
	Sol. HCl. (Evolves Cl)	Much water. A little oxygen	Borax O. F. amethystine	Crystals often grouped in bundles, rarely massive, granular or stalactitic. Streak dark brown.
	Soluble	Reddens. Yields much water	Often reacts for manganese and may give jelly residue	Cellular and pulverulent or as compact masses often radiated or stalactitic and with varnish-like surfaces. Never crystallised. Streak yellowish-brown.
	Soluble	Reddens. Yields water	Often reacts for manganese. (Soda bead O. F. bluish green)	Occurs massive but is best known as crystals, often flattened like scales, or needle-like, or in parallel position. These shade into feather-like and velvety crusts.
	Sol. HCl. (Evolves Cl)	.....	Borax O. F. amethystine	Granular masses occasionally in twinned pyramids. Streak chestnut brown.
	Partial	.....	Solutions become deep blue on addition of tin. Solution of S. Ph. bead in HCl best	Heavy monoclinic crystals and cleavable, bladed and granular masses. Streak brownish black.
	Partial	.....	Filtered solution boiled with tin becomes violet	Usually compact masses, often thin plates or imbedded grains or as sand. Rarely in tabular hexagonal crystals. Streak brownish red.

Table I (Continued). — Minerals of Metall

Crystal system: name, composition, hardness and specific gravity	On coal in O. F. and R. F.				
	Odor:		Sublimates	Residue:	
	As	SO_2		None	Metallic
2. WITH STREAK NOT BLACK					
I. Uraninite..... $UO_2 \cdot UO_3$, etc. H=5.5 G=9 to 9.7					
I. Chromite..... $FeCr_2O_4$ H=5.5 G=					
H. Hematite..... Fe_2O_3 H=5.5 to 6.5 G=4.9 to 5.3					
O. Columbite..... $Fe(CbTa)_2O_6$ H=6 G=5.4 to 6.5					
I. Franklinite..... $(FeMnZn)O(FeMn)_2O_3$ H=6 to 6.5 G=5 to 5.2			Slight white		
T. Rutile..... TiO_2 H=6 to 6.5 G=4.1 to 4.2					
T. Cassiterite..... SnO_2 H=6 to 7 G=6.8 to 7.1			Some varieties give white sublimate		

Minerals of Metallic or Sub-Metallic Lustre, Black

allic Lustre, Black or Nearly Black in Color

ity al	Solubility	Heated in closed tube	Other tests	Appearance
e (or hffi-	Sol. HNO_3 (to yellow liquid)	.....	Solution gives yellow ppt. with ammonia. Borax R. F. green, near saturation black- ens	Botryoidal or granul with pitch-like lust Rarely in small isom ric crystals. Stre dark green.
e dly	Insoluble	.....	Borax or S. Ph., O. F. or R. F., em- erald green cold	Granular or compa or rarely in small c tahedral crystal Pitch-like lustr Often with serpentir Streak dark brown.
le	Soluble	.....	.....	Coarse to fine micaceo masses and tabular coarser crystals wi brilliant lustre. Oc sionally kidne shaped. Streak brow ish red.
ble	Insoluble	.....	Fused with KOH and boiled with HCl and tin gives blue solution.	Masses and brillian often iridescent, pr matic crystals. Stre dark red.
ble	Slowly in HCl evolving Cl	.....	Soda bead O. F. blu- ish green	Compact masses, roun ed grains and octal dral crystals. Slight magnetic. Red zi ite and yellow to gre willemite are asso ates. Streak da brown.
ble	Insoluble	.....	S. Ph. bead in R. F. violet	Masses and crystals wi considerable lustr Streak pale yellow.
ble	Insoluble	.....	With soda or sul- phur on coal in strong heat a subl. yellow hot, white cold, made bluish green by ignition with cobalt solu- tion	Brilliant crystals, u ally with brown tin Streak pale yellow.

Table II. Minerals of Metallic Lustre, T

	Crystal system: name, composition, hardness and specific gravity	On coal in O. F. and R. F.					
		Odor:		Sublimates	Residue:		
		As	SO_2		None	Metallic	Magnetic
I. WITH BLACK STREAK	O. Stibnite Sb_2S_3 H=2 G=4.5	.....	+	Dense white. Volatile	+	....	....
	I. Galenite PbS H=2.5 G=7.4 to 7.6	.....	+	White O. F. Yellow R.F.	+	+	....
	O. Jamesonite $PbSb_2S_3$ H=2.5 G=5.5 to 6	.....	+	Dense white. Some yellow, volatile	+	+	....
	I. Tetrahedrite $Cu_8Sb_2S_7$ H=3 to 4.5 G=4.5 to 5.1	Sometimes	+	Dense white	....	+	....
	I. Stannite (Cu · Sn · Fe) S H=4 G=4.5	.....	+	Non-volatile white sublimate (yellow hot)	....	+	....
	I. Linnæite $(Co \cdot Ni)_3S_4$ H=5.5 G=4.8 to 5	.....	+	.....	....	....	+
	I. Cobaltite $CoAsS$ H=5.5 G=6 to 6.1	+	+	Volatile white sublimate	....	....	+ (weak)
	I. Smaltite CoAsS H=5 to 6 G=6.4 to 6.6	+	.....	Volatile white sublimate	....	....	+
	O. Arsenopyrite FeAsS H=5.5 to 6 G=6 to 6.2	+	+	Volatile white sublimate	....	....	+
	I. Sperrylite $PtAs_2$ H=6 to 7 G=10.6	+	.....	Slight vol. sublimate	....	+	....

...-White, Lead-Gray or Steel-Gray in Color

...ility	Heated in closed tube	Other tests	Appearance
...dor ...	Fuses, yields subl. black hot, brownish red cold	Solution in upper part test tube repptd. as orange by H_2S from dissolving portion	Lead-gray columnar or fine-grained masses or prisms. Cleaves into lath-shaped fragments.
... ...	Decrepitates. A little white sublimate	Greenish yellow subl. on coal with Bi flux	Lead-gray granular and cleavable masses and cubic crystals which cleave into cubes.
... H_2S)	Subl. brownish red cold	Like galenite	Steel-gray to dark-gray needle crystals, or hair-like or felted; also compact and fibrous massive.
... NO_3	Fuses. Dark-red sublimate	Metallic residue ignited with HCl gives azure blue flame	Steel-gray fine-grained masses and tetrahedral crystals.
... NO_3	.....	Subl. on coal becomes bluish green by ignition with cobalt solution	Steel-gray, massive, granular. Often intermixed with yellow chalcopyrite.
... NO_3	Slight yellow sublimate	Borax deep blue O. F. and R. P.	Steel-gray granular or compact masses, or small octahedral crystals.
... NO_3	Unaltered	Like linnaeite	Gray masses and tin-white crystals. Often a red tarnish.
... NO_3	Mirror and black sublimate	Like linnaeite	Steel-gray masses and tin-white crystals usually cubes, often with erythrite.
... NO_3	Brownish red subl. Later mirror and black	After short ignition on coal, dissolves in HCl with odor of H_2S and yellow ppt.	Tin-white to gray masses or crystals often striated, the sections of which are rhombic and rectangular.
...	.....	In open tube white subl. and spongy residue	Tin-white grains and minute crystals

Table III. Minerals of Metalli

	Crystal system: name, composition, hardness and specific gravity	On coal in O. F. and R. F.					
		Odor:		Sublimates	Residue:		
		As	SO_2		None	Metallic	Magnetic
1. WITH BLACK STREAK	I. Bornite Cu_3FeS_3 H=3 G=4.9 to 5.4		+				+
	H. Millerite NiS H=3 to 3.5 G=5.3 to 5.6		+				+
	T. Chalcopyrite $CuFeS_2$ H=3.5 to 4.1 G=4.1 to 4.3		+				+
	I. Pentlandite (Fe · Ni) S H=3.5 to 4 G=4.6 to 5		+				+
	H. Pyrrhotite FeS H=3.5 to 4.5 G=4.5 to 4.6		+				+
	H. Niccolite NiAs H=5 to 5.5 G=7.2 to 7.7	+		Volatile, white			+
	I. Pyrite FeS_2 H=6 to 6.5 G=4.9 to 5.2		+				+
	O. Marcasite FeS_2 H=6 to 6.5 G=4.6 to 4.9		+				+
2. WITH STREAK NOT BLACK	I. Gold Au H=2.5 to 3 G=15.6 to 19.3					+	
	I. Copper Cu H=2.5 to 3 G=8.8 to 8.9					+	

Metallic Yellow, Bronze or Red in Color

...ability	Heated in closed tube	Other tests	Appearance
HNO_3 ...ndue	Blackens	Magnetic globule is brittle with red fracture and ignited with HCl gives azure blue flame	Red bronze on fresh fracture. Tarnishes in blue, purple and black tints. Very brittle and usually massive.
...aqua ...gia	.....	Roasted colors borax O. F. red hot, brown cold	Brass colored in hair-like or needle crystals. Crusts made up of radiating needles.
... HNO_3 ...ndue	Darkens, may give yellow sublimate	Like bornite except gray "fracture"	Bright-yellow brassy masses and crystals, tarnishing in peacock colors.
.....	.....	Fused globule yellow on fracture. Borax O. F. reddish brown	Light bronze-yellow masses resembling pyrrhotite but not attracted by a steel magnet. Cleavage octahedral.
...rvescs ...der ...)	A little S	Slightly magnetic before fusion	Bronze-yellow masses, tarnishing brown. Powder attracted by a steel magnet.
...tal	Mirror sublimate	Borax and roasted material give blue, green, brown, successively as borax is changed	Pale copper-red masses sometimes enclosed in white metallic crust.
... HNO_3 ...ndue	Fusible sublimate red hot, yellow cold	Fused mass effervesces in HCl with odor H_2S	Pale brass-yellow cubes or other crystals, isolated or grouped in crusts or bounding a mass. Also massive globular, nodular stalactitic.
HNO_3 ...ndue	Fusible sublimate red hot, yellow cold	Like pyrite	Pale brass-yellow "spear," "cockscomb" and simple tabular crystals. Often radiated. Fresh fracture whiter than that of pyrite.
aqua ...gia	.....	The fused mass is yellow	Golden yellow to pale yellow nuggets, grains and scales or distorted crystals passing into wire, fern and leaf forms. Malleable. Streak golden yellow to pale yellow.
HNO_3 ...ued)	.....	Fused mass is red and ignited with HCl gives azure blue flame	Copper red, disseminated grains sheets and irregular masses or groups of extended and branching crystals. Malleable. Streak copper red and shining.

Table IV. Minerals of Non-Metallic

	Crystal system: name, composition, hardness and specific gravity	Taste	Heated on cha
FUSED, POWDERED, MOISTENED WITH CONC. H_2SO_4, AND IGNITED ON PLATINUM WIRE			
1. FLAME YELLOW. ABSORBED BY "COLOR SCREEN"	H. Soda nitre..... $NaNO_3$ H=1.5 to 2 G=2.2	Cooling	F.=1. Deflagr
	M. Mirabilite,..... $Na_2SO_4+10\,H_2O$ H=1.5 to 2 G=1.5	Bitter	Fuses and will silver
	I. Halite..... NaCl H=2.5 G=2.4 to 2.6	Salty	F.=1.5
	M. Trona..... $NaCO_3$. $NaHCO_3+2\,H_2O$ H=2.5 to 3 G=2.1	Alkaline	Fuses easily
2. FLAME VIOLET THROUGH COLOR SCREEN	O. Carnallite..... $KCl.MgCl.6\,H_2O$ H=1 G=1.6	Salty and bitter	F.=1 to 1.5. with Co. sol.
	O. Nitre..... KNO_3 H=2 G=2.1	Salty and cooling	F.=1. Deflagr
	I. Kalinite..... $KAl(SO_4)_2+12\,H_2O$ H=2.5 G=1.7	Astringent	F.=1. Swells, will stain silve
	M. Kainite ..... $MgSO_4KCl+3\,H_2O$ H=2.5 to 3 G=2 to 2.2	Salty and bitter	Fuses easily fused with sc stain silver
3. FLAME GREEN	M. Borax. ... $Na_2B_4O_7+10\,H_2O$ H=2 to 2.5 G=1.7	Sweetish alkaline	F.=1 to 1.5. and gives cle
	Tri. Sassolite. ..... H_3BO_3 H=1 G=1.4	Acid	F.=2. With in cence to clea
	Tri. Chalcanthite . $CuSO_4.5\,H_2O$ H=2.5 G=2.1 to 2.3	Metallic, nauseous	Fuses. Reduc effervescence per button
4. FLAME NOT COLORED	M. Alunogen ... $Al_2(SO_4)_3.15\,H_2O$ H=1.5 to 2 G=1.6 to 1.8	Astringent	Fuses, loses wa comes infus Deep blue w sol.
	[illegible] $MgSO_4.7\,H_2O$ H=2 to 2.5 G=1.7	Bitter	Like alunoge junk with Co
	M. [illegible] $FeO.Al_2O_3.4\,SO_3.15\,H_2O$ H=2.5 G=2.1	Metallic nauseous	Fuses, become netic

With Decided Taste (Soluble in Water)

...ed in closed tube.	Recrystallization in a drop of water	Appearance
...: $KHSO_4$, brown ...por	Rhombic outlines	White, pale red or pale yellow masses and crystals with forms and angles of calcite.
...ter	Lath shaped	White efflorescence or powdery crust
...le water	Square (cubes)	White or colorless or impure brown, yellow or red masses and crystals with cubic cleavage.
...er	Spherulitic	White glistening crusts.
...ch water	Rectangles	White to reddish granular masses. Very deliquescent
...: $KHSO_4$, brown ...por	Lath shaped	White needles or thin crusts.
...ch water. Acid ... high heat	Octahedron. (Three-six- and four-sided polygons)	White fibres or mealy efflorescence.
...er	Rectangles	White to browniah red granular masses.
...h up. Gives ...ch water	Unsymmetrical polygons	Snow white crystals, crusts or porous masses.
...er and a little ...monia	Six-sided plates and threads	White pearly scales or plates.
...ds, whitens. ...ds water	Blue crystals	Blue glassy crystals, veins and crusts
...ch water. SO_2 and SO_3 at high ...t	Feathery	White efflorescence or fibrous crusts.
...er. Acid at high ...perature	Lath shaped	White fibres or crusts.
...ch acid water	No recrystallization	Yellow scales or granular masses.

Table V. Minerals of I

	Crystal system: name, composition, hardness and specific gravity	In powder boiled with hydrochl		
		There is efferves-cence	There is a residue of jelly	Simple solution
1. WITH BLUE STREAK	M. Vivianite $Fe_3(PO_4)_2+8H_2O$ H=1.5 to 2 G=2.6 to 2.7	.....	.....	+
	—. Chrysocolla $CuSiO_3 \cdot 2H_2O$ H=2 to 4 G=2 to 2.3	.....	.....	+ (Separatior of silica)
	M. Azurite $Cu_3(OH)_2(CO_3)_2$ H=3.5 to 4 G=3.8	+	.....	.....
	I. Lazurite $Na_4(NaS,Al)Al_2(SiO_4)_3$ H=5 to 5.5 G=2.6 to 2.9	+ (Odor H_2S)	+	.....
2. WITH GREEN STREAK	Chlorite Group H=1 to 2.5 G=2.6 to 2.9 (Micaceous dark-green minerals such as clinochlore $H_8Mg_5Al_2Si_3O_{18}$)	.....	.....	.....
	Garnierite $H_2(Ni \cdot Mg)SiO_4+H_2O$ H=2 to 3 G=2.3 to 2.8	.....	.....	+ (Separatior of silica)
	O. Atacamite $Cu_2(OH)_3Cl$ H=3 to 3.5 G=3.7 to 3.8	.....	.....	+ (Green)
	M. Malachite $Cu_2(OH)_2CO_3$ H=3.5 to 4 G=3.9 to 4	+	.....	.....
	O. Brochantite $CuSO \cdot 3Cu(OH)_2$ H=3.5 to 4 G=3.9	.....	.....	+
	Turquois $Al_2(OH)_3PO_4H_2O$ H=6 G=2.6 to 2.8	.....	.....	+

...ss, Tasteless and With Colored Streak

...ity ...ak	Heated in closed tube	Other tests	Appearance
...25 ...per)	Water. (Turns brown)	Momentary blue flame with H_2SO_4 (conc.). Yellow pt. with ammonic molybdate	Bluish-green to dark blue, often earthy and filling shells, horn, etc. Rarely as colorless or glassy crystals, gradually becoming blue.
...ble. ...kens)	Blackens. Yields water	.....	Enamel-like crusts, veins, or masses.
3 ...ack)	Blackens. Yields water	Emerald-green flame. Solutions blue with ammonia	Dark-blue glassy crystals, lighter blue crusts, velvety, dull or earthy masses.
35 ...ze)	Water. (Green glow)	Yellow flame	Deep-blue fine-grained masses usually spangled with pyrite and intermixed with other minerals.
...ored)	Much water	Milky solutions with concentrated H_2SO_4	Dark-green masses of coarse to very fine scales. Tabular and curiously twisted six-sided crystals and fan-shaped groups which cleave into thin soft pliable but not elastic plates. Also as a pigment in other minerals.
...ble. Mag- ...se)	Blackens. Yields water	Borax O. F. violet hot, brown cold	Dark emerald-green masses often cellular and very crumbly and paler-green masses and crusts. Lustre dull.
1 20 4 ...pper)	Acid water	Azure blue flame. White and red sublimates	Deep emerald-green, confused aggregates and slender prisms. Formerly found as a sand.
3 ...ack)	Blackens. Much water	Green flame, blue with HCl	Bright-green radiating fibres or crusts, often banded in shades of green, sometimes stalactitic. Also dull green and earthy. Rarely, slender crystals.
...per)	Water. May blacken. Acid at high temperature.	Green flame, blue with HCl. Fused with soda stains silver	Emerald-green needle crystals. Fibrous veins and crusts.
...ens)	Blackens. Yields water	Yellow ppt. as in vivianite. Green flame, blue with HCl	Sky-blue to green nearly opaque nodules or veins, with lustre of wax.

Table V (*Continued*). — Minerals of Non-

Crystal system: name, composition, hardness and specific gravity	In powder boiled with hydrochloric		
	There is effervescence	There is a residue of jelly	Simple solution
A. WITH YELLOW TO YELLOWISH-BROWN STREAK			
O. Goethite FeO (OH) H=5 to 5.5 G=4 to 4.4			+
Limonite $Fe_2O_3 \cdot Fe_2(OH)_6$ H=5 to 5.5 G=3.6 to 4			+
T. Rutile TiO_2 H=6 to 6.5 G=4.1 to 4.2			
T. Cassiterite SnO_2 H=6 to 7 G=6.8 to 7.1			
B. WITH RED TO REDDISH-BROWN STREAK			
M. Erythrite $Co_3(AsO_4)_2 \cdot 8H_2O$ H=1.5 to 2.5 G=2.9			+ (Light red)
H. Cinnabar HgS H=2 to 2.5 G=8 to 8.2			
H. Proustite Ag_3AsS_3 H=2 to 2.5 G=5.8 to 6.2			
H. Pyrargyrite Ag_3SbS_3 H=2.5 G=5.8			
Realgar As_2S_2 H=1.5 to 2 G=3.4 to 3.5			+

...ss, Tasteless and With Colored Streak

...ility ...	Heated in closed tube	Other tests	Appearance
... 5.5 ...	Water. Reddens	On coal R. F. strongly magnetic	Occurs massive but is best known as yellowish to brown and red needles, scales and velvety crusts. Streak yellow to yellowish brown.
... 5.5 ...	Much water. Reddens	Like goethite	Brown dull-lustred heterogeneous bog ore, cellular stalactitic and pipe-like concretions of rusty brown to nearly black, often fibrous smooth masses.
...ble	.....	S. Ph. O. F. slowly to yellow, made violet R. F.	Brownish red to nearly black crystals with brilliant lustre often parallel or netted. More rarely massive. Streak pale brown.
...ble	.....	On coal strongly heated and aided by soda or sulphur gives button and subl. yellow hot, white cold, bluish green if ignited with cobalt solution	Brown to red and nearly black. Dull, kidney-shaped and rounded pebbles. Brilliant crystals, and disseminated grains. Streak pale brown.
...	Water	Borax deep blue. O. F. and R. F.	Pink earthy crusts or powder or crimson fibres. Streak pink to crimson.
...	Black subl., red if rubbed	Closed tube with soda metallic mirror which can be collected into visible globules. Soluble aqua regia	Vermilion, scarlet and dark brownish-red fine-grained masses. Crystalline crusts. Streak scarlet.
...	Fuses. Slight red subl., yellow cold	Soluble HNO_3. Decomposed by boiling KOH and a yellow ppt. by HCl	Scarlet to vermilion crusts or masses. Rare six-sided prisms with brilliant adamantine lustre. Streak scarlet.
...	Fuses. Subl. black hot, red cold	As with proustite but ppt. is orange	Blackish red veins or crusts with a brilliant adamantine lustre. Red tint stronger in thin layers. Rare crystals. Streak purplish red.
...ite	Water at high heat	Deep blue if ignited with cobalt solution	Red to reddish-brown masses of rounded grains or clay-like. Dull in lustre. Streak reddish brown.

Table V (*Continued*). — Minerals of Ni

	Crystal system: name, composition, hardness and specific gravity	In powder boiled with hydrochlor		
		There is effervescence	There is a residue of jelly	Simple solution
4. WITH RED TO REDDISH-BROWN STREAK.	I. Cuprite..... Cu_2O H=3.5 to 4 G=5.8 to 6.1	.....	.....	+ (Brown)
	H. Ilmenite..... $FeTiO_3$ H=5 to 6 G=4.5 to 5	.....	.....	+ (Slowly)
	H. Hematite..... Fe_2O_3 H=5.5 to 6.5 G=4.9 to 5.3	.....	.....	+
5. WITH ORANGE STREAK.	M. Realgar..... As_2S_2 H=1.5 to 2 G=3.4 to 3.6	.....	.....	.....
	M. Crocoite..... $PbCrO_4$ H=2.5 to 3 G=5.9 to 6.1	.....	.....	+
	O. Descloizite..... $(PbOH)VO_4$ (Pb, Zn) H=3.5 G=5.9 to 6.2	.....	.....	+
	H. Zincite..... ZnO H=4 to 4.5 G=5.4 to 5.7	.....	.....	+

Lustre, Tasteless and With Colored Streak

[illegible]	Heated in closed tube	Other tests	Appearance
[illegible] (Copper)	.....	Ignited with HCl, azure-blue flame	Dark-red to brick-red masses Deep-red to crimson isometri crystals, sometimes hair-like Streak brownish red, shining
[illegible] ? [illegible]	.....	Solution boiled with tin becomes violet. Fused with soda is magnetic	Brownish-black to rusty-brown plates, grains and masses and thin tabular crystals. Streak brownish red.
[illegible]	.....	Dark-blue ppt. with potassic ferrocyanide	Dull dark red, massive, oolitic or earthy, sometimes kidney shaped and fibrous. Streak brownish red.
[illegible]	Boils gives subl. black hot, red cold	Soluble KOH. HCl ppts. yellow flakes. Soluble HNO_3	Orange-red granular masses o resinous lustre and transpar ent crystals. Streak orang red.
[illegible] (Lead)	With $KHSO_4$ dark violet hot, greenish cold	In S. Ph. O. F. and R. F. bright green. On coal Bi flux greenish yellow	Hyacinth red prisms. Streak orange yellow.
[illegible] (Lead)	Some water	In S. Ph. O. F. amber, R. F. green. On coal with Bi flux greenish yellow	Black, brown or red crusts of minute crystals. Streak brownish orange.
[illegible] [illegible]	Blackens	Sublimate ignited with cobalt solution is bright green	Deep red to brick-red adamantine masses. Granular or cleavable. Very rare crystals Streak orange yellow.

Table VI. Minerals of Non-Metallic Lustre, Tasteless and With

FUSED ON CHARCOAL WITH SODIC CARBONATE YIELDS:

1. NOTABLE FUMES OR SUBLIMATES

Crystal system: name, composition, hardness and specific gravity	The color of the mineral is						
	Blue	Green	Yellow	Brown	Red	Gray	Colorless
O. Sulphur..... S H=1.5 to 2.5 G=2 to 2.1	....	+	+	+	....	....	...
–Hydrozincite..... $Zn_5CO_8(OH)_4$ H=2 to 2.5 G=3.6 to 3.8	....	....	....	....	....	....	+
O. Valentinite..... Sb_2O_3 H=2.5 to 3 G=5.6	....	....	....	....	....	....	+
O. Anglesite..... $PbSO_4$ H=3 G=6.1 to 6.4	....	....	....	....	....	+	-
T. Wulfenite..... $PbMoO_4$ H=3 G=6.7 to 7	....	....	+	+	+	+	...
H. Vanadinite..... $Pb_5Cl(VO_4)_3$ H=3 G=6.6 to 7.2	....	....	+	+	+	....	.
O. Cerussite..... $PbCO_3$ H=3 to 3.5 G=6.5 to 6.6	....	....	+	+	....	+	
H. Pyromorphite..... $Pb_5Cl(PO_4)_3$ H=3.5 to 4 G=5.9 to 7.1	....	+	+	+	....	+	.

Soak, and Yielding Reactions on Charcoal With Sodic Carbonate

Solubility	Flame coloration	Heated in closed tube	Other tests	Appearance
Soluble	Blue	Yellow fusible. Sublimate brown hot	Takes fire and burns with odor SO_2	Bright translucent crystals and masses or powder with resinous or dull lustre.
Sol. with effervescence	.....	Water. Yellow hot	On coal R.F. heavy white subl. made bright green by ignition with cobalt solution	Chalk-like masses or crusts on other zinc minerals.
Volatile	Yellow green (Coal R. F.)	Fuses, partly sublimes	Volatile white subl. on coal	White silky minute crystals or radiating fibres.
Insoluble	.....	Decrepitates	Bright yellow subl. on coal with Bi flux	Simple crystals, often transparent and colorless. White brittle masses and compact granular masses of gray color from intermixed galenite.
Partial to green solution	.....	Darkens. Decrepitates	Like anglesite. Also solution, cooled, diluted boiled with tin is blue	Tabular square crystals of resinous lustre.
Sol. HNO_3 to yellow solution	.....	With $KHSO_4$ yellow to red hot, yellow cold	S. Ph. O. F. amber, R. F. green	Sharp hexagonal prisms, sometimes hollow. Also parallel groups and globular masses.
Sol. with effervescence in hot HCl. Crystals on cooling	.....	Turns yellow then red, cools yellow	Like anglesite	Twinned crystals or interlaced fibres or granular masses, often with galenite.
Sol. HNO_3	.....	.....	Like anglesite. Also fuses O. F. and on cooling has facets	Hexagonal prisms and tapering groups in parallel position. Also rounded and moss-like aggregations.

Table VI (*Continued*).— **Minerals of Non-Metallic Lustre, Tasteless**

FUSED ON CHARCOAL WITH SODIC CARBONATE YIELDS:		Crystal system: name, composition, hardness and specific gravity	The color of the mineral is:						
			Blue	Green	Yellow	Brown	Red	Gray	Colorless or white
	1. NOTABLE FUMES OR SUBLIMATES	I. Sphalerite ZnS H=3.5 to 4 G=3.9 to 4.1	....	....	+	+	+	....	+
		Calamine $(ZnOH)_2SiO_3$ H=4.5 to 5 G=4.3 to 4.5	+	+	+	+	....	....	+
		H. Smithsonite $ZnCO_3$ H=5 G=4.3 to 4.5	+	+	+	+	....	+	+
		H. Willemite Zn_2SiO_4 H=5.5 G=3.9 to 4.2	....	+	+	+	+	....	....
		T. Cassiterite SnO_2 H=6 to 7 G=6.8 to 7.1	....	....	+	+	....	....	....
	2. NO FUMES OR SUBLIMATES BUT IS REDUCED TO METALLIC OR MAGNETIC PARTICLES	I. Cerargyrite AgCl H=1 to 1.5 G=5 to 5.5	....	....	....	+	....	+	....
		I. Embolite Ag (Cl · Br) H=1 to 1.5 G=5.3 to 5.8	....	+	....	....	....	....	....

Yielding White Streak, and Yielding Reactions on Charcoal With Sodic Carbonate

[illegible]	Flame coloration	Heated in closed tube	Other tests	Appearance
with [illegible] and [illegible] H_2S	.....	.....	On coal R. F. heavy white subl. made bright green by ignition with cobalt solution	Transparent to translucent crystals and cleavable masses with strong resinous lustre. Compact masses or alternate layers with galenite. Rarely a white powder.
[illegible] with [illegible]	.....	Water	Like sphalerite (best if soda and borax added in ignition)	White masses, the cavities lined with crystals, often showing only ends, usually parallel, forming ridges. The fracture shows the crystals like parallel fibres.
[illegible] with [illegible]	.....	Yellow hot, if pure	Like sphalerite	Porous, cellular masses. Crusts with smooth rounded surfaces. Occasional drusy surfaces, the crystal ends being three-faced.
[illegible] with [illegible]	.....	.....	Like sphalerite	Granular masses intermixed usually with black and red grains. Rarely large reddish or brownish crystals. Lustre resinous.
[illegible]	.....	.....	On coal with soda non-volatile subl. made bluish green by ignition with cobalt solution	Crystals with brilliant adamantine lustre, disseminated grains and rounded heavy pebbles dull and often with radiating structure.
[illegible]	.....	With $KHSO_4$ yellow hot, white cold, violet in sun	On coal acrid odor and silver button	Thin crusts which darken in sunlight and cut like wax with shining surface after cutting.
[illegible]	.....	With $KHSO_4$ dark red hot, dark yellow cold, dark green in sun	Like cerargyrite	Like cerargyrite.

Table VI (*Continued*). — Minerals of Non-Metallic Lustre, Tasteless

FUSED ON CHARCOAL WITH SODIC CARBONATE YIELDS:		Crystal system name, composition, hardness and specific gravity	The color of the mineral is:						
			Blue	Green	Yellow	Brown	Red	Gray	Colorless or white
	2. NO FUMES OR SUBLIMATES, BUT IS REDUCED TO METALLIC OR MAGNETIC PARTICLES.	Chrysocolla $CuSiO_3 \cdot 2\,H_2O$ H = 2 to 4 G = 2 to 2.3	+	+					
		H. Siderite $FeCO_3$ H = 3.5 to 4 G = 3.8 to 3.9				+		+	
	3. NO FUMES, SUBLIMATES OR REDUCED PARTICLES, BUT IF MOISTENED WILL STAIN SILVER	M. Gypsum $CaSO_4\,2\,H_2O$ H = 1.5 to 2 G = 2.3				+	+	+	+
		O. Barite $BaSO_4$ H = 2.5 to 3.5 G = 4.3 to 4.6	+		+	+	+	+	+
		O. Celestite $SrSO_4$ H = 3 to 3.5 G = 4	+					+	+
		O. Anhydrite $CaSO_4$ H = 3 to 3.5 G = 2.9 to 3	+				+	+	+
		H. Alunite $K\,Al_3\,(SO_4)_2 + 6\,H_2O$ H = 3.5 to 4.5 G = 2.6 to 2.8							+
		I. Lazurite (also Lazurite) $Na_4\,Na_2S_3O$ $Al_3\,Si_3O_{12}$ H = 5 to 5.5 G = 2.4	+						

With White Streak and Yielding Reactions on Charcoal With Sodic Carbonate

Solubility	Flame coloration	Heated in closed tube	Other tests	Appearance
[illegible] with [illegible]	Emerald green	Blackens. Yields water	With soda on coal a copper button	Enamel-like crusts, veins or compact masses. Never crystals.
Slow effervescence in cold acid	.....	Black and magnetic	On coal becomes black and magnetic	Compact, fine-grained and cleavable masses and rhombohedral curved crystals. Cleavage angles 107°.
[illegible] Recrystallizes on evaporation	Yellowish red	Whitens. Yields water	.....	Soft, colorless or slightly tinted masses, which may be scaly, silky fibrous or compact or may be masses and crystals, cleaving in three directions to a rhombic plate of 66°.
Insoluble	Yellowish green	.....	.....	Granular, cleavable and fibrous. Crystals common. Cleaves in three directions to rhombic plates of 78½°.
Insoluble	Crimson	.....	.....	Colorless to white crystals and fibrous, lamellar and granular masses. Cleaves in three directions to rhombic plates of 76°.
Soluble [illegible] with $BaCl_2$	Yellowish red	.....	.....	Cleavable and fine-grained masses. Cleaves in three directions at 90°.
Mineral	Violet. (Color Screen)	Water at high heat	Blue by ignition with co. solution	Fibrous or lamellar or small cuboids, usually mixed with hard, siliceous material.
[illegible] with [illegible] and odor H_2S	Yellow	Water and green glow	Blue in fine powder	Fine-grained, usually spangled with pyrite and intermixed with other minerals.

Table VII. Minerals of Non-Metallic Lustre, Tastel

Crystal system: name, composition, hardness and specific gravity	The color of the mineral is: Purple or violet	Blue	Green	Yellow	Brown	Red	Gray
2. IF FRAGMENT HEATED IN THE PLATINUM FORCEPS AT THE TIP OF THE BLUE FLAME							
1. IS FUSED EASILY (1 TO 4) TO A WHITE GLASS OR ENAMEL							
M. Lepidolite $(KLi)_2Al(SiO_3)_3$ H=2 to 2.5 G=2.8 to 3.2	+	.....	.....	.....	.....	+	+
M. Cryolite $AlNa_3F_6$ H=2.5 G=2.9 to 3	.....	.....	.....	.....	+	.....	+
I. Fluorite CaF_2 H=4 G=3 to 3.3	+	+	+	+	+	+	.....
M. Stilbite $H_4R_2Al_2(SiO_3)_6 + 4H_2O$ H=3.5 to 4 G=2.1 to 2.2	.....	.....	.....	+	+	.....	.....
H. Chabazite $(CaNa_2)Al(SiO_4)_2 6H_2O$ H=4.5 G=2.0 to 2.1	.....	.....	.....	.....	.....	+	.....
T. Apophyllite $H_{14}K_2Ca_8(SiO_3)_{16} 9H_2O$ H=4.5 to 5 G=2.3 to 2.4	.....	.....	.....	.....	.....	+	.....
T. Wernerite Group Silicates of NaCaAl H=5 to 6 G=2.7	.....	+	+	.....	.....	+	+
Tri Plagioclase $nNaAlSi_3O_8 + mCaAl_2Si_2O_8$ H=5 to 7 G=2.6 to 2.7	.....	.....	.....	.....	.....	.....	.....
(Albite)	.....	.....	.....	.....	.....	.....	.....
(Labradorite)	.....	.....	.....	.....	.....	.....	+

In White Streak and Yielding No Tests With Sodic Carbonate

[illegible]	Flame coloration	Heated in closed tube	Other tests	Appearance
[illegible]	Crimson	.....	.....	Masses of coarse or fine scales with easy cleavage into thinner plates.
[illegible]	Yellow	Etches tube	Blue if ignited with Co. solution.	Translucent masses resembling watery snow. Rarely small six-sided monoclinic crystals nearly cubes.
[illegible]	Red to orange	Phosphorescent. With $KHSO_4$ etches tube	.....	Transparent cubes and masses of glassy lustre which cleave in four directions at angles 70° 31′. Color usually brilliant. Sometimes banded.
[illegible] with [illegible]	.....	Water	Swells greatly during fusion	Sheaf-like groups or many small crystals forming a crust or lining. One easy cleavage giving symmetrical pearly face.
[illegible] with [illegible]	.....	Much water	Intumescence during fusion	Groups of small rhombohedral crystals which are nearly cubes.
[illegible] with [illegible]	Pale violet (Color Screen)	Much water	Exfoliates during fusion	Tabular or "cubic" or pointed crystals with internal opalescence. Occasionally lamellar. One easy cleavage.
[illegible]	Yellow	.....	Bubbles in fused material	Coarse thick crystals with octagonal or square cross-section. Cleavage angles 135° or 90°. Cleavages faintly fibrous. More rarely massive columnar or fine-grained aggregates.
[illegible]	.....	.....	.....	Masses and crystals with two easy cleavages, nearly but not exactly 90°. Often show parallel striations. Sometimes opalescent.
[illegible]	Yellow	.....	.....	
[illegible]	.....	.....	.....	

Table VII (*Continued*). — Minerals of Non-Metallic Lustre, Ta

A THIN FRAGMENT HEATED IN THE PLATINUM FORCEPS AT THE TIP OF THE BLUE FLAME	Crystal system: name, composition, hardness and specific gravity	The color of the mineral is:						
		Purple or violet	Blue	Green	Yellow	Brown	Red	Gray
1. IS FUSED EASILY (1 TO 4) TO A WHITE GLASS OR ENAMEL	Tri. Amblygonite..... $Li(AlF)PO_4$ H=6 G=3 to 3.1	.....	.....	.....	.....	.....	.....	.....
	O. Prehnite..... $H_2Ca_2Al_2(SiO_4)_3$ H=6 to 6.5 G=2.8 to 2.9	.....	.....	+	+	.....	.....	.....
	H. Tourmaline..... $R_{18}B_2(SiO_3)_4$ H=6 to 6.5 G=2.8 to 2.9	.....	.....	.....	.....	+	.....	.....
	I. Boracite..... H=7 G=2.9 to 3	.....	.....	.....	.....	.....	.....	.....
2. IS FUSED EASILY (1 TO 4) TO A COLORLESS GLASS	Ulexite..... $CaNaB_5O_9 \cdot 8H_2O$ H=1 G=1.65	.....	.....	.....	.....	.....	.....	.....
	M. Colemanite..... $Ca_2B_6O_{11} \cdot 5H_2O$ H=4 to 4.5 G=2.2 to 2.3	.....	.....	.....	.....	.....	.....	.....
	O. Natrolite..... $Na_2Al_2Si_3O_{10}+2H_2O$ H=5 to 5.5 G=2.2	.....	.....	.....	+	.....	.....	.....
	I. Analcite..... $NaAl(SiO_3)_2 \cdot H_2O$ H=5 to 5.5 G=2.2 to 2.3	.....	.....	.....	.....	.....	+	.....
	M. Datolite..... $Ca(BOH)SiO_4$ H=5 to 5.5 G=2.9 to 3	.....	.....	.....	.....	.....	.....	.....

...d With White Streak and Yielding No Tests With Sodic Carbonate

...ubility	Flame coloration	Heated in closed tube	Other tests	Appearance
...uble	Crimson to yellowish red	Water and etching of tube	Momentary blue-green flame with H_2SO_4	Cleavable masses and rough crystals. One easy cleavage.
...ble	.....	A little water	After fusion dissolves leaving jelly	Lining cavities as smooth rounded crusts or as sheaf-like groups of tabular crystals. Sometimes in barrel-shaped crystals.
...uble	Green with $KHSO_4$ + CaF_2	.....	After ignition dissolves leaving jelly	Prismatic crystals often hemimorphic and roughly triangular in cross-section.
...ble	Yellowish green	.....	Violet if ignited with cobalt solution	Minute glassy crystals.
...ble	Reddish yellow	Much water	Green flame with $KHSO_4$ and CaF_2	Nodular masses of silky fibres.
...uble with ...stals ...cool- ...ng	Green	Water	Decrepitates before fusing	Highly modified crystals with one easy cleavage, cleavable or fine-grained compact "porcelain-like" or loose, chalk-like masses.
...ble with ...ly	Yellow	Water	.....	Slender prisms with square cross-section and flat pyramid at end.
...ble with ...ps ...ly	Yellow	Water, but keeps lustre	Becomes opaque before fusion	Trapezohedral crystals usually forming a lining. Rarely granular.
...ble with ...y	Green	Water at high heat	.....	Brilliant small highly modified glassy crystals lining a cavity, also porcelain masses. No easy cleavage.

Table VII (*Continued*). — Minerals of Non-Metallic Lustre, Tastel

A THIN FRAGMENT HEATED IN THE PLATINUM FORCEPS AT THE TIP OF THE BLUE FLAME	Crystal system: name, composition, hardness and specific gravity	The color of the mineral is:							
		Purple or violet	Blue	Green	Yellow	Brown	Red	Gray	Colorless
1. IS FUSED EASILY (1 TO 4) TO A WHITE GLASS OR ENAMEL	Tri. Amblygonite $Li(AlF)PO_4$ H=6 G=3 to 3.1								+
	O. Prehnite $H_2Ca_2Al_2(SiO_4)_3$ H=6 to 6.5 G=2.8 to 2.9			+	+				
	H. Tourmaline $R_{18}B_2(SiO_3)_4$ H=6 to 6.5 G=2.8 to 2.9					+			
	I. Boracite H=7 G=2.9 to 3								+
2. IS FUSED EASILY (1 TO 4) TO A COLORLESS GLASS	Ulexite $CaNaB_5O_9 \cdot 8H_2O$ H=1 G=1.65								+
	M. Colemanite $Ca_2B_6O_{11} \cdot 5H_2O$ H=4 to 4.5 G=2.2 to 2.3								+
	O. Natrolite $Na_2Al_2Si_3O_{10}+2H_2O$ H=5 to 5.5 G=2.2				+				+
	I. Analcite $NaAl(SiO_3)_2 \cdot H_2O$ H=5 to 5.5 G=2.2 to 2.3						+		+
	M. Datolite $Ca(BOH)SiO_4$ H=5 to 5.5 G=2.9 to 3								+

Table VII (*Continued*). — Minerals of Non-Metallic Lustre, Th

A THIN FRAGMENT HEATED IN THE PLATINUM FORCEPS AT THE TIP OF THE BLUE FLAME	2. IS FUSED EASILY (1 TO 4) TO A COLORLESS GLASS	Crystal system: name, composition, hardness and specific gravity	The color of the mineral is: Purple or violet	Blue	Green	Yellow	Brown	Red	Gray
		H. Nephelite..... $Na_6Al_6Si_6O_{24}$ H=5.5 to 6 G=3.2 to 3.6	.....	.....	+	.....	+	+	+
		Pyroxene (diopside)..... $CaMg(SiO_3)_2$ H= 5 to 6 G=3.2 to 3.6	.....	.....	+	.....	.....	.....	.....
		M. Amphibole (tremolite).. $CaMg_3(SiO_3)_4$ H=5 to 6 G=2.9 to 3.4	.....	.....	+	.....	.....	.....	+
		Tri. Plagioclase..... $nNaAlSi_3O_8 + mCaAl_2Si_2O_8$ H=5 to 7 G=2.6 to 2.7	.....	.....	.....	.....	.....	.....	.....
		(Albite) $NaAlSi_3O_8$.....	.....	.....	.....	.....	.....	.....	+
		(Anorthite) $CaAl_2Si_2O_8$...	.....	.....	.....	.....	.....	+	+
		(Oligoclase) $Ab_{2\text{ to }6}$ An	.....	.....	.....	.....	.....	.....	+
		(Labradorite) $AbAn_{1\text{ to }3}$..	.....	.....	.....	.....	.....	.....	+
		M. Spodumene..... $LiAl(SiO_3)_2$ H=6.5 to 7 G=3.1 to 3.2	+	.....	+	.....	.....	.....	.....

With White Streak and Yielding No Tests with Sodic Carbonate

[illegible]	Flame coloration	Heated in closed tube	Other tests	Appearance
[illegible]	Yellow	.....	Blue if ignited with cobalt solution	Translucent masses and coarse hexagonal crystals with peculiar greasy lustre. More rarely highly modified small white crystals.
[illegible]	.....	.....	.....	Usually prismatic crystals with eight-sided cross-section and angles between alternate faces 90° or 87°. Cleavage angle of 87°.
[illegible]	.....	.....	.....	Fibrous and columnar, often radiating. Also crystals with cross-section, a rhomb of 124° or six-sided section. Cleavage at 124°.
[illegible]	.....	.....	.....	Masses and crystals with two easy cleavages, nearly but not exactly 90°. Often show parallel striations. Sometimes opalescent.
[illegible]	Yellow	.....	.....	Usually pure white, often granular with curved cleavage surfaces.
[illegible]	.....	.....	White ppt. with H_2SO_4	Highly modified glassy crystals or grayish-white larger crystals
[illegible]	Yellow	.....	.....	Broad cleavages. Good striations. Sometimes spangled (Sunstone).
[illegible]	.....	.....	.....	Good striations. Beautiful play of color.
[illegible]	Crimson	.....	Sprouts and becomes opaque during fusion	Coarse, flattened prisms with cleavage at 87°. Often separate in broad plates (bisecting cleavage angle). Often striated and etched or roughened.

Table VII (*Continued*). — Minerals of Non-Metallic Lustre, T

A THIN FRAGMENT HEATED IN THE PLATINUM FORCEPS AT THE TIP OF THE BLUE FLAME

2. IS FUSED EASILY (± TO 4) TO A COLORED GLASS

Crystal system: name, composition, hardness and specific gravity	The color of the mineral is:						
	Blue	Green	Yellow	Brown	Red	Gray	Colorless
M. Titanite..... $CaSiTiO_5$ H=5 to 5.5 G=3.4 to 3.5	.....	+	+	+	.....	.....	...
Pyroxene..... $RSiO_3$. Many varieties (Augite) (R = CaMgFeAl) H=5 to 6 G=3.2 to 3.6	.....	+	.....	+	.....	.....	...
M. Amphibole $RSiO_3$ H=5 to 6 G=2.9 to 3.4 (Actinolite)..... $Ca(MgFe)_3(SiO_3)_4$	.....	+	.....	.....	+	.....	...
(Hornblende)..... CaMgFeAl, etc.	.....	+	.....	.....	.....	.....	...
O. Hypersthene..... $(Mg \cdot Fe)SiO_3$ H=5 to 6 G=3.4 to 3.5	.....	+	.....	.....	.....	.....	...
Tri. Rhodonite..... $MnSiO_3$ H=6 to 6.5 G=3.4 to 3.7	.....	...	....	+	+	.....	...
M. Epidote ..... $Ca_2(Al \cdot Fe)_3(OH)(SiO_4)_3$ H=6 to 7 G=3.3 to 3.5	.	+	..	.	+	...	...
T. Vesuvianite $Ca_8Al_2(OH \cdot F)_2Si_5O_{19}$ H=6.5 G=3.3 to 3.4	.	+	+	+	..	....	..
[illegible] $K_2Al_2(SiO_4)_3$ H=[illegible] G=4.1 to 4.5 [illegible]		+	+	+	+	....	...

and With White Streak and Yielding No Tests With Sodic Carbonate

[illegible]	Flame coloration	Heated in closed tube	Other tests	Appearance
[illegible] slowly	.....	May become yellow	S. Ph. O. F. slowly soluble. Undissolved portion milk white, R. F. violet	Wedge-shaped or tabular crystals, with adamantine lustre. Also massive. Easy cleavages give monoclinic shapes.
[illegible] nearly	.....	.....	.....	Usually eight-sided prisms with angles between alternate faces 90° and 87°. Cleavage angle 87°.
[illegible]	.....	.....	.....	Bladed non-terminated crystals, divergent fibres and granules.
[illegible] nearly	.....	.....	.....	Crystals six-sided cross-section, with angles 124° and 116°, also fibrous and compact masses. Sometimes with lustre of horn.
[illegible] [illegible]	.....	.....	After fusion attracted by magnet	Foliated aggregates sometimes with peculiar "Schiller" or pearly effect.
[illegible] with [illegible]	.....	.....	Borax, O. P. amethystine	Fine-grained or cleavable masses and disseminated grains, often coated with a black oxide. Sometimes in crystals.
[illegible] nearly	.....	Water at high heat	After fusion will gelatinize	A secondary mineral often with the original mineral as grains or needles. Less frequently in distinct crystals.
[illegible] nearly	.....	Water at high heat	After fusion will gelatinize	Square and octagonal prisms and radiated columnar or granular masses or compact resembling jade.
[illegible]	.....	.....	After fusion will gelatinize	Imbedded crystals, often nearly spherical or in druses and granular, lamellar and compact masses. Also found in alluvial material as rounded grains.

Table VII (*Continued*).—Minerals of Non-Metallic Lustre,

A THIN FRAGMENT HEATED IN THE PLATINUM FORCEPS AT THE TIP OF THE BLUE FLAME

	Crystal system: name, composition, hardness and specific gravity	The color of the mineral is:						
		Blue	Green	Yellow	Brown	Red	Gray	Colorless
3. (*Continued.*)	H. Tourmaline..... $R_{15}B_2(SiO_3)_4$ H=7 to 7.5 G=3 to 3.2	+	+	.....	+	.....	.....	...
4. IS FUSED WITH DIFFICULTY ONLY (5 TO 6) SOMETIMES BARELY ROUNDING THE EDGES	Pyrophyllite..... $HAl(SiO_3)_2$ H=1 to 2 G=2.8 to 2.9	.....	+	.....	.....	+	.....	+
	O. Talc..... $H_2Mg_3(SiO_3)_4$ H=1 to 1.5 G=2.5 to 2.9	.....	+	.....	.....	.....	+	+
	M. Chlorite Group (Prochlorite)..... $H_9(Mg, Fe)_5Al_2Si_3O_{20}$ H=1 to 2 G=2.8 to 2.9	.....	+	.....	.....	.....	.....	.....
	(Clinochlore)..... $(H_8Mg_5Al_2Si_3O_{18})$ H=1 to 2.5 G=2.6 to 2.9	.....	+	.....	.....	.....	.....	...
	–Sepiolite..... $H_4Mg_2Si_3O_{10}$ H=2 to 2.5 G=1 to 2	.....	+	.....	.....	.....	.....	+
	M. Muscovite..... $(HK)AlSiO_4$ H=2 to 2.5 G=2.8 to 3	.....	+	+	+	.....	+	....
	M. Biotite..... $(H \cdot K)_2(Mg \cdot Fe)_2Al_2(SiO_4)_3$ H=2.5 to 3 G=2.7	.....	+	.....	.....	.....	.....	.....
	M. Phlogopite $(K \cdot H)_2Mg_3Al_2(SiO_4)_3$ H=2 to 3 G=2.8	.....	.....	...	+	+	.....	.....

With White Streak and Yielding No Tests With Sodic Carbonate

	Flame coloration	Heated in closed tube	Other tests	Appearance
	Green with $KHSO_4$ - CaF_2	.....	After fusion will gelatinize	Prismatic crystals, the cross-section often showing a triangular prism. Often the color is different at opposite ends or centre and outer shell. Also radiating aggregates and in compact masses.
	.....	Water	Blue if ignited with cobalt solution	Radiated folia or fibres and compact masses. Smooth and soft like talc.
	.....	Water	Pink if ignited with cobalt solution	Foliated compact and fibrous masses with soapy feeling. The foliated talc cleaves into non-elastic plates.
	.....	Much water	.....	Masses of coarse to very fine scales. Tabular and curiously twisted six-sided crystals and fan-shaped groups which cleave into thin, soft pliable but not elastic plates. Also as a pigment in other minerals.
	.....	Much water	.....	
	.....	Water	Pink if ignited with cobalt solution	Soft, compact, smooth feeling masses of very light weight. Rarely fibrous.
	.....	Water at high heat	.....	Plates and masses of scales and crystals, often large and rough, with rhombic or hexagonal cross-section. Lustre pearly, cleavage very easy into thin elastic plates.
	.....	Water at high heat	.....	Scales or aggregates. Rarely large sheets or pseudo hexagonal crystals cleaving easily into thin elastic plates. Lustre pearly.
	.....	Water at high heat	.....	Rough prisms with hexagonal or rhombic sections. Also disseminated scales. Cleaves easily into thin elastic plates.

Table VII (*Continued*). — Minerals of Non-Metallic Lustre, T

Crystal system: name, composition, hardness and specific gravity	The color of the mineral is: Blue	Green	Yellow	Brown	Red	Gray	Colorless or white
A THIN FRAGMENT HEATED IN THE PLATINUM FORCEPS AT THE TIP OF THE BLUE FLAME							
4. IS FUSED WITH DIFFICULTY ONLY (5 TO 6) SOMETIMES BARELY ROUNDING THE EDGES							
—Serpentine..... $H_4Mg_3Si_2O_9$ H=3 to 4 G=2.5 to 2.6	.....	+	+	+	+	.....	+
O. Strontianite..... $SrCO_3$ H=3 to 3.5 G=3.7	.....	.....	.....	.....	.....	.....	+
M. Wollastonite..... $CaSiO_3$ H=4 to 5 G=2.8 to 2.9	.....	.....	.....	.....	.....	+	+
T. Scheelite..... $CaWO_4$ H=4.5 to 5 G=5.9 to 6.1	.....	.....	.....	+	.....	+	+
H. Apatite..... $Ca_5(Cl \cdot F)(PO_4)_3$ H=4.5 to 5 G=3.2	+	+	+	+	+	+	+
O. Enstatite..... $(Mg \cdot Fe)SiO_3$ H=5.5 G=3.1 to 3.3	.....	+	.....	+	.....	+	.....
Orthoclase..... $KAlSi_3O_8$ H=6 to 6.5 G=2.5 to 2.6	.....	+	+	.....	+	.....	+
H. Tourmaline..... $R_{10}B_2(SiO_3)_4$ H=7 to 7.5 G=3 to 3.2	+	+	.....	+	.....	.....	.....
H. Beryl..... $Be_3Al_2(SiO_3)_4$ H=7.5 to 8 G=2.6 to 2.8	+	+	+	.....	+	.....	+

[illegible] White Streak and Yielding No Tests With Sodic Carbonate

[illegible]	Flame coloration	Heated in closed tube	Other tests	Appearance
[illegible]	.....	Water	Pink or brownish red if powder ignited with cobalt solution	Compact masses with little lustre and smooth somewhat greasy feel, often with veins of silky fibres or foliated.
[illegible]	Crimson	.....	Sprouts and glows intensely during fusion	Masses of parallel or radiating imperfect needle crystals. More rarely fine granular.
[illegible]	Pale red	.....	.....	Fibrous to compact masses. Rarely tabular crystals. Usually intermixed with calcite.
[illegible]	Pale red	.....	S. Ph. O. F. colorless to milk white. R. F. deep blue	Very heavy masses with resinous lustre. Square pyramids and drusy crusts.
[illegible]	Yellowish red. Momentary green with H_2SO_4	.....	Solution added to nitric sol. of ammonia molybdate gives bright yellow ppt.	Usually hexagonal prisms. Lustre of oiled glass, dull if altered. Also compact, dull, massive bone phosphate.
[illegible]	.....	.....	Like serpentine	Lamellar to fibrous masses, often with pearly lustre.
[illegible]	Violet (Color screen)	.....	.....	Masses and crystals which cleave in two directions at exactly 90°. Except in the variety microcline the surfaces resulting are not grooved. Sometimes opalescent.
[illegible]	Green with $KHSO_4$ CaF_2	.....	After fusion will gelatinize	Prismatic crystals, often showing a triangular prism. The color may differ at opposite ends or centre and outer shell. Also radiating aggregates and in compact masses.
[illegible]	.....	.....	Often becomes white on fusion	Hexagonal prisms, from mere threads to several feet in length. Sometimes also in columnar or granular masses.

Table VII (*Continued*). — Minerals of Non-Metallic Lustre.

A THIN FRAGMENT HEATED IN THE PLATINUM FORCEPS AT THE TIP OF THE BLUE FLAME

5. IS INFUSIBLE BUT IN POWDER IS MADE DEEP BLUE BY IGNITION WITH COBALT SOLUTION

Crystal system: name, composition, hardness and specific gravity	The color of the mineral is: Purple or violet	Blue	Green	Yellow	Brown	Red	Gray	Colorless
M. Aluminite $Al_2SO_{11}9H_2O$ H=1 to 2 G=1.6								+
–Bauxite $Al_2O(OH)_4$ H=1 to 3 G=2.4 to 2.5					+	+		+
–Kaolin (in part) $H_4Al_2Si_2O_9$ H=2 to 2.5 G=2.6			+	+	+	+		+
M. Gibbsite $Al(OH)_3$ H=2.5 to 3.5 G=2.4								+
O. Andalusite (Chiastolite) Al_2SiO_5 H=4 to 5 G=2.1							+	+
Tri. Cyanite Al_2SiO_5 H=5 to 7 G=3.6 to 3.7		+	+					+
Leucite $KAl(SiO_3)_2$ H=5.5 to 6 G=2.4 to 2.5							+	+
O. Sillimanite Al_2SiO_5 H=6 to 7 G=3.2					+		+	
O. Andalusite $Al(AlO)SiO_4$ H=7 to 7.5 G=3.1 to 3.2			+			+	+	
Topaz $Al_{12}Si_6O_{28}F_{10}$ H=8 G=3.4 to 3.6		+		+	+			+

[illegible] With White Streak and Yielding No Tests With Sodic Carbonate

[illegible]	Flame coloration	Heated in closed tube	Other tests	Appearance
[illegible]	.....	Much acid water. Odor SO_2	Infusible with soda but mass will stain silver	Rounded chalky masses with peculiar, harsh (meagre) feel.
[illegible] slowly	.....	Water at high heat	May become magnetic in R. F.	Masses of rounded grains (pisolites or oolites) or earthy or clay-like. No lustre.
[illegible]	.....	Water	Often plastic with water	Dull clay-like or mealy masses. Greasy feel.
[illegible]	.....	Water	Exfoliates on heating	Small stalactites or thin, smooth crusts, with internally fibrous structure. Rarely in small crystals.
[illegible]	.....	.....	.....	Coarse, rounded prisms. Often superficially black. Cross-sections show a cross or checked figure.
[illegible]	.....	.....	.....	Triclinic blade-like crystals and blade-like masses, cleaving parallel largest face. Color deeper along centre.
[illegible] with [illegible]	Violet. (Color screen)	.....	.....	Translucent nearly spherical crystals and grains in volcanic rocks.
[illegible]	.....	.....	.....	Thin, almost fibrous prisms and tough fibrous aggregates.
[illegible]	.....	.....	.....	Coarse, nearly square prisms or tough, columnar or granular masses.
[illegible]	.....	.....	Heated in open tube with fused S. Ph. Etches glass	Glassy crystals with one easy cleavage. Also columnar aggregates, and water-worn crystals in alluvial deposit.

Table VII (*Continued*).—Minerals of Non-Metallic Lustre, 1

A THIN FRAGMENT HEATED IN THE PLATINUM FORCEPS AT THE TIP OF THE BLUE FLAME	Crystal system: name, composition, hardness and specific gravity	The color of the mineral is:							
		Purple or violet	Blue	Green	Yellow	Brown	Red	Gray	Colorless
5. IS INFUSIBLE BUT IN POWDER IS MADE DEEP BLUE BY IGNITION WITH COBALT SOLUTION	I. Spinel $MgAl_2O_4$ H=8 G=3.5 to 4.5	....	....	+	....	....	+	+	....
	O. Chrysoberyl $BeAl_2O_4$ H=8.5 G=3.5 to 3.8	....	....	+	+	....	....	....	....
	H. Corundum Al_2O_3 H=9 G=3.9 to 4.1	....	+	....	....	+	....	+	....
	(Sapphire) or (Ruby)	+	+	+	+	....	+	.....	+
	(Emery)	....	....	....	....	....	....	+	....
6. IS INFUSIBLE AND IS *NOT* MADE DEEP BLUE BY IGNITION WITH COBALT SOLUTION	H. Calcite $CaCO_3$ H=3 G=2.7	+	+	+	+	+	+	.....	+
	O. Aragonite $CaCO_3$ H=3.5 to 4 G=2.9	+	....	....	+	+	+	+	+
	H. Dolomite $CaMg(CO_3)_2$ H=3.5 to 4 G=2.8 to 2.9	....	....	....	....	+	+	+	+

... With White Streak and Yielding No Tests With Sodic Carbonate

...	Flame coloration	Heated in closed tube	Other tests	Appearance
...	.....	.....	Often changes color on heating	Simple or twinned octahedral crystals and rolled pebbles.
...	.....	.....	.....	Usually pseudohexagonal crystals or pebbles. Emerald green crystals by transmitted light are purplish red; some pebbles show an internal opalescence.
...	.....	.....	.....	Coarse crystals or masses with partings in four directions at 86° and 57°, or granular, slightly translucent.
...	.....	.....	Color changed by heating	Transparent to translucent, usually in crystals and of fine colors.
	.....	.....	.....	Opaque, granular corundum, intimately mixed with hematite or magnetite.
...	Orange red	.....	Unchanged when boiled with cobalt solution	Crystals of many shapes which cleave in three directions to rhombohedron of 105°. Cleavable, coarse and fine-grained, fibrous and loosely coherent masses. Crusts, stalactites.
... calcite	Orange red	.....	Becomes lilac if boiled with cobalt solution	Simple or pseudohexagonal crystals. Also columnar and needle masses, oolitic, stalactitic and coral-like. Two easy cleavages with angles near 120° (116°, 122°).
...	Orange red	.....	Pink if ignited with cobalt solution	Curved rhombohedral crystals, or coarse to fine-grained masses. Cleaves in three directions to rhombohedron of 106°.

Table VII (*Continued*). — Minerals of Non-Metallic Lustre, T

A THIN FRAGMENT HEATED IN THE PLATINUM FORCEPS AT THE TIP OF THE BLUE FLAME	Crystal system: name, composition, hardness and specific gravity	The color of the mineral is:							
		Purple or violet	Blue	Green	Yellow	Brown	Red	Gray	Colorless
6. IS INFUSIBLE AND IS *NOT* MADE DEEP BLUE BY IGNITION WITH COBALT SOLUTION	H. Magnesite $MgCO_3$ H=3.5 to 4.5 G=3 to 3.1	....	....	....	....	+	....	....	+
	H. Rhodochrosite $MnCO_3$ H=4.5 G=3.5 to 4.5	....	....	....	....	+	+	....	....
	M. Monazite $(Ce \cdot La \cdot Di)\ PO_4$ H=5 to 5.5 G=4.9 to 5.3	....	....	....	+	+	+	....	....
	Turquois $Al_2(OH)_3PO_4H_2O$ H=5 to 6 G=2.6	....	+	+	....	....	....	....	....
	M. Chondrodite $H_2Mg_{10}Si_4O_{18}F_4$ H=6.5 G=3.1 to 3.2	....	....	....	+	+	+	....	....
	O. Chrysolite $(MgFe)_2SiO_4$ H=6.5 to 7 G=3.3 to 3.6	....	....	+	+	....	....	....	....
	Opal SiO_2nH_2O H=5.5 to 6.5 G=2.1 to 2.2	....	+	+	+	+	+	+	+
	T. Rutile TiO_2 H=6 to 6.5 G=4.1 to 4.2	....	....	....	....	+	+	....	....
	Chalcedony SiO_2 H=6.5 G=2.6	....	+	+	....	....	+	+	+

...ut With White Streak and Yielding No Tests With Sodic Carbonate

...olubility	Flame coloration	Heated in closed tube	Other tests	Appearance
...ervescences ...ly in ...arm ...cid	.....	.....	Like dolomite	Compact, dull nodules or veins in serpentine. Shell-like fracture. Rarely cleavable.
...ite dolo...	.....	.....	Darkens on ignition. Borax, O. F. amethystine	Rhombohedral crystals often with curved edges cleavable and granular masses. Sometimes as a crust.
...luble ...ite ...ence	Momentary green with H_2SO_4	.....	Yellow ppt. if solution added to nitric solution of ammonic molybdate.	Translucent grains in some sands and small imbedded resinous crystals.
...d. HNO_3	Green (Cu)	.....	Like monazite	Nearly opaque material with wax-like lustre found filling cracks and cavities in igneous rocks.
...s with ...	.....	.....	Heated in open tube with fused S. Ph. etches glass	Compact masses, disseminated grains and crystals of great complexity.
...s with ...	.....	.....	Whitens on heating	Transparent to translucent granular masses or glassy grains, or sand.
...luble	.....	A little water Becomes opaque	Slowly soluble in caustic alkali	Translucent veins or lining with internal color reflections, or without "opalescence" and with waxy lustre, and shell-like fracture. Also dull like pumice and like drops of melted glass.
...uble	.....	.....	In S. Ph. R. F. gives violet	Crystals with brilliant lustre often parallel or netted. More rarely massive.
...luble	.....	.....	.....	Translucent crusts and cavity linings with smooth rounded surfaces, often in concentric layers with wax-like lustre. Never in crystals.

Table VII (*Continued*).—Minerals of Non-Metallic Lustre, T

A THIN FRAGMENT HEATED IN THE PLATINUM FORCEPS AT THE TIP OF THE BLUE FLAME		Crystal system: name, composition, hardness and specific gravity	The color of the mineral is:							
			Purple or violet	Blue	Green	Yellow	Brown	Red	Gray	Colorless or white
6. IS INFUSIBLE AND IS *NOT* MADE DEEP BLUE BY IGNITION WITH COBALT SOLUTION	H.	Quartz..... SiO_2 H=7 G=2.6	+	+	+	+	+	+	+	+
	I.	Garnet (Ouvarovite)..... $Ca_3Cr_2(SiO_4)_3$ H=7 G=3.1 to 4.3	....	....	+	....	....	....	.....	....
	T.	Zircon..... $ZrSiO_4$ H=7.5 G=4.7	....	....	+	+	+	+	+	+
	O.	Staurolite..... $Fe(AlO)_4(AlOH)(SiO_4)$ H=7.5 G=3.6 to 3.7	....	....	....	....	+	....	.....	....
	H.	Tourmaline..... $R_{12}B_2(SiO_3)_4$ H=7 to 7.5 G=3 to 3.2	....	+	+	....	....	+	.....	+
	I.	Diamond..... C H=10 G=3.5	....	....	....	+	+	+	.....	+

...ut With White Streak and Yielding No Tests With Sodic Carbonate

...bility	Flame coloration	Heated in closed tube	Other tests	Appearance
...ible	.....	.....	.....	Glassy hexagonal crystals and glassy shapeless material between crystals of other minerals. Also nearly opaque material, containing much iron and alumina.
...ble	.....	.....	.....	Crystals only.
...ble	.....	.....	Glows intensely on heating	Sharp-cut square prisms, long or short, usually imbedded in the associated mineral. Lustre usually adamantine or greasy. Also rounded pebbles.
...ble	.....	A little water	.....	Prisms often twinned, or in threes, crossing at 90° and 120°. Surfaces bright if unaltered.
...ble	Green with $KHSO_4$ CaF_2	.....	.....	Glassy hexagonal prisms differently faced at the two ends. Cross-section often suggests a triangle.
...ble	.....	.....	In powder is burned to CO_2	Crystals often rounded with lustre suggesting oiled glass, and cleavage in four directions at 70° 31′.

Table VIII.—Mineral Substances not easily Determinable by a Sche

The following mineral substances of economic importance have not been inc the determinative tables, some because they lack fixed characters, others beca characters are lost in those of their associated substances and others because th only in one known locality.

Amber, once the most prized of gems, now used sometimes in jewelry, oft mouthpiece for pipes, is a name given to those fossil resins which contain succ and were derived from a particular extinct species of pine. The amber of th Sea and the Sicilian amber are the most valued. Color, garnet red, reddish brownish, sometimes with bluish fluorescence. Lustre resinous, streak white, H 2.5. G = 1.096. Melts quietly at 125° to 150° C. and gives off a choking vap

Asphalts are rather indefinite mixtures of hydrocarbons and their oxidized p They vary from thick, highly viscous liquids to solids, are generally black in co pitch-like lustre, and burn easily with a pitchy odor. They are slightly heav water. Examples: the pitch lakes of Trinidad and of Bermudez, Venezuela; the of Barbados; the elastic elaterite of Derbyshire, England; the albertite of New wick, and the gilsonite of Utah. Sandstones and limestones impregnated with occur in many localities.

Carnotite, $2\ UO_3V_2O_5K_2O\ 3\ H_2O$ (?). A canary yellow, pulverulent mineral, in scales, filling the interstices of sandstone in several counties in Colorado. Rar pact and wax-like. It contains radium, and is an impure vanadate of uranium tassium, or uranium and lime, or both. Is a commercial source of radium, urani vanadium.

Clays are mixtures of mineral fragments, due to rock decay. They are usuall when wet, can be molded, and harden on heating. By analysis they are principal and alumina, with some iron oxide and small amounts of other elements. Mi cally they contain hydrous silicates of alumina, free quartz, and varying amounts other minerals. In origin they may have resulted from decay in place (residu or may have been transported by water, ice, or wind (sedimentary clays). T important clays are:

Kaolins. White-burning, residual clays, often not plastic, approaching kaol composition, but not necessarily composed chiefly of that mineral. They are t of white wares and porcelain, etc.

Ball clays. White-burning sedimentary clays. They are highly plastic added to kaolin to give plasticity.

Fire clays. Either sedimentary or residual clays, which stand high degrees without fusion. Composition very variable and apparently best with little fre lime, magnesia, or alkalis.

Stoneware clays. Clays sufficiently plastic and tough to be turned on a potter'

Terra-cotta clays. Usually buff-burning clays, with low shrinkage and dense character.

Sewer pipe and paving-brick clays. Vitrifiable, high in fluxes.

Brick clays. Low-grade clays, with considerable plasticity, which harden at paratively low temperature.

Slip clays. Melt at a comparatively low temperature and form a glaze.

Paper clays. White clays free from sand; used for mixing with pulp fibre.

Kieserite ($MgSO_4 + H_2O$) is the source of Epsom salts, and an important so magnesium oxide and basic carbonate (magnesia alba). It occurs at Stassfurt, I as about one-fifth of a layer 190 ft thick, chiefly halite and carnallite, and as one constituents of the overlying mixed salts. Exposed to the air it becomes ep After removal of associates there remains a mass slowly soluble in water and easily H = 3 to 3.5, G = 2.5. Rarely orthorhombic crystals.

Livingstonite ($HgSb_2S_4$). Found in Mexico at Huitzuco and Guadalcazar and said to be used as a source of mercury. It resembles stibnite in appearance, has metallic lead-gray color, red streak, H = 2, G = 4.81, and occurs in groups of slender prismatic crystals.

Mottramite ($(CuPb)_2V_2O_8 \cdot 2H_2O$). The vanadium of commerce was formerly obtained from this, blackish incrustations of mottramite upon the Keuper sandstone, Cheshire, England. Streak yellow, H = 3, G = 5.9.

Ochre, commercially, is a golden-yellow intimate mixture of clay with 20% or more of hydrated ferric oxide. Mineralogists use the name also for pulverulent yellow iron oxide (xanthosiderite) and for pulverulent red hematite.

Ozocerite, or mineral wax, is essentially a paraffine, colorless to white when pure, but usually greenish or brown, and possessing all the properties of beeswax except its stickiness. A little is mined in Utah and about 3,000 tons are imported annually from Galicia and Moravia. Used in crude state as insulation for electric wires. By distilling it the paraffine, used for candles, burning oils, paraffine, a product like vaseline and a heavy oil which, with india-rubber, constitutes the insulating material called okonite.

Patronite (vanadium sulphide). At the one locality of Cerro de Pasco, Peru, there is a vein 7 or 8 ft. thick of a nearly black material resembling slaty coal. About two-thirds of this is patronite and one-third metallic sulphides and free sulphur. Below this is 1 ft of coke-like material, chiefly carbon, which blends into a lustrous black asphaltite, 4 to 6 ft thick, containing more sulphur than carbon, but known as asphaltite. Specimens of these two associates are also rich in vanadium, and the roasted or burned ore is exported.

Petroleum is a mixture of hydrocarbons, obtained from the earth. It varies from a readily flowing liquid, to a thick viscous oil, and is usually of a dark brown or greenish color with a distinct fluorescence. Chemically the American petroleum consists largely of hydrocarbons of the paraffine series C_nH_{2n+2}, with smaller amounts of the series C_nH_{2n}, and C_nH_{2n-6}. The oils from Baku, on the Caspian, Rangoon, Galicia, and Java, contain more of the C_nH_{2n} or olefin series.

Roscoelite (vanadium mica). A mica of brown to brownish-green color, long known as an associate of gold in certain mines of California, and containing approximately 20% V_2O_3. It is now commercially obtained from a soft Colorado sandstone of greenish color, in which the roscoelite fills the interstices between the grains.

Thorianite ($ThO_2U_3O_8$). Small water-worn blackish cubic crystals found in the monazite gravels and used as a source of thoria. H = 5.5 to 6, G = 9.3. It is radioactive.

Thorite ($ThSiO_4$). Black or orange-yellow, zircon-like crystals and masses, occurring rarely in small quantity; used as a source of thoria. H = 4.5 to 5, G = 4.8 to 5.2. Decomposed or gelatinized with acids.

Umber is a drab-colored mixture of iron and alumnum silicates, containing manganese which becomes reddish brown on burning. Sienna is similar, but with less manganese and lighter in color.

Wad. Earthy to compact indefinite mixtures of oxides, especially of manganese, sometimes of copper, are known as wad. They have no constant characters, but may be determined as ores. Usually dark brown to black in color.

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SECTION 2

GEOLOGY AND MINERAL DEPOSITS

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GEOLOGY

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GEOLOGY

1. Introduction

A rock is a mineral or aggregate of minerals, forming an essential the earth; but many important mineral bodies, such as ores of metals, to be considered as rocks. Of about 1 000 species of minerals, only 2 are important as rock constituents.

The three great classes of rocks are: IGNEOUS, solidified from fusion; MENTARY, deposited in water or air; METAMORPHIC, recrystallized or otl altered igneous and sedimentary rocks, such that their original charac been obscured. Igneous rocks are believed to have been the predecesso source of all others (1, 2, 3).

An analysis, illustrating GROSS COMPOSITION of the outer 10 miles of the earth, in Sec I, Art 1. Compared with the percentages there stated, nickel and iron p become increasingly abundant toward the earth's center.

Most abundant elements of rock-forming minerals are: silicon, oxygen, alu iron, magnesium, calcium, sodium, potassium, and hydrogen; secondarily, chlorine, phosphorus, titanium, manganese, and sulphur. All other element the familiar copper, lead and zinc, and the precious metals, or an abundant atm gas, as nitrogen, are comparatively unimportant.

2. Chemical Composition of Rock-forming Minerals

Rock-forming minerals comprise silicates, oxides, carbonates, sul chlorides, phosphates, sulphides, and native elements.

Silicates are the most important, whence silicic acid, in various forms foremost acid in Nature. Three principal forms of silicic acid are repre in the rock-making minerals: H_2SiO_3 (metasilicic), H_4SiO_4 (orthosilicic $H_6Si_2O_8$. Pyroxenes, amphiboles, and leucite are salts of metasilicic Micas, olivine, anorthite, nephelite, garnet, and many minor miner orthosilicates. Orthoclase and albite are salts of $H_6Si_2O_8$. Some silicate only the usual bases, aluminum, iron, magnesium, calcium, and the al and are called ANHYDROUS; others, usually formed by weathering or alte of the first, contain hydrogen and oxygen in such proportions as to be off as water, and are called HYDRATED SILICATES. This distinction is re important by the general secondary character of hydrated silicates. Th anhydrous silicates in igneous rocks embrace the following mineral g feldspars and feldspathoids, pyroxenes, amphiboles, micas, and olivine. and less important are: zircon, titanite, tourmaline, and analcite. On we ing or other alteration, the hydrated silicates, kaolinite, chlorite, and serpe usually result. Metamorphic rocks contain a few characteristic silicate sides the common ones of igneous rocks, viz: staurolite, sillimanite, cy andalusite, scapolite, and epidote.

Oxides are next important, of which quartz (SiO_2) stands first, being dant in all the great classes of rocks. The related form of silica, chalce and the hydrated variety, opal, should also be noted. Next are the oxi iron, magnetite and hematite, and the hydrated form, limonite. With ma ite are associated chromite and ilmenite ($FeO \cdot TiO_2$). Water, whether or ice, is also technically a mineral.

Carbonates are calcite, dolomite, and siderite, with their intermediate tures. They are of chief importance in sedimentary and metamorphic occurring rarely in the igneous, except as products of weathering. The two SULPHATES, anhydrite and gypsum. One CHLORIDE, common salt,

...xtention. There is one PHOSPHATE, apatite, which, strictly speaking, is ...sphate and chloride or fluoride. Two SULPHIDES, pyrite and pyrrho... widely distributed. The one NATIVE rock-forming element is graphite.

3. Rock-forming Minerals (1, 2, 3)

Minerals of the igneous rocks are grouped according to their usual order ...stallization into: 1. Iron ores and minute associates. 2. Ferromag... silicates (olivine, pyroxenes, amphiboles, and micas). 3. Feldspars ...eldspathoids (plagioclase, orthoclase, nephelite, leucite, and analcite). ...artz, in acidic and higher medium rocks only. For descriptions, see ... Tables I–VIII.

Minerals of the sedimentary rocks are principally fragments of minerals ... igneous rocks. Quartz is most resistant to solution, alteration, and abra... and therefore appears in almost all sands and sandstones. The others ... frequent. After quartz, carbonates are of chief interest. Calcite and ... constitute the limestones, sometimes with slight admixture of siderite. ...ite, from feldspars, enters the fine sediments. The two sulphates, ... the more abundant, and anhydrite, appear only in sedimentary rocks. ...me is true of the chloride, rock salt.

Minerals of the metamorphic rocks. The components of both sedimentary ...eous rocks, when deeply buried, with attendant heat and pressure, re... at times to distinctively metamorphic minerals. Silica, being om... survives as quartz. The aluminous components afford andalusite, ...ite, and cyanite. Magnesian, iron, and aluminous compounds yield ...ast biotite and occasional epidote. Lime, in association with ferric iron ... alumina, makes garnet possible, but orthoclase may become muscovite. The ...es are important components. The ferromagnesian minerals (chlorite ... serpentine) are derived from magnesium- and iron-bearing originals.

Summary of Rock-forming Minerals

Igneous Rocks. QUARTZ
FELDSPARS: orthoclase, plagioclase
FELDSPATHOIDS: nephelite, leucite, analcite, melilite
PYROXENES: hypersthene, diopside, augite, soda-pyroxenes
AMPHIBOLES: hornblende, soda-amphiboles
MICAS: biotite, muscovite
OTHER MINERALS: olivine, magnetite, ilmenite, apatite, zircon

Sedimentary Rocks. Fragments from igneous rocks, especially quartz and feld...spars; kaolinite, calcite, dolomite, siderite, limonite

Metamorphic Rocks. Quartz, feldspars, biotite, muscovite, hornblende, epidote, ...garnet, sillimanite, andalusite, calcite, dolomite, serpentine, talc, chlorite

4. Igneous Rocks

Structures and textures. In a broad way, igneous rocks, as contrasted ... sedimentary and metamorphic, have a massive structure; that is, their ...s are not arranged in parallel or distinct layers. Massive is some... used as a synonym of igneous. Examined more in detail, as in hand-...s, they have 4 common textures. Where the molten mass has been ...dly chilled to crystallize, the texture is GLASSY. This texture appears ... borders of thin masses, on upper surfaces of lava flows, and, in rela...tively basible varieties, it may extend through an entire flow. It is most ...nt in siliceous rocks, which have high fusing points; it is rare in the ...ic and scarcely known in the basic. Where molten masses have cooled ...ewhat rapidly, and yet not so quickly as to prevent crystallization, very fine-

grained textures result, called FELSITIC. But, if older, larger, and **alrea** formed crystals at the time be swimming in the magma, which then **cry** in relatively small components, the texture is called PORPHYRITIC. **T** crystals are called PHENOCRYSTS and the matrix the GROUND-MASS. crysts of acidic rocks are chiefly quartz and feldspars; the dark ferroma silicates are much less common. In medium rocks, quartz practically **fa** feldspars are associated with more of the ferromagnesian minerals. **I** porphyritic rocks, feldspars decline, while augite and olivine, and **very** biotite and hornblende, gradually replace them. When a molten **magm** tallizes into an aggregate of fairly coarse components of about the **same s** texture is GRANITOID (like granite). Rarely, in these coarsely **crystallin** the feldspars become unusually large and stand out in contrast **with the**

As a result of explosive outbreaks at volcanic vents, igneous rocks are **sometime** out as fragments of all sizes, from impalpable dust to large bombs. **The fra** settle down on the sides of the cone or at greater distances, and yield rocks **with** fragmental texture, allied to sediments. If coarse they are called BRECCIAS; **if fine**

Chemical composition of igneous rocks. SILICA ranges from **abou** to a theoretical minimum of 0% in certain igneous iron ores; **only i** cases does it fall below 40%. Igneous rocks containing above 65% **sili** called ACIDIC; those with 55 to 65%, MEDIUM; below 50%, BASIC. Of **AL** the superior limit is 25 to 30%; general range, 12 to 18%; minimum, **ne** IRON OXIDES are low, 1% or less, in the most acidic rocks, but **increase** basic to 10 to 20%; in rare extremes, 90 to 95%. MAGNESIA **sinks to a** trace in the acidic, rising with fall of silica to 30% in the extremely **basic.** is low in the acidic, gradually increasing to about 15% maximum **in c** basic rocks. POTASH is highest in the rare leucite rocks, reaching 10 or it ranges from 4 to 7% in medium rocks with much orthoclase, and **disap** in basic types. SODA has a similar maximum in the rare nephelite **rocks** the same range in medium rocks rich in albite, approaching extinction **i** extremely basic. WATER, of 0.5 or 1%, usually indicates weathered **rocks**

It is important to connect chemical compositions with the **resultant mineral** vice versa. Chemical composition obviously determines the minerals, and, in so extremely acidic rocks have relatively high fusing points and chill more **easily, it al** fluences texture.

Classification of igneous rocks shown in Table 1 has general **accept** by geologists. Rocks range from acidic on left of table to basic on **right;** quickly-chilled rocks above to slowly-cooled rocks below, a still **lower lin** fragmentals marking transition to sediments. The forms assumed in **Na** are in extreme left-hand column; to be defined after the **descriptions.** rocks are further subdivided in vertical columns on basis of **mineralogy.** I spars and feldspathoids are the fundamental basis of subdivision; **other mi** als are subordinate.

The table gives a general view of the igneous rocks and defines those commonly in mining. For close determination greater refinement may be **desirable.** In s mining districts in the western U S are found the GRANO-DIORITES (intermediate betw granites and quartz-diorites), not mentioned in table. They have about the **same amo** of orthoclase and plagioclase. Intermediate between syenites and diorites are the M ZONITES. If they have a little quartz, but not as much as grano-diorites, they are ter quartz-monzonites. Butte granite, containing the copper veins, is usually describe quartz-monzonite. Several great bodies of "porphyry coppers" are in monzon porphyries. The varieties of gabbro containing hypersthene instead of common au are called NORITE; important because they contain the nickel-copper ores at Sudbu Ont. (For meaning of names of other rare igneous rocks, sometimes appearing in porta, see glossary in later editions of Kemp's "Handbook of Rocks.")

Crust Surface Flows	Glass	Acid Glasses, Obsidian, Pearlite, Pumice, Pitchstone			Andesite-Obsidian		Basic Glasses, Scoriae, Tachylyte, Basalt-Obsidian						Ultra-basic rocks
		Chief Feldspar, Orthoclase			Chief Feldspar, Plagioclase				Nephelite, Leucite		No Feldspar		
		Biotite (or) (and) Hornblende (or) (and) Augite			Biotite (or) (and) Hornblende		Pyroxenes		Pyroxenes		Augite (or) (and) Hornblende (or) (and) Biotite		
		+Quartz	−Quartz	Nephelite or Leucite	+Quartz	−Quartz	−Olivine	+Olivine	−Olivine	+Olivine	−Olivine	+Olivine	
									Basalt Group				
Surface Flows	Cellular, Glassy or Felsitic. Phenocrysts few	Rhyolite (Felsite)	Trachyte (Felsite)	Phonolite (rare), Leucite rocks (very rare)	Dacite (Felsite)	Andesite (Felsite)	Augite-andesite	Basalt	A series of rare basaltic rocks with nephelite (or) (and) leucite, seldom melilite or analcite. Not readily distinguished from basalt without the microscope. Extremely rare in America		Augitite	Limburgite	Basic segregations in normal magmas
Dikes, Intrusive Sheets, Laccoliths	Porphyritic. Phenocrysts prominent	Rhyolite-porphyry (Quartz-porphyry)	Trachyte-porphyry	Phonolite-porphyry	Dacite-porphyry	Andesite-porphyry	Augite-andesite-porphyry	Basalt-porphyry			Augitite-porphyry	Limburgite porphyry	In normal magmas
	Porphyritic. Phenocrysts predominate	Granite-porphyry	Syenite-porphyry	Nephelite-syenite-porphyry	Quartz-diorite-porphyry	Diorite-porphyry	Gabbro-porphyry	Olivine-gabbro-porphyry			Pyroxenite-porphyry	Peridotite-porphyry	Meteorites
									Gabbro Group				Water
Laccoliths, Batholiths	Granitoid	Granite	Syenite	Nephelite-syenite	Quartz-diorite	Diorite	Gabbro	Olivine-gabbro			Pyroxenite	Peridotite	Ice
...ata	...ag-...en-...al	Rhyolite tuffs and	Trachyte tuffs and	Phonolite tuffs and	Dacite tuffs and	Andesite tuffs and	Andesite tuffs and	Basaltic tuffs and breccias					

Glassy rocks are the most evident results of cooling from fusion. T almost always acidic and are represented by the rhyolites and daci scribed later. More basic varieties are known, but are less frequent. monest glasses are the OBSIDIANS, black, red, and brown, with 0.5 to 1% They may be assumed to be quickly-chilled rhyolites or dacites. PUMIC excessively cellular obsidian. A rarer glass, which chills into an aggre shot-like spheroids, is PEARLITE or PEARL-STONE, usually containing 2 water. The last glass deserving mention is the rare, resinous variety PITCHSTONE, having 5 to 10% water and is more easily fusible with bl than the others.

Rhyolite-granite series embraces igneous magmas containing: silici 80%; alumina, 12 to 15%; iron oxides, 1 to 3%; magnesia, less tha lime, 1 to 2%; potash and soda, 5 to 8%. They are common in Natu on crystallizing yield finely to coarsely crystalline rocks, consisting ch orthoclase, acidic plagioclase, and quartz, together with relatively small a of the dark silicates, biotite, hornblende, and augite, stated in order quency. Light-colored minerals are in great excess.

Rocks of this series are: RHYOLITE (syn, liparite), felsitic or partly glassy text phenocrysts; RHYOLITE-PORPHYRY (syn, quarts-porphyry), felsitic ground-mass dant phenocrysts; GRANITE-PORPHYRY, predominant phenocrysts, subordinate mass; GRANITE, granitoid texture, components of about the same size, but f sometimes abnormally large. PEGMATITES: crystallization of granite is often panied by separation of portions of the magma, in association with abnormall admixtures of dissolved gases. These portions pass outward into wall-rocks a often for great distances and in large size; on crystallizing, they yield very co gregates of same minerals as appear in granite itself, with many rare elements trated in them, and are called pegmatite.

In this series the prominent minerals are feldspars and quartz; dark silicates a ordinate. They are closely related to the dacite-quarts-diorite series, from which tinguish them microscopic examination may be necessary. The distinction is pra of small moment. This series is very abundant and widely distributed. Its tu breccias are also frequent.

Trachyte-syenite series embraces igneous magmas containing: silica, [illegible]; alumina, 15 to 20%; iron oxides, 1 to 5%; magnesia, 1 to 2%; lim [illegible]; potash and soda, 7 to 12%. They are much less common than the rh granite series. On crystallizing they yield finely to coarsely crystalline rock consisting of orthoclase, acidic plagioclase, and usually notable proportions dark silicates, biotite, hornblende, and augite, one or several. Quartz fai at most, is extremely subordinate. Light-colored minerals are in excess.

Rocks of this series are: TRACHYTE, felsitic texture, few phenocrysts; TRAC [illegible] phenocrysts; SYENITE, granitoid texture, components [illegible] ground-mass; SY [illegible] sometimes abnormally large feldspars. Syenitic-pegm [illegible]

In this series [illegible] dark silicates, subordinate. Li [illegible] granites. Tuffs and breccias are k

Phonolite-nepheline-syenite series embraces igneous magmas contai [illegible] magnesia, 1 to [illegible]. They occur infrequently. [illegible] consisting of o [illegible] sodalite, and pyro [illegible]

Rocks of this series are [illegible] with few phenoc [illegible] phenocrysts, of which orth

...dalite being usually confined to ground-mass; NEPHELITE-SYENITE-PORPHYRY, ... phenocrysts, subordinate ground-mass; NEPHELITE-SYENITE, granitoid ... rather fine to extremely coarse varieties, shading into pegmatites.

... these rocks, orthoclase is most prominent, but in the last-named, nephelite ... abundant. Many varieties have been recognized, depending on entrance of ... less usual than those named, and the decline of normal components. Sodalite ... very prominent. Biotite and hornblende are not so frequent as pyroxene. ... with leucite are known, but are far less common than those with nephelite.

...ite-quartz-diorite series embraces igneous magmas containing: silica, ... alumina, 12 to 15%; iron oxides, 1 to 3%; magnesia, 1 to 3%; ... to 4%; soda and potash (soda in excess), 4 to 7%. They are common ... centers. On crystallizing they yield finely to coarsely crystalline ... consisting of plagioclase, less orthoclase, and quartz, as the most promi... minerals, with biotite, hornblende, and pyroxene, one or several. The ... colored minerals are in excess.

... of this series are: DACITE, felsitic or partly glassy textures, few phenocrysts; ... PORPHYRY, felsitic ground-mass, with abundant phenocrysts; QUARTZ-DIORITE ... ..., predominant phenocrysts, subordinate ground-mass; QUARTZ-DIORITE, granit... ...

...desite-diorite series embraces igneous magmas containing: silica, ... to 60%; alumina, 15 to 18%; iron oxides, 4 to 9%; magnesia, 2 to 7%; ... to 8%; soda, 3 to 5%; potash, 2 to 3%. They are very widespread. ...stallizing, they yield finely to coarsely crystalline rocks, consisting of ... ...clase, a little orthoclase, and biotite, hornblende, or augite, one or several. ...-colored minerals are in excess and are the chief phenocrysts. The ... have usually light gray colors.

... of this series are: ANDESITE (varieties, mica-andesite, hornblende-andesite, ...-andesite), felsitic textures, few phenocrysts; ANDESITE-PORPHYRY, felsitic ground... with abundant phenocrysts; DIORITE-PORPHYRY, predominant phenocrysts, ground... subordinate; DIORITE, granitoid texture. Andesites are important in many western ... districts; in the recently extinct and active volcanoes along Pacific coast, in ..., and in other parts of world.

Basalt-gabbro series embraces igneous magmas containing: silica, 40 to ... alumina, 16 to 20%; iron oxides, 6 to 15%; magnesia, 5 to 10%; lime, ... to 12%; soda, 2 to 4%; potash, 1 to 2%. They are very widespread. On ... stallizing they yield finely to coarsely crystalline heavy rocks, consisting of ...oclase, little or no orthoclase, and large proportions of pyroxene, olivine, ... magnetite. The dark silicates are in excess and give rocks dark gray or ... colors.

Rocks of this series are: BASALT, felsitic textures, few phenocrysts; BASALT-PORPHYRY, ... ground-mass, abundant phenocrysts; GABBRO-PORPHYRY, predominant pheno... subordinate ground-mass; DIABASE, granitoid texture, feldspars long rectangular, ... irregular and occupying spaces among the well-crystallized feldspars; GABBRO, ... components as broad as long.

...-gabbro rocks are very abundant. Phenocrysts are almost entirely olivine and ... The peculiar texture of diabase, due to feldspars completing their crystalliza... before the pyroxenes, contrary to rule, gives them a special place. Varieties of ... afford special varieties of both basalts and gabbros. Hornblende and biotite ... observed; nephelite, leucite, analcite, and melilite sometimes appear and may ... the plagioclase. All feldspars and feldspathoids may fail, giving the rare ba... ...ingite and augitite, and the rare gabbros, peridotite and pyroxenite (Table 1).

Ultra-basic rocks. There are a few rare igneous rocks with less than 40% silica and ... high bases. The most important are the igneous magnetites, often ... ; in some places they are independent dikes and sheets, in others, segre... ... in igneous rocks.

Igneous	Eruptive	Limestones	Regional: Crystalline Schists corresponding in mineralogy to the Granitoid Igneous Rocks, & how others Derived from Igneous	Regional: Crystalline Schists Derived from Sedimentary		Regional: Crystalline Schists	Regional: Quartzites and Slates	Regional: Crystalline Limestones	Regional: Ophicalcites, Serpentines, Soapstones	Products of Weathering
Granites	[illegible]		Granite gneiss or gneiss proper	Granitic gneiss or gneiss proper	Mica	Mica schists often with garnet, staurolite, etc Phyllites	Quartzites (Quartz schists)	Marbles often with mica, tremolite, etc Dolomites often with mica, tremolite, etc	Serpentinous marble or ophicalcite, verd-antique Serpentine often with chromite, garnet, etc Soapstone	Residual soils Subsoil Saprolites in general Laterites, especially from basalt
Gneisses	Andesite bombs Lava-crete bombs Spotted slates		Syenite gneiss	Syenitic gneiss						
Porphyries	Garnet, diopside, vesuvianite, epidote, wollastonite, scapolite, ores		Diorite gneiss	Dioritic gneiss, etc	Hornblende	Hornblende schists often with garnet, epidote, scapolite Amphibolites	Gray-wackes Slates (Phyllites)			
	Natural coke		Gabbro gneiss Pyroxenite gneiss Peridotite	Conglomerate gneiss Gneisses with a prominent mineral, such as biotite-	Various schists	Chlorite schist or green schists Talc schists Eclogite Glaucophane schists				

...up of the original rock, partly to alteration and disintegration by removal of ... constituents. Quartz and aluminous hydrated silicates, with ferric hydrate, be... relatively enriched, while the other oxides go off in solution. Soils and subsoils ... and sediments are afforded for making sedimentary rocks. General names for ... of weathered products are: SAPROLITE or rotten rock; LATERITE, residual ... Laterization is most pronounced in tropical climates.

...rmination of metamorphic rocks is rarely difficult. Definitions convey the ... characters. Dense hornfelses sometimes resemble felsites, and may require ... determination. Gneisses, with increasingly fine foliation, shade into mica... and mica-schists into slates, so that distinctions may be matters of judgment. ... rocks of this series give little difficulty.

7. Forms Assumed by Igneous Rocks

... field, igneous rocks are found in dikes, necks, bosses, stocks, surface ...sive sheets or sills, laccoliths, and batholiths. The size and shape ... bodies exercises an important influence on texture of the component ... small bodies chill quickly and are glassy or felsitic; large bodies cool ... and are porphyritic or granitoid. Designating one horizontal dimen... length (*L*), the horizontal dimension at right angles to *L* as breadth (*B*), ... vertical dimension as depth (*D*), a mathematical expression can be ... formulated for each type.

...ikes are long, narrow bodies of igneous rock, filling fissures in older rocks, ... it has entered in molten condition. In dikes *L* and *D* are great, *B* ... small; they vary from less than 1 in wide and a few yards long, to ... of a mile in width and many miles in length. They usually have steep ... often intimately associated with ore-bodies, and in one place or an... embrace all varieties of igneous rocks. They may mark the last out... of a series of eruptions in a particular district, and are then usually very ... at Cripple Creek, Colo.

...ikes may radiate from an igneous center for miles into the surrounding strata, as in ... Mts, Mont, or the Trinidad coal region, Colo. They may appear hundreds of ... other known igneous rocks, as in the coal measures of S W Pennsylvania. A ... from solidified molten rock, even though a magma be regarded as a solu... fully all of which crystallizes *in situ*, is contrasted with a vein, similar in shape ... to the walls, but which is deposited from solution, the solvent passing ... in the case of pegmatites, it is not clear whether the term dike or vein should ... they may be considered the result of aqueo-igneous processes of fusion.

...eck is the solidified mass of lava that remains in the throat of a volcano ... last outbreak. When first congealed, it connects the lava that has ... from the crater with the unerupted residue in depth, just as a human ... ects head and trunk. As seen in the field, it is usually a decapitated ... that it is exposed to view only after removal of the lava flow and much ... by erosion. *L* and *B* are small, *D* great.

...anoes, which yield both lavas and explosive products, the neck may be part ... and part breccia. Necks project in a rudely columnar manner from remnants ... crater and from débris furnished by their own disintegration.

...sses are roughly cylindrical masses of igneous rock, projecting above sur... wall rocks like the boss on an old-time circular shield. Coarse granite ... rite, because of its relative resistance to erosion, often projects from ... mica-schists or other softer rocks. Bosses differ from necks in ... due to volcanic activity. As in case of necks, however, *L* and *B* are ... small, *D* great.

...cks are large, roughly cylindrical masses of intrusive porphyritic or ... rock, in the midst of older walls. They do not necessarily stand in

relief, but otherwise are much the same as bosses; *L* and *B* are small w spect to *D*, although absolutely of rather large size.

The name "stock" is the German word for floor or story in a house, and was to masses of igneous rock of cylindrical shape, because certain granite bodies of r outline, containing disseminated cassiterite, were formerly mined in horizontal s floors. Finally the mass of rock itself came to be called a stock. For good i tions, see Telluride folio of U S Geol Surv.

Surface flows are produced when lava wells out from a vent, and flow surface in a relatively thin sheet; *L* and *B* are large, *D* small. In upp under portions are many cavities, caused by expanding gases; the middl is usually dense, and in a thick flow may be comparatively coarse-gr The cavities are flattened and rounded like an almond, whence, from the they are called AMYGDALOIDS.

The top of a flow may be a rough, slaggy scoria, even consisting of cakes of and broken crust. Where dissolved gases have all escaped before consolidatic while lava is yet molten, the final chilled surface may be comparatively smooth. flows may bury one another in succession, or be covered with later sediments. are distinguished from intrusive sheets, because their heat can at most affec underlying rocks, not those formed above them after cooling; whereas intrusive bake both walls. More than 100 successive surface flows of basalt have been cut h shafts in the Lake Superior copper district.

Intrusive sheets or sills are masses of igneous rock which have been between strata of older rocks, and have solidified parallel with them. *L* are great, *D* small. The shape resembles that of a surface flow, and w surface flow resting on sediments is buried under subsequent beds the re much the same. The heat of intrusive sheets, however, always affects the ments above and below them, and sometimes produces important contact

Intrusive sheets vary from a few feet thick, and of no great known extent, to sill as the Palisades of Hudson River, visible 50 miles, traceable by drill 25 miles its thickness reaches 600 ft, but is usually less. Intrusive sheets doubtless rise fro depths along fissures, like dikes, but then turn sidewise between strata along a least resistance. They are sometimes associated with ore-deposits, as at Lea Colo, and Mercur, Utah.

Laccoliths are a variation of the intrusive sheet and are lenticular in shape. sill be supposed to start from its feeding dike, sidewise between strata, and to easier to raise the overlying beds of a limited area than to force its way with u thickness far and wide, a lenticular mass will result, tapering from a central max thickness to a thin edge. Hence, *L* and *B* are relatively large, *D* smaller but va Laccoliths which are fed outwardly from a central supply fissure are symmetrical this fissure is sometimes a fault, with hard strata opposite soft ones, so that the sive can penetrate outwardly only on one side. Unsymmetrical masses, practically laccoliths, result. Laccoliths heave up overlying strata in domes, and when the eroded the laccolith is exposed in midst of outwardly dipping beds. The entra laccoliths may have been aided by incipient folding or arching of beds under co sion. Laccoliths are widespread in the western states. The name was coined by Gilbert from the Greek word for cistern, as the shape suggested the ancient dome-co vaults for storing water.

Chonoliths are irregular intrusive bodies, either filling a pre-existing cavity, or ing apart the rocks to make a way for itself. The name was coined by R. A. Daly the Greek word for a mould in which metal is cast. No definite expression in ter *L*, *B*, and *D* is possible.

Batholiths are huge masses of intrusive rock, of irregular shape and great ex *L*, *B*, and *D* are all great. Granite masses, square miles in area and sometimes miles in volume, are illustrations. They are specially abundant in pre-Cam strata.

1. Forms Assumed by Sedimentary and Metamorphic Rocks

distinguishing feature of sedimentary rocks is their arrangement in parallel during formation. Variations in deposition of sediment from high and storms and calms, floods and droughts, produce contrasts in coarseness ness. At the outset they are flat, except for the slight inclination of the en, and irregularities due to delta formation and swift currents. The in- sition often seen in exposures today is due to subsequent disturbances.

Stratification. The smallest division of a sedimentary rock is a LAYER or It may be a fraction of an inch thick and marks one period of es- abundant deposit. Layers go together to form BEDS, the natural sedimentary rocks. Bedding planes are recognizable and thick- and ed sedimentaries are distinguished. Beds combine to constitute a or tabular mass of one kind of sedimentary rock between others are different. A stratum may range from 1 to 1 000 ft thick. Thin are usually called SEAMS, as in case of coal. Limestones, shales, and s afford thick strata. In geological mapping, a thick and persistent is often called a FORMATION.

sedimentary rocks present all the features of the bottom, or of the strand between low water: as ripple marks, tracks, stranded shells, rill-marks, mud-cracks, ange from swift currents, irregular beddings, and cross-bedding in individual deltas. Since the greatest thickness of sediments gathers along subsiding with attendant advances of sea over land, there are found in normal suc- conglomerates which represent old shore shingle, followed by sandstones, repre- shore shallows; next shales, corresponding to deeper, quieter water; lastly, ating still deeper water, free of mechanical sediments, are limestones, con- ny of organic remains. This normal succession is not always found, since ren, etc. destroy uniformity, but it is not infrequent. There are also desert ons, wherein wind-blown particles are important, and are associated with temporary streams, lakes, and floods. Land accumulations are character- ed from oxidation of iron.

2. Rock Disturbances

wide observation has shown that the rocky outer portion of the earth subject to many disturbances. Great masses may rise or sink without the local attitude of the rocks. These continental movements are of interest, but seldom of importance to the engineer. Localized move- to elevation of a long and relatively narrow belt in a mountain chain, rbances incident to intrusive entrance of bodies of igneous rock, are portant. The results of these movements are termed FOLDS and FAULTS.

lds are bendings in strata, whereby each layer assumes a curved form, ing a portion of a cylinder. When classified in order from least to lds embrace MONOCLINES, ANTICLINES, INES of several types, DOMES and BASINS.

Monoclines (Fig 1) are terrace-like bendings of with inclination varying in amount, but same direction, as the name implies. A p of the terrace marks a belt of especial the strata affected, and may be accom- numerous cracks. At foot of the terrace d roll in reversed direction, with attendant nd cracks. In the upper roll, overlying beds are subject to tension. g to compression; in the lower roll, the upper beds are compressed, tense. Between these areas is necessarily a surface of no strain.

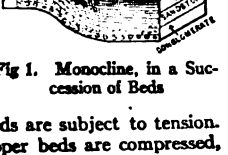


Fig 1. Monocline, in a Succession of Beds

Monoclines which involve porous beds, such as open-textured sandstones tight shales, are sometimes important places for accumulation and storage o gas and petroleum. The search for these is essentially an endeavor to locate, drill, favorable monoclines or gentle anticlines. Monoclines have been desc arrested anticlines. In a series of sediments comprising shales or other so monoclines or even more violent folds in stiffer strata may at depth disappea in the adjustment of soft overlying shales, the plastic movement of which take distributes the fold until it is diffused and lost. The name monocline (or m structure) is sometimes applied to a remaining half of an eroded anticline or the other half of which is not apparent; inclination of the beds is all in one dir

Anticline and syncline (Fig 2) are complementary terms; one ra pears without the other. An anticline is an arch-like bend, a synclin responding trough. The upper pa anticline is called the CREST; its sides, LIMBS or LEGS; the central running parallel with the axis of t centric partial cylinders, the surf which are represented by each fold is the AXIS. The bottom of a syn the trough. Beneath anticlinal cr synclinal trough the beds are es strained and cracked; the cracks to gape upward in the anticline and ward in the syncline. The limbs of each type of fold are less straine crest or trough. Thus, good building stone will be found on the limbs than at crest or in trough. On the contrary, veins and mineral deposi circulating waters find natural resting places in crest and trough.

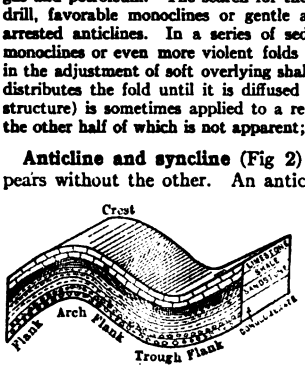


Fig 2. Anticline and Syncline, with Horizontal Axis

Anticlines and synclines, when followed for a mile or more, seldom have ho axes as shown in Fig 2. The axes usually pitch downward (Fig 3), though th afterward rise again. In Fig 2, the component beds, if eroded, would appear at surface in parallel bands. When pitching folds are eroded, the several beds appear at surface as concentric curves (Fig 3). In anticlines the upper or later beds are outside, the under or older, inside; in synclines the under or older beds are outside, the upper or later, inside. Because of these relations, geologic structure may sometimes be inferred from a colored geologic map.

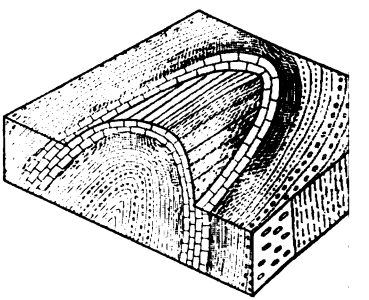


Fig 3. Pitching Anticline, Showing Concent Curving Outcrops of Eroded Beds

Anticlines received their name because the observer was assumed to stand at the crest, from which the beds inclined outwardly in opposite directions; hence the prefix "anti," for "opposed." Standing in the trough of the syncline the observer se inclined beds sloping toward him, hence the prefix "syn," for "together." Ant and synclines of which the inclination is the same on both sides of axis (see dia are called SYMMETRICAL. Symmetrical folds may vary from those of compara slight disturbance to tightly compressed folds. In the former, where the limbs of are separated by other beds, the fold is called OPEN; but where from extreme co sion the limbs of a single bed are brought tightly together, the fold is CLOSED. shows open symmetrical folds; Fig 4 closed symmetrical folds. A limiting case anticline, speaking mathematically, is the DOME, in which the beds pitch radially directions from a central point. The variable direction of the inclination has sug

...ne QUAQUAVERSAL. A dome is an anticline of which the axis is reduced to a point. ... are chiefly developed above laccoliths; seldom in other relations. A BASIN is a ... the axis of which is a point, toward which the strata converge.

... this strict sense are rare, and result from local removal of ... and collapse of strata. The term is also used in the geology ... of a synclinal arrangement of strata, wherein a rising pitch ... in opposite directions brings the measures to the surface. ... thus form concentric canoe-shaped or spoon-bowl syn-...

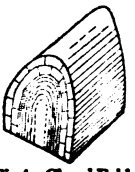


Fig 4. Closed Fold

...symmetrical folds. Strains which caused a fold may ... bed one limb under or over the other, thus producing ...metrical inclinations. From relatively slight differences, ...ture may increase until the overturned portion rests on an underlying ... Such folds may even be S-shaped (SIGMOID) or RECUMBENT. On a ... scale, these often occur in metamorphic districts; on a large scale they ... mainly in regions of violent disturbance.

... names may be used, such as the Jura type for symmetrical folds; Appalachian ... ...e those steeper on one side than the other (Fig 5). Closed folds are those of which the limbs are squeezed so tightly at one spot as to cause a great bulge of an upper or under core of rock. When the surrounding strata incline away radially from the compressed area, like ribs of a fan, the fold is called a fan-fold.

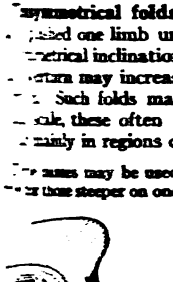


Fig 5. Overturned Fold ... Appalachian Type

Folds vary in size from small wrinkles and puckers, as in schists, to arcs having chords of yards, miles, or hundreds of miles. Folds are sometimes designated as of the first, second, third, or higher orders. A mountain range, consisting of an anticline or a syncline, is respectively called an ANTICLINORIUM or SYNCLINORIUM, the Greek word for mountain being added to type of fold. When great flat folds occupy an appreciable part of earth's surface, they are called respectively GEANTICLINES and GEOSYNCLINES, prefixing Greek word ...th to name of fold. Folds are of great importance in engineering work, not only in ... bedded minerals like coal, salt, and some iron ores, or in the discovery of petro- ... and gas, but also in connection with railway tunnels, aqueducts, and other engi- ...ing work.

Dip and strike. DIP is the angle of inclination of a vein or bed below hori- ... STRIKE (course or bearing) is the direction of line of intersection of an ... vein or bed with a horizontal plane.

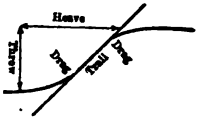
... dip angle is the angle between two perpendiculars, one in the inclined plane, the ... the horizontal, let fall from a common point on their line of intersection (the ... The strike is stated in degrees and minutes, E or W of N or S, for example ... E. Since plane of dip is at right angles to line of strike, it is recorded in de- ... E or W of strike; thus, a strike of N 25° E and dip of 50° N W signifies that the ... in which dip is measured runs 65° west of north. Some observers note exact ... and amount of dip, leaving strike to be inferred. Thus, a record of a sand- ... dipping 50° in a direction N 65° W, implies a strike of N 25° E. First mode ... is customary in America. A GEOLOGIST'S COMPASS has one flat side, and ... pendulum, swinging around a graduated semi-circle, so as to give direct dip read- ... In plotting, each observation of strike may be corrected for variation of needle, or, ... more elaborate compasses, the graduated circle may be turned to read directly ob- ... reduced to true north.

10. Faults

... ...lt is a dislocation in otherwise continuous strata or masses. It results ... rocks are so excessively strained that they yield along a crack or series of ... one side altering its position with respect to other. One side may rise, ... move laterally, or (as resultant of all 3 movements) diagonally, with

respect to other side. In nearly all cases the fault plane or planes are inc
horizontal, the upper and under sides being designated by the miner's
HANGING WALL and FOOT WALL. Limiting cases are vertical and horizonta

Classification of faults (9, 10). The commonest are: normal, reve
shift faults (Fig 6, 7 and 8). In NORMAL FAULT ("normal" here m



[Fig 6. Normal Fault, Displacing Flat Coal-Seam. Cross-section

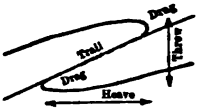


Fig. 7. Reverse Fault. Begun as a Overturned Fold. Cross-section

"usual" or "common") the hanging wall has slipped down with refere
foot wall. The movement is rarely directly down line of dip of fault plar
usually on a diagonal. The position of any point in dislocated portion
ferred to the 3 axes of solid geometry: the vertical component is the r
horizontal component perpendicular
strike of the fault plane is the HEAVI
horizontal component in fault plane
SHIFT. These mathematical factors as
determining direction and amount of
ment, the line of which is the diagonal
rectangular prism the edges of which a
heave, throw, and shift.

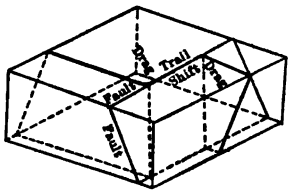


Fig 8. Shift Fault in Vein Dipping 50°. Same Effect would be Produced by Normal Fault, with Diagonal Displacement Involving Shift Component Away from Observer; or by a Large Throw, Straight Down the Dip

Fault-breccia. Movement of fault
or of one wall on the other, often cr
adjacent rock to a mass of angular
ments, mixed with more finely commi
material. Circulating waters may ce
the whole into a solid mass, by depo
new minerals, sometimes producing val
ore. This mass is a FAULT-BRECCIA. Fragments of any bed, dike, or
involved in the fault movement, will be dragged along from stationary si
direction of movement; or will be left behind by moving side; and if foll
along fault plane, will indicate direction of movement. Such fragments fu
valuable evidence and by F. T. Freeland have been aptly termed the TRA
THE FAULT. Should a vein be cut off by a fault, with attendant breccia,
ments of the vein should be sought in the breccia and the trail followe
pick up continuation of vein.

Drag. Faults often cut relatively soft beds, as shales or shaly sandsto
Friction of the walls upon each other causes a downward bend in the bec
stationary or lifted side and an upward bend in those of the moving or drop
side. These bends, called DRAG, show the direction of movement (Fig
Drag is not found in strong rocks, like granites or heavily-bedded limesto

Slickensides are polished and usually grooved surfaces, often caused
movement of walls of a fault or vein. Upon the wall-rock the grooves indi
direction of movement, but do not necessarily show which side has gone up
down, or laterally.

Some observers have thought that by scraping finger nail or finger across the groo
one side of them will be found steeper than the other. If the grooves are tested in sl
ensides on underside of plane of movement, such steep ridge is considered to be

side of groove, or the side which resisted bearing down of hanging wall while moving normally downward in fault plane. It will thus indicate the actual direction of movement. Should the steep ridge be on upper side of grooves, an upward movement of hanging wall is indicated. Others have thought that when the finger is moved along a groove the greater roughness is felt in the direction of movement of the part felt. Striæ on fragments in fault-breccia are of little significance, since they are not in place.

Horses are large disconnected masses of wall rock, involved in faults, or produced by forking of a fault fissure around a split-off fragment, and especially closely related to subsequent vein-formation along fault.

Shear-zones. When a fault movement is distributed along a number of closely spaced planes not widely spaced, the wall rocks are broken into parallel tabular masses, and are said to be SHEETED. The resulting fault is "distributed," and the sheeted strip is a SHEAR-ZONE. GOUGE is a sheet of clay, often occurring along the outer edge of fault breccias, especially those subsequently mineralized by circulating waters. Other names are: SELVAGE and FLUCAN.

Fault-scarp. If a fault involves an appreciable vertical component, the relatively lifted side may stand out as a terrace or escarpment, the fault-scarp. This soon wears it down, so that fresh fault-scarps are rarely recognizable. They have sometimes aided the deposition of ore bodies by furnishing waterfalls. When they are developed across an older mineral deposit, serious displacement may be caused.

Rules for solving faults have been formulated by Schmidt (11), Zimmermann (13), Freeland (7) and others. In studying a fault, observe trail, drag and slickensides. Stratigraphical succession, if known, will reveal amount of displacement. Bore-holes are useful. Models assist, and are sometimes equivalent to projections on paper. If there be no evidence to contrary, the assumption that fault is normal is justified, because most faults are such. Nevertheless, experience shows that a reverse fault occasionally appears in a series of normal faults, that shift faults may occur, and that fault movement may be oblique (normal at one extreme of fault plane, reverse at other). On entering a fault, a mathematical solution is attractive, but, despite many text-book discussions, the necessary data are seldom obtainable. Attention should be concentrated on the fault plane and the movement along it (12). The dislocated portion of a tabular body is to be sought, presenting a broad face if rightly attacked. As a rule, it is easier to drift horizontally, than to sink or raise; the procedure is largely determined by the way the vein or bed lies. Where a series of stratified rocks, the succession and thickness of which are known by previous mining operations, by study of the surface, or by borings. Beds on far side of fault is recognizable, and its place in the series known, direction and amount of movement may be determined. As gulches often follow faults, because of easy erosion of crushed rock, faults may sometimes be found more readily by study of surface exposures than by observation underground. Directions of slickensides, drag, and trail, commonly in faulted stratified rocks, are highly significant. If none of these evidences decisive in dealing with a mineral deposit cut by a fault, there is strong probability that the fault is normal. On this assumption, if a fault be encountered on its under side the rule is to cross it and sink; if on its upper side, to cross it and raise. This is expressed in the old rule: "follow the obtuse angle." But, if the fault happen to be reverse, the rule would lead in wrong direction.

In dealing with steeply dipping veins in massive rocks, or steeply dipping beds of stratified rocks containing coal seams or other interstratified deposits, the suc-

cession of strata must be known to determine the movement. Then, s tentatively as a normal fault, due weight must be given to throw and sh possible components of diagonal movement. That is, besides the heav throw of a normal fault, a large shift-component might cause displac

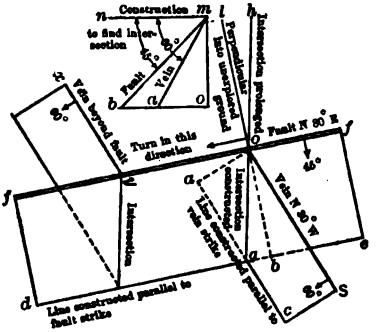


Fig 9. Zimmermann's Solution of a Fault (Projection on Horizontal Plane)

opposite to that anticipated, i of straight down the dip. I case, the occurrence of slicker trail, and drag may be essen correct solution.

Zimmermann's rule (13), fo faults, cutting steeply dipping Suppose (Fig 9), in driving a l vein *so* striking N 30° W and d 60° W, a fault *ff* is met, strik 80° E and dipping 45° S. At in tion *o*, draw *ob* perpendicular to of fault, and prolong it toward *l*, the fault. Project upon plane the intersection *og* of fault an Line *og* is horizontal, and passes ob through *o*, into unexplored grou ward *h*, on one side or other Then, if exploratory drift on f of fault be turned from *oh* tow and parallel to strike of fault, the displaced segment *xy* of vein will in most ca found.

The horizontal projection *og* is found as in small diagram of Fig 9. Draw hor line *mn*, lay off *ma* and *mb* to represent dips of vein and fault; draw *mo* perpen and *ob* parallel to *mn*. Then *oa* and *ob* are the distances by which, in descending a cal distance *mo*, the planes of vein and fault depart horizontally from vertical. I part of Fig 9, *oa* and *ob* are drawn respectively perpendicular to strikes of vein and fau *ac* and *de*, passing through *g*, are parallel to those strikes. Whence *g* lies in the horizontal projection of intersection of vein and fault.

In Fig 10 is shown a similar relation of vein and fault, except that the vein lies east and exploratory drift should turn east, as shown. Both solutions depend on assumption that the hanging wall of fault (*i e*, its south wall) has slipped with little shift down on its footwall. With a strongly diagonal movement the fault might still be normal, but the solution might lead miner in wrong direction. Therefore check all rules by trail, drag, slickensides, etc.

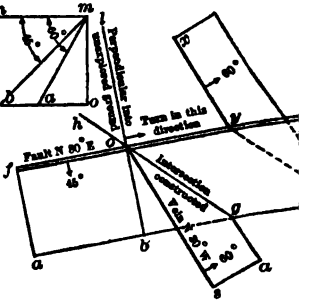


Fig 10. Zimmermann's Solution of a Construction as in Fig 9, but Vein I Whence Exploratory Drift Turns *E*

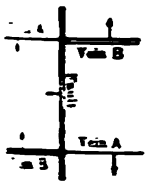
The following additional terms apply to faults. In tilted, stratified rocks, faults striking parallel with the strata are STRIKE-FAULTS, often resulting when folds pass into faults. Faults running across strike and parallel wit of dip are DIP-FAULTS. STEP-FAULTS are series of parallel faults, dipp same direction. The HADE is the angle made by a fault plane with a v plane; hence, hade is complement of dip, and is a superfluous term.

Various puzzling cases of faulting have become classic. Fig 11 shows a Cornis from de la Beche; Fig 12, a case of two contrasted pegmatites in Sweden, obser A. G. Högbom. Two parallel veins may be so faulted as to bring dislocated part opposite sundered end of another, and temporarily conceal the existence of a fau

... shift fault, cutting a vein at right angles to vein's strike, the amount of shift ...ved to grow gradually less in depth, leading to inference of a HINGE-FAULT, or ... ROTATIONAL FAULT.

...mal faults are often explained as due to tension strains in earth's crust, leading ... apart of the two sides of fault, and the slipping down of upper portion on ...face. they have been called tension or gravity faults.

...ults, by contrast, are called compression or THRUST ... they often begin as overturned folds. If these stresses do produce their respective faults, then reverse faults should customarily have low dips, since, on approaching the perpendicular, friction would increase prohibitively. But, if tensional stress were relieved by a series of parallel faults, and one fault block were to drop below its neighbors, there would be a normal fault on one side of dropped block and a reverse fault on other.



... 12 Two Veins, ...verging Dips, ... Below their In... by a Normal

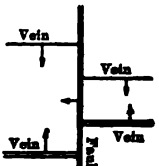


Fig 11. Two Veins, with Converging Dips, Normally Faulted

Compressive strains along the strike can easily develop normal faults by downward bulge of hanging wall and upward bulge of foot. Where comparatively short faults die out at each end, this explanation has weight. Again, assuming that in depth rocks are capable of viscous flow and transfer, pressure transmitted upward from such moving masses ... faults from stresses wholly different from any previously mentioned. Where ... inclined, fault blocks with the larger base would be relatively lifted, as com... those having smaller base. Foot-walls would therefore rise relatively, caus... normal faults (4, 5, 6, 8).

12. Joints, Unconformities, Outcrops, Erosion

Joints are cracks which cross strata and masses, without producing dislocation. Notwithstanding absence of dislocation, there may be difficulty ...inguishing between joints and distributed faults of slight displacement, ... produce sheeted structure.

... are due to easing of some kind of strain; as contractions in cooling of igne... expansion of cold rock under the sun's heat, shrinkage from drying of water-... sediments, tensional strains at crests of anticlines or in outer layers in bottoms ... and torsional stresses produced over wide areas by warping of earth's crust. ... rocks break into polygonal columns of greater or less regularity. In joints ... by contraction of igneous magmas during consolidation, the long axes of the ... are theoretically perpendicular to cooling surface. If the magma be homogeneous ... of heat uniform, regular hexagonal columns result, parted also across their ... cup-shaped joints. Occasionally, as at Giant's Causeway, theoretical perfection ... attained; usually, the columns are of all numbers of sides, from 3 to 8. Similar ... from drying.

... heat of sun and weathering cause massive rocks to shell off in thin layers. ... blocks produced by jointing may become rounded boulders. In granite quarries ... in great concentric curves, like a huge onion, has probably been caused ... contraction strains in cooling, or compression strains in earth's crust. Cracks ... upward at crests of anticlines, and gape downward in troughs of synclines, ... features of folds. In certain districts of flat sedimentary rocks (as in south... New York), joints in two series, intersecting each other at nearly right angles ... about 80°), run with remarkable regularity; probably due to torsional strains ... warping. In areas of massive or metamorphic rocks, while a principal series can ... other joints show no regularity. Observed strikes may be plotted over a wide ... lines intersecting at a common center like a clock-face, resulting in detection of ... strikes. The predominant joint is called a MASTER JOINT; the others, minor ... Joints are of great practical importance in quarrying and in mining.

16. Minerals and Localization of Ore Deposits

Ore minerals are primary and secondary. The PRIMARY are those ori deposited in forming an ore body; the SECONDARY are produced by alt of primary minerals under certain conditions. Except aluminum, iron, ganese, chromium, platinum and tin, all primary ore minerals are sul arsenides, sulpharsenides, sulphantimonides, or similar compounds. Sul are of chief importance. Though secondary minerals are largely ox compounds, they also comprise a few very important sulphides.

The importance of the distinction lies in the following relations to the surface. G waters stand at varying depths, depending on local rainfall, rock texture and loc logical structure. Between ground-water level and the surface, is a zone cal Posepny the vadose zone, by Van Hise the zone of weathering, through which t dizing and dissolving rain waters freely descend. Within this vertical range, sul become oxidized to sulphates, and pass extensively into solution. Migrating dow the solutions merge into the standing and protecting ground-waters, and often p tate their dissolved metals in a zone of SECONDARY ENRICHMENT, at or near g water level. The reaction is especially important in copper mines.

Gangue minerals comprise quartz, calcite, fluorite, barite, rhodochr rhodonite, and admixed minerals of the country rock. Decomposition or ation under influence of thermal waters gives rise to much sericite, kao and related species.

Localization of ore deposits. Ore deposits resulting from processes outlined are developed where circulating mineral-bearing solutions find favorable places t cipitate their contents. One method of classification is to arrange in a logical s the favorable geological places for this reaction. For the formation of some kinds deposit, however, circulating solutions are not required. Ores may crystallize d from molten magmas, and, either by sinking in the fluid mass because of higher s gravity, or for some reason not well understood, may enrich the rock mass to the re ments of mining. Again, and in contrast with the reactions above indicated, m waters in streams, or by wave action, may liberate and concentrate heavy miner sedimentary deposits, to the point of profitable mining. Again, in residual dep heavy and resistant minerals may be left behind in a concentrated condition by re of products of weathering. In a few cases, chiefly iron ores, the processes of sed tation, or associated precipitation, have given rise to bedded ore deposits. Expe shows, therefore, that it is difficult consistently to classify ore bodies on any one of subordinate principles. But, as compared with old-time schemes based on shap broad principle of MODE OF ORIGIN has become increasingly important. In the foll classification, the endeavor has been to pass from igneous phenomena, pure and si to surface reactions not connected with igneous phenomena, emphasizing especiall places where ore bodies originate.

17. Classification of Ore Deposits

I. Primary magmatic origin.

(a) Masses produced by crystallization and segregation in cooling and soli ing igneous magmas: titaniferous and non-titaniferous magnetites, chro nickeliferous platinum, and probably sulphides of iron, nickel, and copper. A dikes and sheets of igneous magnetite are known.

II. Deposited by emissions from cooling and solidifying magmas.

(a) Various of the nature of pegmatites, varying from the mineralogy of gra to [illegible] with tourmaline, fluorite and minerals contai [illegible] rare elements, and exception [illegible] between dikes and veins; sometimes [illegible]

[illegible] by emissions from an [illegible] such as: garnet, d

...ntonite, vesuvianite, epidote; also of magnetite, specular hematite, ... of iron, copper, and less often of other metals.

... **Deposited by circulating ground-waters.**

... Deposits in or along faults, with greater or less replacement and im... of the walls; often called "true fissure" veins. They vary from the ... an open and clean-cut fissure, to impregnation and replacement of ... faults of small individual displacement.

... Saddle-reefs: precipitations of quartz or other minerals, at crest and be... layers of an anticline, and extending with diminishing thickness for ... but usually moderate distances down its flanks. Inverted saddles ap... synclines. Apparently arching of the strata has aided precipitation.

... Deposits in joints with greater or less replacement and impregnation of the ... often called "gash veins," because limited to a single stratum or sheet.

... Impregnations of volcanic agglomerates in the conduits of extinct ex... volcanoes.

... Impregnations with greater or less replacement of permeable rocks, ... amygdaloids, volcanic tuffs and breccias, open-textured sandstones ... conglomerates, autoclastics, etc. Supply conduit may be obscure.

... Replacements of limestones, calcareous shales or other beds, which have ... to circulating ground-waters. Supply conduit may be obscure.

... **Deposited or concentrated by aid of surface waters.**

... Surface precipitations, which may later be involved in stratified series. ... ores.

... Resistant or insoluble minerals, concentrated as a residuum by weather... removal of the matrix. Residual deposits. Cuban brown hematites.

... Placers or concentrations of heavy minerals in sands and gravels by ... moving water. Gold placers. Stream tin.

... above definitions are self-explanatory, and will receive no further comment. ... detail will be found in textbooks (22).

18. Iron Ores

... **ores minerals are chiefly productive** of iron: limonite (brown hematite, ... $2\,Fe_2O_3 \cdot 3\,H_2O$, Fe 59.8%; siderite (carbonate, spathic ore) $FeCO_3$, ... hematite (red and specular) Fe_2O_3, Fe 70.0%; magnetite (magnetic ... Fe_3O_4, Fe 72.4%. Associated with limonite, but less common, are ... ferric hydrates, as: turgite ($2\,Fe_2O_3 \cdot H_2O$); goethite ($Fe_2O_3 \cdot H_2O$). ... silicates may also occur, as: chamoisite (hydrated ferrous aluminum ... thuringite (hydrated ferrous or ferric aluminum silicate); greenalite ... ferrous silicate). Siderite may have its iron partly replaced with ... and calcium. With brown hematites, manganese minerals are not ... Pyrite (FeS_2) may appear with all the ores, and, when largely ... sulphur, may yield a residue possible of utilization for the poorest ... iron. Pyrrhotite (Fe_7S_8) is frequent with magnetites. Ilmenite ... is mechanically mingled with many magnetites. The objection... elements of iron ores are sulphur and phosphorus. Definite limits of ... merchantable grades of ore are variable, because of possible admixtures ... practice. In general, the less the better. In ores used for acid ... e.g. the permissible phosphorus maximum is 0.001 part of the per... of iron. Roughly, therefore, about 0.06% is the maximum with ...

... *ores and specular hematites* are the richest iron ores. In America lump mag... have in the past exceeded 65% iron, but today practically only magnetically

concentrated ore reaches this figure. Kiruna, Swedish Lapland, however, ca a great tonnage of this grade, even to American furnaces. Average of all iron in the U S in 1914 was not far from 50% iron. The grade will doubtless gradual Alabama Clinton red hematites run 36 to 37%. Some crude ore is even lo principal local supply of continental Europe, from the minette ores, averages a The nearness of good fuels, markets, mixtures, etc, determine limiting percenta

In the U S about four-fifths of the ore comes from Lake Superior region. order are the red hematites of Alabama and Tennessee, the brown hematite magnetites of Appalachian belt. Individual mines in Wyoming, Colorado, Mexico have fed the iron and steel plant at Pueblo, Colo. Magnetites will be in time on the Pacific coast for a future industry to be located presumably in Sound region.

Lake Superior iron districts. In order of productiveness in t Superior region are Minnesota, Michigan, and Wisconsin. Ontario productive range, and possibility of developing others. The ores of th are all in pre-Cambrian strata, now classified as follows:

Keweenawan: sandstones, basalt flows. Copper.
Huronian, Upper, Middle and Lower: sedimentary and some igneous roc ores in the sediments.
Laurentian: granites.
Keewatin: green schists, from ancient basic eruptives. Some sediments with

As now mined the ores are chiefly soft, partially hydrated hematit percentage in water not reaching that of limonite. They have been p by alteration of great beds of cherty carbonates of iron, of hydrated silicate, and of associated pyrite, under the general processes of wea Soft earthy masses of ore have thus resulted; of enormous volume, ac cheaply mined, and of relative purity as regards phosphorus and They occupy synclinal basins, troughs produced by intersection of dikes with each other or with impervious strata, or other minor place circulating and oxidizing meteoric ground-waters have been tempora structed in their flow. Besides soft ores there are lenticular bodies specular hematite, produced by metamorphism of ancient soft-ore bodi great bodies of jaspery or siliceous iron-bearing strata, of 35% and a iron, which, partly as concentrating ore, partly as low-grade lump or the gradual exhaustion of better grades, will be available for a long come. The grade is well above that of present European ores. Th originally formed beds precipitated at surface (Art 17, *j*). They becam in a stratified series, and afterward by weathering yielded residual (Art 17, *h*), some of which extend to great depths and are cases of se enrichment. Some have been metamorphosed to specular hematite, a magnetite. In northeastern Minnesota gabbros occur with igneous t ous magnetites, not yet shown to be valuable (Art 17, *a*) (23).

Clinton red hematites are next in productiveness. They appear of oölitic, often fossiliferous ore, associated with olive-green shales a ordinate limestones of the Clinton stage, at base of Silurian system. outcrop in S E Wisconsin, western Ohio, central Kentucky, western Ne south of Lake Ontario, and farther east at town of Clinton (whence their in Pennsylvania, Virginia, eastern Tennessee, Georgia, and Alabama. greatest development is in Alabama, where they form an inner terrace Red Mountain, in the Birmingham anticlinal valley. Good coking co limestone are near, so that low-cost pig can be produced even from 35 a ores. At outcrops the ores are siliceous; below ground-water level, t come basic. All are moderately high in phosphorus. They are p oölitic beds, precipitated in shallow estuaries, fed by iron-bearing d

Smyth). Although utilized in Tennessee and Georgia, they are most ... in Alabama. In time those in New York, Kentucky and Wisconsin ... to be of greater moment than now.

Brown hematites ("brown ores") of the U S are produced chiefly along that portion ... Appalachian mountains formed by early Paleozoic strata, and just west of the ... crystallines. They are products of weathering of ferruginous rocks, especially ... ores. In almost all the mines they must be freed of ochers and clays by washing.

Magnetites occur in two chief types of deposits. The commoner is a ... or pod-shaped mass, in gneisses, parallel with the foliation. They ... widely in the ancient Appalachian crystallines, but are most productive ... Adirondacks. The second type appears in the contact zones (Art 17, *c*), ... by intrusive igneous rocks on limestones or limey shales. The greatest ... of this type in the East is at Cornwall, Penn. The igneous rock is ... and the limestone, Cambro-Ordovician. In the West many such de- ... are known, in Utah (Iron Springs Dist), Nevada, California, and along ... west coast.

Europe, Germany, England, and France follow the U S in order of production of iron ... greatest single source being found in a series of Jurassic beds in and near Luxem- ... in the northeast of England. The ores (called "minette") are of 30% or a ... ther, and are brown hematites, carbonates, and various silicates. Minor ore ... in veins, of carbonates, and in beds of clay iron-stone and black-band. Spain ... Bilbao great quantities of partially hydrated hematite, the weathered product ... ores in depth. Sweden is a heavy exporter of magnetites, especially from the ... chest of magnetite at Kiruna, Lapland. Many lenticular magnetites have been ... in middle Sweden. Great bodies of igneous magnetites exist in the Urals. Algiers ... important amounts of red, partly hydrated hematites.

... ore bodies have recently been developed in N E Cuba, where ancient serpen- ... weathered for ages, leaving a residual soil, rich enough in iron to form ore. ... reserves of specular hematite are reported from eastern Brazil, state of Minas ... They are of ancient geological age, but as yet (1914) lack rail connection with ... With the opening of the Panama canal, iron ores will reach American fur- ... Chile. In S E Newfoundland, at Wabana, extensive beds of red hematite ... been developed in recent years (24).

19. Copper Ores

... following minerals constitute the common ores of copper:

Table 4. Copper Ores

Sulphides	% Cu	*Sulphates*	% Cu
...rrite,		Chalcanthite, $CuSO_4 \cdot 5H_2O$.....	25.38
...($Cu_2S \cdot Fe_2S_3$).....	34.60	Brochantite, $Cu_4(OH)_6SO_4$.....	56.1
... (Cu_2FeS_2(2 $Cu_2S \cdot Fe_2S_3$)	55.50	*Carbonates*	
... CuS	66.48	Malachite, $CuCO_3 \cdot Cu(OH)_2$....	57.27
... Cu_2S.....	79.80	Azurite, $2CuCO_3 \cdot Cu(OH)_2$.....	55.10
... and Sulphantimonides		*Silicate*	
... Cu_3AsS_4(3 $Cu_2S \cdot As_2S_3$)	48.40	Chrysocolla, $CuSiO_3 \cdot 2H_2O$.....	36.00
...		*Oxychlorides*	
...($4Cu_2S \cdot Sb_2S_3$).....	52.06	Atacamite, $Cu_2Cl \cdot (OH)_2$.....	59.29
Oxides		*Native Metal*	
..., CuO.....	78.86	Native copper, Cu.....	100.00
... Cu_2O	88.80		

... distinction between primary and secondary minerals is more important ... copper than with any other metal. Some copper minerals appear in bo...h

groups. PRIMARY: chalcopyrite, bornite, chalcocite, enargite, tetr
(possibly native copper in Lake Superior mines). SECONDARY: ch
covellite, melaconite, cuprite, chalcanthite, brochantite, malachite,
chrysocolla, atacamite, and native copper. Possibly chalcopyrite and
are secondary in some cases. As a primary mineral lean copper-bearin
is very important, especially in intrusive rocks. When oxidized by r
waters in the vadose zone (belt of weathering), all copper-bearing s
yield some form of sulphate. This soluble salt, in deposits in siliceou
trickles downward until, in contact with some reducing agent, like py
copper is precipitated as chalcocite. This causes great concentration of
at or near ground-water level, termed SECONDARY ENRICHMENT. From b
copper-bearing sulphides, in regions of abundant rainfall, as at Ducktow
an upper zone or GOSSAN of brown hematite results, which may form an i
Below this, near ground-water level, a belt of rich chalcocite (black ore)
containing most of the copper once distributed throughout upper par
posit. Still lower are unaltered, primary sulphides. In a comparativ
region, when copper-bearing sulphides, usually in form of cupriferous
are disseminated in intrusive igneous rocks, or quartzites, or schists
may be crushed or rendered open-textured along a zone of moveme
scending waters of the vadose zone develop an upper leached belt, unde
a chalcocite-bearing section of maximum richness; and below this is a
secondary enrichment. Thus have originated the disseminated copp
now being extensively mined in the southwest. If oxidizing reactions
open-textured tuffs, or contact lime silicates, chrysocolla often results,
of chalcocite; if in presence of limestone, the blue and green carbona
cuprite are characteristic products. In North America most of the cop
duced comes from chalcocite.

Examples of copper deposits (letters in parentheses refer to Classi
Art 17).

A. Bodies of copper-bearing sulphides, chiefly chalcopyrite o
copper-bearing pyrite in igneous rocks (*a*). Chalcopyrite may be as
with a nickel-bearing sulphide, pentlandite, and with pyrrhotite, in b
trusives (Sudbury, Ontario). Lean copper-bearing pyrite of igneous i
masses, usually monzonites, requires secondary enrichment for profitab
ation (Bingham Canyon, Utah, and near Ely, Nevada).

B. Irregular masses of copper-bearing sulphides in contact zones
associated with lime silicates. Secondary enrichment, incident to oxidati
be necessary to increase percentage to mining requirements. Various min
of deposit may be associated. Bisbee and Morenci, Ariz, are best illust

C. Veins along faults, with greater or less replacement and impre
of the walls (*d*), as at Butte, Mont, where walls are granite. Innu
other veins are known in all parts of world.

D. Lenticular or pod-shaped bodies of pyrite or pyrrhotite, with
pyrite (usually of later introduction). The lenses favor schists or slat
lie parallel with the foliation. These rocks may be sheared eruptives, or
sediments (Ducktown, Tenn, and many ore bodies along Appalachians).
examples appear in foot-hills of Sierra Nevadas, Cal. Rio Tinto, Spain,
haps the largest body yet discovered.

The ore bodies were probably originally veins (*d*) parallel to structural planes
rocks, and subsequently pinched into lenses by pressure. The type called "Ki
may be of sedimentary origin or introduced as veins and pinched by pressure. So
been considered igneous intrusives, as at Sulitelma, Norway, and Bodenmais, Bav

Native copper in nodules, sheets, minute scales, and sometimes large masses, in amygdaloidal basalts, and associated conglomerates (*h*). Point, Michigan, is chief example. Introduction of the copper is a subject, whether a product of expiring igneous activity, or of circ- meteoric and connate waters.

Impregnations of sedimentary rocks with sulphides or their oxidized often deposited on organic remains (*h*). Mansfeld, Germany, is best example, where a black shale, with abundant organic remains, is im- with copper minerals for a width less than 1 ft, but over a great It is uncertain whether the copper was precipitated from Permian sea- along with the sediments, or introduced by circulating ground-waters after sediments were deposited. Triassic strata of the U S have many impregnations, mostly small.

percentage of copper for successful mining depends on widely varying conditions. copper rock, on Keweenaw Point, Mich, yielding only 0.65% (13 lb per ton), treated successfully. The disseminated chalcocite of Bingham Canyon, Utah, ed average assays over 3 months' periods as low as 1.3%, with approximately h recovery. Raw smelting ores, of slightly above 2% and with little aid from metals, have been worked at Ducktown, Tenn. In early days in western U S, to 20% were frequent in oxidized and enriched parts of deposits. Last quan- 100% ore are now reported from S E Congo State, Central Africa. To be valu- low-grade deposits must be of great size (25).

20. Lead and Zinc

Lead. Following minerals constitute the common ores of lead: galena, 86.6% lead; anglesite, $PbSO_4$, 68.3%; cerussite, $PbCO_3$, 77.5%; pyro- 3 $(PbO \cdot P_2O_5)\,PbCl_2$, 76.2% (much rarer than the others). Other ds sometimes appear in small amount, as wulfenite, crocoite and

na is the chief primary lead ore, of which others are oxidation products. ently associated with zinc blende and pyrite. All lead ores are com- in limestones than with other wall rocks. Many lead ores carry silver mercial amounts, especially in regions characteristically productive of metals. Gold is a rarer associate. The oxidized product of galena is erussite than anglesite. All oxidized ores are mingled with limonite g degree, and with silica and earthy minerals from alteration of wall Lead and zinc can best be discussed together.

Zinc. Following minerals constitute the common ores of zinc: sphalerite or de, ZnS, 67% zinc; calamine, $2\,ZnO \cdot H_2O \cdot SiO_2$, 54.2%; smithsonite, 52.1%; willemite, $2\,ZnO \cdot SiO_2$, 58.6%; zincite, ZnO, 80.3%; frank- $(Fe,Mn,Zn)O(Fe,Mn)_2O_3$, variable, about 6.0%. Willemite, zincite and e are so exceptional in their occurrence in northern New Jersey as to group by themselves.

lende is the almost universal primary mineral; calamine and smithsonite are its products. The latter two are often inseparably mixed, and together are known as "zinc" in Europe. "Dry-bone" is a local name in Mississippi Valley, the oxida- products suggesting old bones. (The significance of the names calamine and smith- England is the exact reverse of the American meaning, smithsonite being used silicate.) The oxidized compounds are characteristic of the vadose zone. coat bedrock beneath cap of residual products. The deposits are summarized with lead alone to those with zinc alone. Intimate mixtures of both ores afford great metallurgical problems today. Neither lead nor zinc deposits have been immediate association with igneous rocks, such that a direct igneous origin could ed to them. They obviously reach their places of precipitation in solution.

Examples of lead and zinc deposits. (Letters in parentheses re Classification, Art 17.)

A. Disseminated and sometimes coalescing deposits of galena associated sulphides, in sedimentary strata. The galena impregnates th sediment; believed to have been introduced in solution, and to have re preëxisting minerals; source is conjectural (*h* and *i*). In S E Missouri, th American source of lead for lead alone, Cambrian limestones are impreg

Near Laurium, Greece, galena replaces limestones involved with mica schis Leadville, Colo, Carboniferous limestone has been replaced with silver-bearing pyrite, manganese compounds and sometimes zinc blende, along under sides of rhyolite-porphyry ("white porphyry"). Extensive oxidation developed carbona for the early miners. Galena may yield to zinc blende in amount. In Belgium, I burg, and near Aix-la-Chapelle, huge subordinate lead ores have been mined vonian and Carboniferous limestone along great faults. Zinc blende was doubtl original mineral. In Silesia the zinc and lead ores are in Triassic limestones.

At Commern, Germany, knots of galena are disseminated in Triassic sandsto Cœur d'Alene district, Idaho, silver-bearing galena, with siderite, appears in bodies in pre-Cambrian quartzite, along or near extensive faults. Zinc blende ha met in some mines; copper ores in a few others. Siderite seems to have first re the quartzite, and then yielded to galena. The reaction is much the same as with c limestones.

In S W Missouri zinc blende and subordinate galena impregnate brec chert, interbedded in Lower Carboniferous limestones.

B. Galena, zinc blende and associated sulphides in joints ("veins") and related cavities (*f*). In S W Wisconsin and neighboring of Upper Mississippi Valley the Ordovician "Galena" limestone has nun vertical gash veins, with horizontal "runs" and inclined "pitches," cont galena, zinc blende, marcasite, and calcite.

C. Galena and zinc blende in fissure veins, often together, often sep usually with other sulphides (*d*). Precious metals are frequently assoc Such deposits are world-wide, and in all kinds of wall-rocks.

D. Lenticular deposits containing willemite, franklinite, subordinate z and many lime silicates, now folded in pitching synclinal troughs in pre-Cam limestones. It is difficult to classify these deposits. Their zinc-bearing mi are unique. The mineralogy suggests contact zones (*c*), but the actual morphosing igneous rock is not apparent.

In recent past, the price of lead has varied from extreme minimum of 3¢ per maximum of 5¢, ruling in late years not far from 4¢. Each per cent in a ton means roughly a value of 80¢, of which the miner may expect about 40¢. Z spelter rules about 1¢ above lead for ordinary brands, and 2¢ above for brands cally free from lead, antimony or arsenic. Each per cent of ordinary spelter in of ore means roughly $1, of which the miner receives about 50¢. Possibilities of n and concentration may be estimated on above basis.

21. Silver and Gold

Though deposits are known containing either gold or silver alone, metals are generally associated and must be discussed together. Both ar tensively obtained in connection with copper and lead. Lead ores are called "wet ores," because metallic lead, freed in smelting, acts as a solver the precious metals, the distinctive ores of which are called "dry ores." desilverization for base bullion, electrolytic refining for copper, and the stitution of cyanidation for amalgamation, have greatly facilitated treat of silver and gold ores.

Silver-bearing minerals: argentite ("silver glance"), Ag_2S, 87.1% silver; [illegible] $AgFe_2$, 62.8%; proustite ("light ruby ore"), Ag_3S_3As or $3\,Ag_2S \cdot As_2S_3$, pyrargyrite ("dark ruby ore"), Ag_3S_3Sb, or $3\,Ag_2S \cdot Sb_2S_3$, 59.8%; [illegible]ite ("brittle silver ore"), Ag_2S_4Sb or $5\,Ag_2S \cdot Sb_2S_3$, 68.5%; cerargyrite ("horn silver"), AgCl, 75.3%; native silver, Ag, 100%.

Silver almost always contains at least a trace of silver, which probably [illegible] as an isomorphous sulphide, but not appearing separately in polished [illegible]. Silver is a component of certain varieties of tetrahedrite, and in this [illegible] often found in copper ores.

[illegible] probably enters as a base into other copper minerals. Silver is sometimes found in [illegible] rarely in pyrite. Modern silver production is chiefly in connection with [illegible] metals. Cerargyrite and native silver are habitually secondary minerals, resulting [illegible] alteration in the vadose zone of other minerals mentioned above, or from silver-[illegible] base-metal minerals. Argentite probably is sometimes secondary; it certainly [illegible] primary. The others are generally primary.

Gold-bearing minerals: calaverite, $AuTe_2$, 44.5% gold; sylvanite ("graphic [illegible]"), $(Au,Ag)\,Te_2$, variable; native gold, alloyed with silver, etc, vari-[illegible] Gold most commonly occurs in quartz veins, both as native, and as [illegible] and wires mechanically mixed in pyrite. It may be set free by oxidation [illegible] removal of the pyrite. It also accompanies mispickel, chalcopyrite, and [illegible] galena. In some of these minerals, when the ores are refractory, it may [illegible] as an involved telluride, or as a bismuth compound (Richard Pearce). [illegible] tellurides of gold (a number of rare mixed tellurides of gold and other [illegible] are not mentioned above) are primary minerals. On oxidizing and [illegible] of tellurium, they yield extremely fine particles, not readily panned and resist-[illegible] amalgamation; called "rusty" gold. Gold, presumably as chloride, some-[illegible] descends in solution from oxidized portions of veins containing manganese [illegible] oxids, and is reprecipitated at or near water level. Presence of calcite may [illegible] here with the reaction (7). The high sp gr and resistance of gold to natural [illegible] cause greatly favor the formation of placer deposits.

[illegible] American writers sometimes classify precious metal deposits into an older series, in [illegible] Mesozoic age, and a later series. The great silver-gold veins associated with mountain [illegible] igneous and igneous outbreaks, at close of the Cretaceous and in the opening Tertiary [illegible] period can thus be distinguished from older ones. Each group can then be subdivided [illegible] associated minerals, of which a series of subtypes can be established. Other writers [illegible] placed less emphasis on variations in time and mineralogy. Admitting some [illegible] characteristic mineral associations, which might make possible finer subdivision, the following [illegible] large types are permissible (see Classification, Art 17).

Predominant silver. A. Fissure veins, with distinctively silver min-[illegible] in quartz gangue, often amethystine and associated with manganese [illegible] and some calcite. Galena, zinc blende, pyrite and copper minerals, [illegible] very subordinate. Many great veins of Mexico, as at Pachuca, Real del [illegible] and Guanajuato, exhibit these characters. The Butte silver veins are [illegible] but now, in instances, tend to develop copper in depth. In general, [illegible] is the chief source of silver (*d*).

B **Fissure veins yielding silver with little gold,** in association with galena, [illegible] blende, copper minerals, and pyrite, in gangue of quartz, calcite, barite, fluor-[illegible] or several (*d*). Veins of this mineralogy are world-wide in distribution.

C **Native silver and minor silver minerals,** with arsenides of cobalt and [illegible] in shrinkage cracks or fissures involving slight displacement (*d* and *f*). [illegible] Ont, is best example, where veins are predominantly in Huronian con-[illegible]rate, associated with a diabase sill, which has some veins, and Keewatin [illegible] schists, which contain a few.

D. Impregnations of porous rocks, sandstones, tuffs, etc, with ar cerargyrite, and native silver; supply fissures obscure; some copper m may occur. (Silver Reef, Utah, Silver Cliff, Colo.)

E. Impregnations and replacements of crushed rocks along (silver-bearing galena in Cœur d'Alene, Idaho, Art 20, A).

F. Replacements of calcareous rocks with silver-bearing galena an ciated sulphides. Leadville, Colo. (See Art 20, A, for other cases.)

Predominant Gold. A. Fissure veins containing native gold, al mechanically mixed in pyrite and much rarer base-metal sulphides, in gangue. Gray, greasy-looking quartz seems to accompany best values common association of quartz with gold makes this type of world-wide bution. Veins appear most frequently in schists, slates, or other metan rocks, and in association with intrusive rocks, of which granite is comm

B. Impregnations and replacements of open-textured rocks wit bearing pyrite. The "banket" of gold-bearing conglomerates of Transv chief producers today, is the best example.

C. Saddle-reefs, or arch-like deposits of gold-bearing quartz at cr anticlines (*e*) (Bendigo, Victoria, and gold reefs of Nova Scotia). Sadd may succeed one another in depth. Slates or slaty schists are commo rocks.

D. Veins carrying gold tellurides. At Cripple Creek, Colo, th associated with an eroded Eocene volcano, often favoring neighborh minor dikes of phonolite and basaltic rocks, with which volcanic activity Purple fluorite is a characteristic associate. In Boulder Co, Colo, veins gneisses; at Kalgoorlie, Western Australia, in amphibolites; in Hung tered andesitic rocks, called propylites. Once considered extremely rare rides have been very productive in Cripple Creek and Kalgoorlie.

E. Lateral impregnations and replacements of calcareous shale tellurides along supply fissures, called verticals. Example, so-called "Pot or "refractory ores," of the Black Hills, S Dak, the walls of which Cambrian age.

F. Contact zones, on the border of intrusive igneous rock and lim containing gold-bearing mispickel in lime silicates (Nickel-plate mine, The usual contact zone of this type carries copper sulphides, with a gold (Art 19, B).

G. Placer deposits of gold-bearing gravels, which may be: residua weathering of rocks *in situ;* river gravels in active streams; river gra abandoned and often buried channels; alluvial fans; sea-beaches with surf; sea-beaches now elevated and inland. Gold in streams favors where current has been checked, as the inside of bends; junctions of tribu heads of quiet reaches. Gold favors gravel next the bedrock, or next a bedrock" of clay, but fine particles may be generally distributed in a thi tical section. Magnetite, zircon, garnet, and various resistant, heavy m are characteristic associates, yielding "black sands" (26).

22. Minor Metals

Aluminum is obtained today from bauxite, representing a series from Al_2O to $Al_2O_3 \cdot 3(H_2O)$, which is treated electrolytically in a bath of cryolite ($3\,NaF$ Bauxite is developed by weathering of aluminous rocks, and may appear as a r product. It is probably also produced by solvent action of sulphuric acid, from

...rite upon aluminous rocks, such as shales. The resulting acid solution of alu... sulphate may be neutralised by limestone, with precipitation of aluminum hydrate, ... may then form concretionary masses. In America bauxite is largely produced in ... where last named reaction is believed to explain its occurrence. In Arkansas, it ...ciated with syenitic eruptives, to the alteration of which, perhaps while yet hot, its ...on is attributed. Cryolite is commercially obtained only on west coast of Green... where it constitutes a large, flat vein in gneiss. Siderite, galena, zinc blende, and a ...ter minerals are sparingly mingled with it.

Antimony is obtained from its sulphide, stibnite (Sb_2S_3); sometimes from the oxide, ...timonite (Sb_2O_3); and as an alloy from antimonial lead ores. Characteristic occur... of stibnite is in quartzose veins, but some less regular deposits in sandstone have ...n recorded. The industry is small.

Arsenic is produced almost entirely as a by-product in smelting arsenical copper ores, ... enargite (Cu_3AsS_4). It is sold as oxide, but much more could be saved were ...e a market for it.

Barium is chiefly consumed as the sulphate ($BaSO_4$). Barite characteristically ap... in veins in limestones; is also a frequent gangue with lead and copper ores, re... of nature of vein walls. Chief output in U S comes from residual clay deposits ... and Texas.

Bismuth is a rare by-product in lead-silver refining. A few districts, Telluride and ...e, Colo, produce small amounts of high-grade bismuth ore.

Cadmium is a minor associate of zinc, and whenever separated is a small by-product ...c metallurgy, or in treatment of zinc-bearing lead ores.

Cesium is a rare alkaline element of much the same associations as rubidium.

Cerium, with didymium, erbium, lanthanum, thorium, and yttrium, constitutes a ... called the cerium group of rare earths. Their compounds, especially those of ...m, have incandescent properties when heated, for which purpose they are sought. ...y are obtained as phosphates in monazite and xenotime, and are characteristic of ...ites. They may appear in normal granite. Being resistant and heavy, monazite ...ains have accumulated in placers in the drainage of pegmatite and granite ... the Carolinas; also on sea coast of Bahia, Brazil. Relatively high percentages ...ium are desirable.

Chromium ($FeO \cdot Cr_2O_3$), sometimes $(Fe \cdot Mg)O \cdot (Cr \cdot Al \cdot Fe)_2O_3$, is a characteristic ...ite of richly magnesian, basic igneous rocks, usually altered to serpentine. The ...ite is believed to be a direct crystallization from molten magma. It forms irregu... sometimes large, distributed masses, and being extremely resistant may be freed ...concentrated as a residual product in weathering. Commercial chromite should ...ain at least 40% Cr_2O_3.

Cobalt forms a variety of arsenides and sulphides, practically always in association ...kel. Linnæite (Co_3S_4), smaltite ($CoAs_2$), cobaltite (CoAsS), and the oxidized ...ty erythrite or "cobalt bloom" ($Co_3As_2O_8 \cdot 8\,H_2O$). Once the object of a small ...metallurgy and produced as a by-product in the larger nickel industry, since ...overy of the rich silver-cobalt veins, at Cobalt, Ont, cobalt is over-supplied.

Didymium (see Cerium).

Erbium (see Cerium).

Iridium see Platinum).

Lanthanum (see Cerium).

Lithium is obtained from 2 silicates: lepidolite, the lithia-mica ($LiK \cdot Al_2O_3F_2Si_3$) with ...0. 2 to 5%; and spodumene ($LiO \cdot Al_2O_3 \cdot 4\,SiO_2$) with lithia 7.5%. Both are ...ite minerals, occurring mainly in Black Hills, S Dak. Lepidolite is more easily

Magnesium is chiefly used as the earthy carbonate, magnesite, a refractory material. ...ite favors association with serpentines, in the alteration of which it is formed ... Hungary, Greece, and California are chief producers.

Manganese has one principal ore, pyrolusite (MnO_2) with Mn, 63.2%, and
minor ores. Other forms of oxides, *viz:* braunite (Mn_2O_3); hausmannite (M
psilomelane, hydrated oxide of variable composition; franklinite (Art 20). I
logical relations of its ores, manganese is similar to the brown hematites (Art 6).
ores are usually residual products of weathering, and are found as nodules in cla
masses on surface. They should be relatively low in phosphorus, for use in s
eisen, and not too high in silica (27).

Mercury has one chief ore, cinnabar (HgS), with which a little native metal
associated. Cinnabar appears in veins with quartz, calcite, altered wall-rock and bi
It may also impregnate porous beds, such as sandstones. In practice mercury is
quicksilver or "quick." Its importance has declined with the increased use of c
solutions for extracting gold and silver.

Molybdenum is obtained from molybdenite (MoS_2). Wulfenite ($PbMoO_4$)
tracted some attention. Molybdenite is a pegmatite mineral, sometimes seen in
quartz veins. It is rather frequent in contact zones, produced from limestones
trusive igneous rocks, but only in small amount.

Nickel has 3 varieties of ores: (*a*) sulphides and arsenides (millerite,
niccolite, NiAs) and related minerals, all of small moment today; (*b*) pen
ite, $(NiFe)_7S_8$, or some similar atomic ratio, the nickel, iron and sulphur
each about one third. (Although nickel was formerly thought to replac
in pyrrhotite, it is now proved to be in mechanically intermingled pentla
(*c*) a series of hydrated silicates of nickel and magnesium, somewhat ana
to serpentine in general composition and forming veins in serpentine,
duced in the alteration of very basic igneous rocks.

The nickel industry today is practically limited to 2 localities. At Sudbury, C
vicinity, pentlandite with pyrrhotite and chalcopyrite are concentrated at botto
huge intrusive sheet, which varies from norite at the base, where the ores appear,
micrographic granite at its upper surface. The sheet is folded into a huge ba
miles across, and is buried in center beneath overlying sediments and volcanic
ores favor embayments in the underlying older rocks, and one offsetting dike
outer periphery. The underside of sheet dips inward at 60°, and is impregnat
nickel and copper ores up to widths of 100 or 150 ft. The gangue is self-fluxin
ore bodies are generally classed under *a*, Art 17, but the 3 sulphides seem to have
lized in serial order, and at times to have undergone some redeposition. The h
silicates of nickel are found, in workable richness, in the serpentinous district
Caledonia. They constitute veins, and are the second important factor in the
supply (28).

Osmium is a characteristic associate of platinum, in the placers of which
source, iridosmine, is found, in scales or scaly nuggets and "colors."

Palladium is a characteristic associate of platinum in placers, and in the fe
where platinum has been discovered in copper ores. In latter case careful as
necessary to avoid mistaking palladium for platinum.

Platinum appears in metallic grains and nuggets, more or less alloyed wi
palladium, and rarer metals of the platinum group. It is characteristically as
with peridotites and pyroxenites, in which it is a direct crystallization from the
fused magma. Platinum is frequently intergrown with chromite. Aside from
alloys, its one compound is sperrylite ($PtAs_2$), a minute associate of the Sudbury
copper ores, and rarely elsewhere (29).

Potassium is treated as a saline (see Non-Metallic Minerals).

Radium is an extremely rare associate of the more abundant uranium, from t
erals of which it is separated (see Uranium).

Rhodium is a minor associate of platinum.

Rubidium is a rare alkaline element, associated in minute amounts with lith
lepidolite and other lithium minerals.

Ruthenium *is* an extremely rare associate of platinum.

...um is treated as a saline (see Non-Metallic Minerals).

...ium is obtained from the sulphate, celestite ($SrSO_4$), occurring like barite, but ... (see Barium).

...um (see Cerium). Thorianite is also found commercially in Ceylon and ...

...n as one ore, cassiterite (SnO_2), and one rare sulphide, stannite. Cassiterite is al... associated with granites and pegmatites, or veins closely akin to pegmatites. ...ering and erosion of these, being heavy and resistant, it is concentrated in ... as pebbles and finer particles, called stream tin. Cassiterite is obtained both by ... and placer working. It has been observed associated with rhyolites. Tin ... obtained from some Bolivian silver-bearing veins.

...um appears in the titaniferous magnetites, in which it has hitherto been a dis... to the iron. Varieties extremely rich in titanium, and pegmatitic veins or ... containing rutile (TiO_2), are now mined as a source of ferro-titanium. Titaniferous ... are intergrowths of magnetite and ilmenite, often extremely intimate.

...sten is obtained from several tungstates, *viz.* wolframite, $(FeMn)WO_4$; huebner... $MnWO_4$; scheelite, $CaWO_4$. It is a characteristic mineral of pegmatite and veins ... to pegmatites. Scheelite appears at times in gold-bearing quartz veins.

...nium has gained great prominence as the associate of radium, but has also uses of ... Pitchblende or uraninite, $(UPb_3)_3U_2O_9$, the earlier and still prized source, is a ... characteristic mineral of pegmatites and related veins. A series of phosphates, ...ite, autunite, etc, have similar geological relations. Carnotite, a vanadate, $K_2O \cdot V_2O_5 \cdot 3\,H_2O$, with 15 to 18% vanadium oxide, is found impregnating ... in western Colorado and eastern Utah, and is now much sought for radium.

...nadium is a minor component of titaniferous iron ores, and of the uranium-bearing ... and, in a series of vanadium sulphides and their oxidized derivatives, appears ... veins in Peru.

...rium (see Cerium).

...conium has one mineral, zircon ($ZrSiO_4$), an associate of granites and other feld... rocks and pegmatites, from which on weathering it is freed and concentrated in ...

NON-METALLIC MINERALS

Here is included a miscellaneous series with no fundamental relations; hence, ... alphabetically. The carbon series is the most important (30).

28. Abrasives; Asbestos; Asphalt

Abrasives. Corundum and emery (emery is a mixture of corundum, spinel, magnet... and other hard and heavy minerals) are found in two principal geological relations: ... crystallizes from rare igneous magmas containing excess of Al_2O_3 above re... ments of ordinary rock-making minerals. Nephelite-syenite, in Ontario, and peri... in North Carolina, fulfil these conditions. EMERY is commonly found at igneous ... or where inclusions of aluminous sediments are involved and partly digested in ... as rocks. A vein or bed at Chester, Mass, containing emery in metamorphic rocks, ... interest. GARNET, either in hornblende schist, as in the Adirondacks, or in mica ... as at Reading, Conn, is a minor abrasive. Crushed, angular fragments of QUARTZ ... used for sand-paper. DIATOMACEOUS EARTH and decomposed chert (TRIPOLI) are soft ... WHETSTONES are made of gritty slates, which sometimes owe their "tooth" ... garnets or other hard minerals; or of novaculite, a fine-grained siliceous rock ... the solution and removal of minute rhombs of calcite have left sharp-edged ... Coarse varieties are sandstones, or sandy schists, in which are set rutile, garnet, GRINDSTONES are made of sandstones sufficiently friable not to wear smooth (31).

Asbestos of commerce, a variety of serpentine, called chrysotile, appears as veins ...-fibers in some serpentine districts (most important is in southern Quebec). ... grades appear along slips in the serpentine and in the mass of the rock. Some

believe the Canadian asbestos to be a deep-seated alteration product of basic rocks; others that it is developed from serpentine in fissures near intrusive aplite, a variety of granite (32).

Asphalt (see Carbon Minerals).

24. Building Stone; Clay; Limes; Cements

The granite industry is mainly developed along Atlantic seaboard; secondarily Wisconsin, Missouri, and California. Among igneous rocks, granite breaks best quarry. When of good grade, it is homogeneous in texture, though sometimes spoiled from black inclusions, local coarse crystallizations, and development of gneissoid ture. SANDSTONES are widely quarried. The "brownstone" of eastern U S is a sandstone, from Longmeadow, Mass; Portland, Conn; Avon, N J; Hummelstown Harrisburg, Penn. "Bluestone," of Hudson River region, is a Devonian argill sandstone, specially adapted to flagstones, curbing, sills, and lintels. Potsdam red stone or quartzite is Cambrian; quarried on western side of Adirondacks. A stone of nearly the same geological horizon is produced on south shore of Lake Su Medina pink sandstone of the Silurian is extensively obtained along the Erie between Rochester and Lockport, N Y. Cleveland or Ohio sandstone is a gray or blue stone, of Mississippian (Lower Carboniferous) age, developed in outskirts of land. LIMESTONES. Preëminent is the Indiana or Bedford oölitic stone, of Mississ (Lower Carboniferous) age, which outcrops in an extended N and S belt in S W In MARBLES, of Cambrian and Ordovician age, are extensively developed along the of Western Vermont, Eastern Tennessee, and Georgia; of other age, in Colo. S appear in S W Vermont and neighboring parts of N Y, and in the Lehigh Valley, They are in less degree produced in Virginia, the Lake Superior region, and Newf land. Wales is a famous source of slate and of skilled workers in slate. SERPE is quarried in southeastern Pennsylvania and the neighboring parts of Maryland (3

Clays are of two kinds: residual, especially from weathering of limestone; and ported, which are the finest sediments of still water. Residual clays are comm south of the terminal moraine of Glacial epoch. They are impure and variable. T ported clays were extensively deposited by the floods which followed the melting of th tinental glacier. They are very abundant in the valleys of the Connecticut and H rivers, and are the basis of a great brick industry. Fireclay for refractory materials s be as free as possible from other ingredients than SiO_2, Al_2O_3, and H_2O. It is often beneath coal seams. In these relations fireclays are mined in Pennsylvania, Mary Ohio, and at Cheltenham, Mo. Other fireclays of Cretaceous age are developed at V bridge and its neighborhood, N J, and near Golden, Colo (34) (see also Sec I, Table V

Shales often possess properties which fit them for vitrified brick. They are ground, moulded, and hard-burned. They are useful for pavements, especially whe good rock is available for macadam.

Limes and cements. For quicklime, calcium carbonate should be pure, free coloring ingredients, such as iron compounds, and is preferred with little magnesium bonate. Silica and alumina together develop hydraulic properties, and injure " limes. Kilns are widespread and, for local use, any reasonably pure limestone ans Rockland, Me, is the principal American producing district. As limestones contai creasing amounts of alumina and silica, they develop hydraulic properties when bu and in varieties of special excellence afford natural rock cement. The crude sto called a "water-lime." Although important in former years, the natural cements given way to "Portland" cement, which is an artificial mixture of limestone and cl shale, entirely under control of the chemist, and being therefore more uniform in pr ties. In Portland cement magnesia is kept very low, not over 2 or 3%. The Le Valley, Penn, is chief center of manufacture in U S, but plants are widely distributed

25. The Carbon Minerals

These embrace Coals and their relatives, and the Petroleum series, incl ing Natural Gas, Maltha, Asphalt, and Asphaltites.

Coals and their relatives are vegetable remains so preserved in sediment strata as to become progressively enriched in carbon. They begin as some f

...y tissue, perhaps also in part spores, algæ, and resins; under conditions ...ded oxidation they pass toward a theoretical limit of nearly pure carbon, ...ally to mineral ash. Cellulose, the principal original contributor, is ... approx, C 50%, H 6%, and O 44%), but there was always also a little ... mineral matter in original deposit. If vegetable tissue accumulates ... protecting layer of water, oxidation is retarded and relative enrichment ...es ensues. On subsidence of the land, or, in case of lakes and swamps, as ... heavy floods, sediments bury the accumulated vegetable tissue. The ... process comprises several stages. PEAT is still brown; a visible aggregate ... leaves, etc; high in O and H and relatively low in C. LIGNITE is firmer, ...ick, but has a brown streak, and usually still shows evidence of vegetable ... less O and H than peat and relatively more C. Sub-bituminous coals ...ch lignites are a stage beyond typical lignite, but are not typically bitumi-... BITUMINOUS COALS (Sec 35) are black, solider, lower in O, higher in C, and ...ost possess coking properties. SEMI-BITUMINOUS and SEMI-ANTHRACITE ...passages to ANTHRACITE (Sec 34) in which the C is greatly enriched and the ...ard and firm. Still further stages toward graphite are known.

Table 5. Characteristic Chemical Composition of Coal Series

...ame	Peat	Lignite	Bituminous	Anthracite
...	59	69.0	82.0	95.0
H	6	5.5	4.5	2.5
...	33	25.0	13.0	2.5
N	2	0.5	0.5	trace

...r is also present in varying percentages, up to several units, and mineral ash ... ash.

...ls are analyzed commercially in 2 ways, proximate and elementary. In PROXI-...ATES, moisture, volatile matter, fixed carbon, ash and sulphur are usually de-...ed. Sample is dried, weighed, ignited until flames cease or for a standard time ... standard Bunsen burner, weighed, ignited again to consume the carbon; after ...duct is weighed for ash. Sulphur is determined in a separate sample. This ... shows if the coal is high or low in water; is high or low in volatiles; has a ... short flame; cokes or not; is high or low in ash; is sulphurous or not. These are ...st important points regarding a fuel. Dividing the percentage of fixed carbon ... percentage of volatiles gives the "fuel ratio," characteristic for each particular ...: anthracites give high, and richly bituminous coals low ratios. Ratios vary from ... 1. to about 30. ELEMENTARY ANALYSIS of a dried sample gives the C, H, O, N, S, ... It affords a better idea of the heat units in the coal, a matter of growing im-...ce each year, but does not indicate coking properties nor volatiles. It is known ... O in coals is already combined with C or H, and is as inert as ash, besides re-... the available C and H. Since H has a high calorific value, the necessary pro-... to yield H_2O with the O present is often called combined hydrogen; the excess, ...ee hydrogen. Relatively high values of the latter are esteemed.

Table 6. Characteristic Proximate Analyses

...	Moisture	Volatiles	Fixed carbon	Ash	Sulphur
...	20.22	52.31	24.52	2.95	.....
...	17.75	37.85	37.40	6.20	0.80
...minous	13.43	37.15	45.57	3.85	.....
...ous	1.26	30.11	59.62	8.23	0.78
...minous	1.23	15.47	73.51	9.09	0.70
...thracite	1.29	8.10	83.35	6.23	1.03
...ite	4.12	3.08	86.38	5.92	0.50

Table 7. Characteristic Elementary Analyses

	Moisture	C	H	O	N	S
Peat (air dried).....	13.60	40.78	5.55	30.95	1.40	0.58
Lignite.....	22.63	54.91	6.39	32.59	1.02	0.59
Sub-bituminous.....	11.05	59.08	5.37	21.52	1.33	1.73
Bituminous.....	3.36	68.69	4.84	11.49	1.54	1.01
Semi-bituminous....	1.53	82.87	4.76	4.99	1.68	0.65
Anthracite	1.97	91.40	2.81	1.83	0.21	0.71

Heat units are expressed as British thermal units (B t u), or as French (High-ash or high-oxygen coals give low thermal values. The usual range is as the peat being exceptionally good (Sussex Co, N J):

Peat	Lignite		Bituminous coal		Anthracit	
High	Low	High	Low	High	Low	H
8 260	7 204	10 143	10 242	13 790	12 047	14

Moisture in coal is a serious drawback, and varies so much in unprotected especially of lignites and sub-bituminous coals, that the sample, as soon as cut in t should be put into an air-tight glass jar, and analyzed as soon as possible afte unscrewed.

Classification. Many attempts have been made to classify coals, but for easte the scheme suggested by H. D. Rogers, State Geologist of Penn, about 1855, widely current:

Bituminous	Volatiles, greater than 18
Semi-bituminous	" 18 to 12
Semi-anthracite	" 12 to 8
Anthracite	" less than 8

More recent schemes are those of: M. R. Campbell, based on $C + H$ ratio, as sh elementary analysis; F. G. Grout, who employs ratios based on sum of volati fixed carbon of a proximate analysis, and the elementary carbon of ultimate a and D. B. Dowling, who employs what he calls the split-volatile ratio. All the greater attention to western lignites and sub-bituminous coals, which in earlie were practically unknown. Elaborate classifications are unimportant comme The B t u 's and the physical and coking properties are the essentials.

Geological associates of coal seams are almost always shales and stones. They often have a fireclay floor; are seldom associated with stones. A seam may be broken up into benches by a "parting" of shale, "slate" by miners. A parting may increase in thickness and separate a into 2 distinct seams. Seams may be cut out by old drainage channels, contemporaneous with the old swamp, or later and long after coal was fo Pot-holes and channels in Carboniferous coals were developed even i Glacial epoch and filled with gravel. Coal seams may be pinched by th ward bulge of a relatively plastic clay floor, and may have cracks filled clay gouge.

Coal seams are subject to faults and folds; are very often in syn ("basins"), left disconnected by erosion of intervening anticlines. Folds be violent, as in middle anthracite fields of Penn, and in Belgian and F areas. Coal seams are of all thicknesses from a fraction of an inch to feet. The thickest single seam, reasonably free from thick partings, rec

erica, is at Adaville, Western Wyoming, with 86 ft of clean coal (except parting of 1 in of sandstone). A thickness of 1 ft is ordinarily considered the minimum of workability.

increase of ash (20% ash is the usual commercial maximum) coals "bony" coals, then into "bone" and into bituminous shale or slate. matter may be minutely interstratified with thin layers of relatively or be invisibly mingled. If the layers are sufficiently coarse, crushand washing may greatly reduce the ash (Sec 34 and 35). In a cross-section good seam can be recognized: bright lustrous "glance" coal and dull, "splint" coal. The proportions vary; one variety may be in great There is a third variety, porous, tender, and often showing plant called "mother of coal" or "mineral charcoal." It affords an unplace of precipitation for gypsum, pyrite, and other undesirables. phur in coal is partly in pyrite, when half passes off in burning; partly in when all passes into the ash; and partly in sulphurous hydrocarbons. ing or other combustion, roughly one-half the sulphur passes off.

geological age coal ranges practically from Carboniferous period through Tertiary. is even reported in remote pre-Cambrian strata of Finland, but no seams of are yet known older than Carboniferous. The oldest seam in America is in sandstone series, of the Mississippian (Lower Carboniferous) strata of S W Alberta and elsewhere). The really important coals begin with the PennsylIn eastern half of North America, they range up into the Permian and even in Va and N C). In the western U S, coals range from early Cretaceous Eocene. The Laramie of Cretaceous and Eocene Tertiary are most productive. The are often lignitic, especially if in relatively undisturbed strata (36).

Petroleum series. Coals are residual accumulations; petroleums are evolved, move from their sources elsewhere for storage. The petroleum series embraces gases, liquids, and solids. The chemistry is very complex, but most the members belong to the marsh-gas or paraffine series of compounds, C_nH_{2n+2}. Up to C_4H_{10} they are gases at ordinary temperatures; from $C_{16}H_{34}$ they are solids. The gases are called natural gas; the liquids, petroleum; thick, black, tarry liquids, maltha; the solids at ordinary temperatures, of leathery character, asphalt; the brittle, coal-like substances, asphaltite; natural paraffines, ozocerite. Shales impregnated with bituminous matter called oil shales. They may be rich in paraffines.

Natural gases are chiefly CH_4 but have some higher members, with a little the olefine series C_nH_{2n}, more or less of H_2S, N, O, CO_2, and others rarer.

Petroleums include the liquids at ordinary temperatures. Some are of low some high. These characters are expressed in degrees Beaumé, a scale which 10° Beaumé is sp gr 1. Others are calculated by the formula: $\frac{140}{\text{sp gr}} - 130$. Light petroleums range from 35° B up. The heavy drop below 20° B. Lighter oils give higher percentages of illuminants and are the more valuable. Heavy oils have an asphalt base. Some oils contain sulphur compounds. Malthas are much rarer than petroleums, and have different asphalts are specially employed for paving; asphaltites for varnishes, etc.

Origin of petroleums. There are two radically different views: the inorganic (now largely held) and the organic. There is some support for the theory attributing hydrocarbons to igneous sources, but most geologists favor the organic explanations. These the original plant or animal matter in the sediments, by decomposition in the rock (Hunt); or distillation from internal heat (J. S. Newberry); or, the modern view, bacterial decomposition while freshly deposited, the hydrocarbons being later squeezed out of the shales and mud-rocks, by pressure of overlying accumulations, into porous beds for storage. In the latter way marsh gas, CH_4, certainly does develop, but oils remain to be demonstrated (Sec 1, Table VIII).

Storage in the rocks is better understood. The most extensive pools are found low, anticlinal folds, in which an impervious shale rests upon a porous sandstone c stone, so as to imprison the gas and oil; which, from some source of the hydroc rise through the heavier ground-waters and are finally caught beneath the crest. retically, and sometimes actually, there is an uppermost layer of gas, a layer ben oil, and a bottom layer of water or connate brine. Usually either gas or oil rests or Other deposits favor lenticular or pod-like bodies of sandstone in shales, apparen sand-bars, or similar accumulations. Prospecting is usually guided in recent ye the anticlinal view. The axes of anticlines rise and fall, and pools are thus n tinuous along their trend. A very gentle anticline or even a slight monocline may Pools sometimes seem to lie to one side of the observed anticlinal crest.

Geologically, the oldest gases and oils are tapped from Ordovician (or Lower Si strata, of which the Trenton limestone is very productive in Ohio and Indiana. In the Paleozoic, the Silurian (or Upper Silurian), Devonian and Carboniferous are all productive in the Eastern States. A sandstone at base of the Coal Measure boniferous) is very productive in Illinois, Kansas, and Oklahoma. Cretaceous stones carry oil in Wyoming;, and Cretaceous limestones are the apparent sou Mexican petroleum, even though tapped from overlying Tertiary beds. Variou tiary horizons yield oils in different parts of the world, but the Miocene is especiall In America are the following fields: Appalachian; Lima-Indiana (western Oh Indiana); Illinois; Mid-Continental, in Kansas, Oklahoma and northern Texas; in Louisiana and Texas; Mexican, in the coastal plain or tierra caliente, west of T and further south; and California. There are many smaller areas in Colorado, Wy Alberta, and Alaska. Trinidad is productive and Venezuela very promising. A Rumania, Baku on the Caspian Sea, Dutch East Indies, and Japan are the chi ducers, but some oil has also been found in Germany (57).

Maltha is a rather unusual product, but appears at times where oils with an as base rise to surface and lose their more volatile constituents. Maltha may impe porous sandstone.

Asphalt, a further stage in the process, may accumulate in pools, c pregnate porous sandstones or limestones. Sometimes oils have risen in fi and have changed with loss of volatiles to brittle substances, suggesting Illustrations: albertite, of N B; grahamite, of West Va; uintaite and wurt of Utah. Oils with a paraffine base have left behind the natural par ozocerite, in fissures in sandstone (Sec 1, Table VIII) (58).

Graphite is the final metamorphic stage of all carbon minerals. It a sometimes in pegmatite veins, but more often impregnates sandstones, s and crystalline limestones, from which it may be separated by concentr the light components (60).

98. Miscellaneous Non-metallic Minerals

Gems are objects of [illegible] but the geological relations are very diverse. Dia [illegible] in certain rocks, notably in very basic igneous rocks, in South Africa western Arkansas. They are also here obtained from placer deposits. Sapphire be in contact zones of components of an igneous dike as at Yogo, Mont. are pegmatite minerals, but the various emeralds may be in contact zones. Turquoi [illegible] and crystalline rocks, where copper salts have circulated. No [illegible] can [illegible] as the [illegible]

Graphite (see Carbon Minerals).

Gems (see Carbon Minerals).

Gypsum (see Salines).

Mica, in [illegible] occurs as a constituent of coarsely crysta pegmatites, but [illegible] and [illegible] amber are utilized, the f produced mainly in India and N Carolina, the latter in Ontario and Quebec prov Canada. [illegible] mainly as rectangular trimmed

... larger sizes bring higher prices). Demand for pulverized mica is amply sup- ... trimmings from sheet-mica mines. (*b*) Softness; the softer the mica the ... adapted it is for electric commutator insulation. (*c*) Freedom from inclusions, ... are generally iron minerals and diminish insulating properties. (*d*) Flexibility. ... a given mica can be ascertained only by submitting samples to a dealer. ... manufacturers are the largest consumers.

... see Building Stone).

... Gas (see Carbon Minerals).

... (see Carbon Minerals).

... (mineral paints, pigments), or the bases for them, are sometimes the objects ... Practically all are minerals of iron. Limonite, deposits containing iron car- ... and red hematite, are all utilized. The crude product is usually calcined to in- ... purity of color or shade. Fine clays stained with limonite yield ochers. Barite ... and refuse slate are ground for "fillers."

... (see Carbon Minerals).

Phosphates are dug or quarried for fertilizers. They are of 2 kinds: ... crystalline apatite, which generally appears on borders of igneous intrusive ... and is especially associated with pre-Cambrian limestones, along the ... River in Ontario and Quebec. Apatite in different geological relations ... used as tailings in magnetic concentration of richly phosphatic magnetites, ... ville, N Y. (*b*) Earthy phosphates, such as fossil bones, coprolites, and ... ments of carbonate of lime in limestones by phosphate of lime.

... the seacoast of South Carolina, Tertiary beds have long been dug for fossil ... and replaced limestone nodules. More recent discoveries of Florida rock ... and of pebble phosphates in deltas in the drainage of rock-phosphate areas, ... a very low-cost product. In the Peace River, Fla, is done much dredg- ... concentration similar to that in placer gold deposits. Beds of rock phosphate ... series have been still more recently discovered and utilized in Tennessee. ... have been discovered by prospectors in N E Utah, but have been withdrawn ... by the Federal Government. These phosphates form interstratified beds, ... produced by reaction upon limestone of phosphoric acid from organic remains. ... caused by the droppings of wild-fowl in regions of slight rainfall, is now practi- ... (41).

... (fossil) (see Carbon Minerals).

Salines. When circulating ground-waters have traversed rocks contain- ... salts, and have afterwards been impounded and evaporated to dry- ... when bodies of sea water are isolated and evaporated, the dissolved ... precipitated in the inverse order of solubility. From sea water, gypsum ... first, then common salt, and then rarely the less abundant potas- ... In general, potassium-bearing final mother-liquors seem to have ... or else their very soluble precipitates were removed in next inrush of ... water.

... isolated bodies of the ocean, cut off perhaps by a barrier cast up during a storm, ... thin beds of salt and their associates have been derived. Thick beds of hun- ... feet in section are difficult to explain in this way. A substitute explanation is ... Theory" of Ochsenius. A deep estuary is assumed to be isolated by a broad ... the open sea. Evaporation on the bar leads to the passage down inner side of ... heavy concentrated brine, until salt is deposited in the estuary's depths. Thick ... also be precipitated in salt lakes, in deep depressions without outlet, yet so ... to be fed by salt-bearing streams. Great Salt Lake is an illustration. Salt ... at Petite Anse, Louisiana, and elsewhere along Gulf of Mexico, appear in co- ... chimney-like form, crossing sedimentary strata as great cylindrical masses, ... are domes or mounds, and around which the beds turn up. These are best ... by uprising salt springs and the expansive force of growing crystals, which may

have thrust the strata aside and upward. The only important beds of potassiu yet discovered are in the Stassfurt region, western Germany. Beds of Na_2SO_4 infrequent in arid regions, and less often waters or even beds charged with Na_2C sodium and calcium borates, likewise occur in a few districts (as southern Ca and western Nevada), where local drainage contains these rare salts. The soils northern Chilean desert have become charged with sodium nitrate, as have the few other arid localities (42).

Borax is obtained, by chemical treatment, mainly from 2 minerals: bo occurring in fumaroles in northern Italy, and in dry-lake deposits in desert regions of the world; and colemanite (borate of lime) occurr bedded saline deposits in southern California. The latter, averaging boric acid, are now the principal source of borax in the U S, refineries at San Francisco, Cal, and Bayonne, N J.

Sulphur. In native state, sulphur is found in 2 types of deposit: (*a*) near volcanic craters, expiring or largely dormant. Emitted as vapor, s condenses on walls of cavities and impregnates porous tuffs and breccias. possible, but not highly probable, that SO_3 and H_2S when mingled hot r deposit sulphur. (*b*) In association with gypsum in sedimentary strata, Sicily, Louisiana, and Texas. The sulphur was formerly believed to from reduction of gypsum ($CaSO_4 \cdot 2\,H_2O$), by organic matter in circ ground-waters. There is at present a disposition to refer it to minute isms now known to secrete elementary sulphur, and to favor its precipi amid deposits of decaying organic matter, at bottom of certain bodies of Gypsum is a universal associate (43).

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SECTION 3

EARTH EXCAVATION

BY

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CONSULTING CIVIL ENGINEER

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Note.—Numbers in parentheses refer to Bibliography at end of this section.

1. Clearing and Grubbing

Methods: (*a*) Grubbing; (*b*) burning; (*c*) blasting; (*d*) stump pullers.

Grubbing by hand. Tools: mattock, axe, round-pointed shovel, and long bar. Two men working on a single stump are more efficient than one man alone. If the large roots are cut in the fall and allowed to lie during the winter, the heavy frosts of the northern states will loosen the stump and materially assist the work of removal.

Burning stumps. The soil should be dug away, partly exposing the largest roots. Dry wood logs are piled about the stump, and kept burning until it and the larger roots are consumed. This method is especially adapted to rotten, hollow stumps, difficult to grub or pull. Crab-pit method consists in placing brush or kindling around the stump and covering all with clay and sods, leaving small openings for admission of air while burning. For best results, stumps should be split; done most cheaply by exploding dynamite in a ship-auger hole in the center of stump.

Blasting stumps. Tools: long-handled wood and earth auger, churn drill or pointed bar for chambering the bottom of a hole, wooden tamping bar, and blasting materials. Results of blasting depend upon: character of the roots (whether fibrous or tap), character of soil and the resistance it offers to the explosive, size and state of preservation of stump. Fresh, undecayed, fibrous-rooted stumps are harder to blast than those partly decayed or have tap roots. For tap-rooted stumps place charges against the side of the root, or in a hole bored into it. For cypress or willow stumps, or those in swampy soil, bore 3 to 6 holes, 12 to 18 in from the stump, and inclined towards it. For ordinary wood stumps, charges of 40% dynamite are given in Table 1. For green stumps, increase these charges by 1.5 to 2; for old, decayed stumps, use less than the charges given. Weaker explosive may be used for hard loam or clay, but in sand or muck 60% dynamite is better.

Table 5. Average Weights of Soils

Soil	Condition	Lb per cu ft	Lb per cu yd	Soil	Condition	Lb per cu ft	L c
Sand....	Wet	122	3 294	Clay.....	Loose, dry	70	1
Sand....	Dry	100	2 700	Clay.....	In place	116	3
Sand....	Packed	110	2 970	Clay.....	Compressed	130	3
Gravel..	Wet	125	3 375	Clayey earth	Rolled dry	110	2
Gravel..	Dry	112	3 024	Mud.....	.....	112	3

Angle of repose. The face of a mass of earth when exposed for a ti the elements assumes a NATURAL SLOPE. The angle which a face makes the horiz when not so exposed is the ANGLE OF REPOSE (Table 6).

Angle of temporary face. Factors affecting the angle at which an eart will stand when first excavated are: kind of material, condition (wet or width of face, height of face and distance below original surface, time of exp

Clay in natural state (especially if it contains some coarse sand and gravel) will vertically for a considerable time if well drained. Saturation causes slipping. Ha holds a vertical face for a long time. Coarse sand will not stand so well as clay, but than fine sand. Saturated sand slides readily; slight dampness aids in holding it i tion. A small face, or a face under small press, will stand where a large area wi In arid regions earth will hold a steep, often vertical face, of greater height and for a time than in rainy regions. Rankine gives for greatest depths of temporary, v faces: clean, dry sand and gravel, 0 ft; moist sand, or ordinary surface mold, 3 t ordinary clay, 10 to 16 ft (13).

Table 6. Slopes and Angles of Repose

Kind of earth	Slope of repose	Angle of repose	Kind of earth	Slope of repose	An re
Sand, clean, loose....	1.5 : 1	34°	Clay, wet.....	3.5 : 1	1
Sand and clay, loose..	1.33 : 1	37°	Rock, hard (riprap).....	1 : 1	4
Sand, wet.....	2.5 : 1	22°	Sand, clay, gravel (suction-dredged).....	2 : 1	2
Gravel, clean, loose...	1.33 : 1	37°			
Gravel and clay, loose	1.33 : 1	37°	River mud (suction-dredged).....	3 : 1	1
Clay, dry, loose.....	1.33 : 1	37°			
Clay, dry, natural....	1 : 1	45°	Gravel & sand on shores, exposed to waves.....	7.5 : 1	7

2. Methods and Cost of Loosening

Table 7. Cost of Plowing (wages $1.50, horse-keep $1, per 10-hr day)

Soil	Labor	Cu yd per hr	Labor ¢ per c
Loam.....	1 driver, 1 holder, 2 horses.....	50	1.(
Gravel and loam....	1 " 1 " 2 "	35	1.4
Fairly tough clay....	1 " 1 " 2 "	25	2.(
Very hard soil.....	1 " 1 " 4–6 " ..... 2 men on plow beam of rooter plow	15–20	5–(
Ordinary soil.....	1 driver, 6 horses, on gang plow.....	40	1.5

Barring down earth banks over 8 or 10 ft high, or undermining it picks, is an economical method. Very tough hardpan, with many boul may be barred down and picked into shape for shoveling at rate of 3 cu y man-hr, costing 5¢ per cu yd.

Loosening by explosives. Hard-... is economically loosened by ... of low-grade dynamite, Jud-... or contractor's powder. Holes, ... pt in high breasts, should be at ... to the vertical. Horiz holes in ... face of a bank are effective. For ... ils of chamber and coyote-hole ... g see Sec 5, Art 7.

Table 8. Cost of Pick-work (@ 15¢ per hr)

Soil	Cu yd per hr	Labor cost per cu yd
Loam	15	$0.010
Gravel and loam	8	0.021
Fairly tough clay	3	0.050
Hardpan	1.5	0.100

4. Methods and Cost of Shoveling

Output depends upon the material, height or distance it must be thrown, ... size of shovel. Very large shovels may be used for light soils.

... S = time filling shovel and straightening up to throw, min; t = time throwing 1 ...shful, min; d = distance carried, ft; w = time walking 1 ft with loaded shovel, min; w' = time returning 1 ft with empty shovel, min; L = load of 1 shovel, cu ft; P = percent-... of day required for rest and delays; T = time shoveling 1 cu yd after it is loosened, min.

Then
$$T = \left([S + t + (w + w')\,d]\,\frac{27}{L}\right)(1 + P),$$

reducing to
$$T = (S + t)\,\frac{27}{L}\,(1 + P) \text{ when no walking is done.}$$

Taylor gives the following values for the above factors:

	S, min	t, min			w and w'	L, cu ft			P*
		Aver	Min	Max		Aver	Min	Max	
...d or sandy loam	0.073	0.045	0.031	0.076	0.004	0.145	0.11	0.20	30
..., gravelly	0.092	0.045	0.031	0.076	0.004	0.134	0.10	0.19	30
...vel, medium	0.084	0.045	0.031	0.076	0.004	0.110	0.08	0.15	30

* This reduces to as low as 5 when much walking is done.

Table 9. Cost of Loading by Shoveling

Method	Cu yd per man-hr	Cost per cu yd, ¢ (Wages, 15¢ per hr)	Authority
...d into wheelbarrows	0.8	19	M. Ancelin
...avel " "	1.7–2.7	7	"
...arth " "	1.6–4.8	5	"
" " " , aver	2.2	7	"
" " "	2.8	5.25	Gillespie
...arth (all kinds) into wagons	2.1	7.5	Cole (*a*)
"	2.0	7.5	D. K. Clark
...and into cars from high face	1.8	8.25	Gillette (*b*)
...ed gravelly soil into wagons	1.3	11.3	" (*c*)
...soil	1.5–2.0	8.5	J. M. Brown
"	2.8	(*d*)	"
...and gravel into carts	1.0	15	E. Morris
... into carts	1.2	12.5	"
...dy earth into carts	1.4	10.75	"
...w sand into carts	2.0	7.5	G. A. Parker
..., tenacious, Chicago	1.25	(*e*)	"
...rgan into low dump cars	1.5	10.00	Gillette
...e earth " " "	1.75	8.6	"

... 22 miles, Erie Canal. (*b*) 10 000 cu yd bank measurement. (*c*) 20 000 cu yd in ...tenement. (*d*) A rush job. (*e*) Spaded out and handled with forks.

Table 10. Cost of Picking and Loading (1)

Soil	Cu yd per man-hr	Cost per cu yd, ¢ (wages 15¢ per hr)	Soil	Cu yd per man-hr	Co cu (wa pe
Hardpan....	2.66	40	Ordinary clay, gravel, loam	1.00	1
Tough clay..	1.33	20	Light sandy or loamy soil..	0.80	1

5. Methods and Cost of Hauling

Speed of horses. A 2-horse team can travel 20 miles (10 loaded and 10 in 10 hr, over fairly hard earth roads, 15 miles over poor roads, and 25 mil good macadam, gravel, or paved streets. This gives in ft per min: poor 132 ft; fair earth roads, 176 ft; best roads, 220 ft; aver speed of 220 ft p may be expected of teams when actually walking (not including delays and

Work of horses. A good aver horse exerts a pull of about 0.1 his own when working constantly for 10 hr. For a short time he can pull about weight, at 220 ft per min (= 2.5 miles per hr). A 1500-lb horse will do 550 ft-lb work per sec, or the commercial horse-power. Two horses w pull twice as much as one, and the exact proportionate work of 2 or more as compared to that of one, is unknown; it is commonly assumed that a 2 team will pull 190% as much as 1 horse; 3 horses, 255%; 4 horses, 320%

Effect of up-grades upon size of load that can be hauled by a team definitely known, authorities differing.

Prof. E. B. McCormick found that increase of grade decreases the load that can be in each of 3 ways; (*a*) required pull per ton is increased; (*b*) possible pull of the decreased by the effort required for him to raise his own weight up the grade; (*c*) tive pull is reduced by a change in the angle of application of the pull. The first are unavoidable, but the third can be nearly, if not quite, eliminated by changing th of hitching. McCormick assumes that on grades the team's tractive effort decrease amount equal to the grade resistance due to wt of team, as calculated by the form

$$X = \frac{W}{3} - \frac{W}{100}G;$$

where W = weight of team; X = tractive effort of team; G = percentage of grad

Table 11. Loads which 1 Horse can Haul on a Grade (in terms of percent load on level, when exerting a uniform pull equal to 0.1 his weight) (Ira O. Bak

Grade	Kind of pavement						
	Iron rails	Sheet asphalt	Broken stone	Stone blocks	Best earth	Spongy earth	Mu ea
Level	100	100	100	100	100	100	1
1%	30	45	56	60	62	75	9
2%	16	27	36	40	50	57	6
3%	10	18	25	28	37	44	5
4%	7	12	17	20	27	33	4
5%	4	8	12	14	20	25	3
6%	3	6	8	10	14	18	2
7%	2	4	6	6	10	12	1
8%	1	2	4	4	6	8	1
9%	1	1	2	2	3	4	
10%	0	0	0	0	0	0	

Loads given in Table 11 are not the max loads that a team can pull. If a tea pull 2 000 lb on a very poor road for a whole day, and can exert 4 times its aver for a short time, it can easily pull the same load up a 5% grade, provided the climb is

Table 12. Tractive Resistance per Ton of 2 000 lb (E. B. McCormick)

Road surface	Tractive resistance, lb per ton	Road surface	Tractive resistance, lb per ton
[illegible] packed and dry	100	Macadam, very good	38
[illegible]sty	106	" average	46
[illegible]ddy	190	Sheet asphalt	38
[illegible]ose	320	Asphaltic concrete	40
[illegible]d, good, compact	51	Wood block, good	33
[illegible]oose	147	" " poor	52
[illegible]s, well-packed	92	Cobble stone	54
[illegible] road, dry	61	Granite tramway	27

Table 13. Loads Hauled by 2-Horse Team (1)

Road surface	Short tons	Earth, cu yd	Road surface	Short tons	Earth, cu yd
[illegible] poor earth road	1.0	0.8	Good hard earth road	2.0	1.6
[illegible]r earth road	1.25	1.0	Good clean macadam	3.0	2.4

[illegible] distances, or over first-class pavement, much larger loads may be hauled than [illegible] Table 13.

Formula for Transportation (11): Let E = total expense per day, ct; w = net load [illegible] cu yd; s = aver speed when loaded, ft per min; ks = aver speed when empty, ft [illegible], D = length of haul in ft, measured one way; l = time lost while loading, resting, [illegible]ing, turning, and miscellaneous, per round trip, min; W = number of minutes in work[illegible]day; C = cost per lb or cu yd, ct; L = a constant representing the cost of loading, ct.

Then $\frac{D}{s}$ = time for loaded trip, min. $\frac{D}{ks}$ = time for empty trip, min.

$l+\frac{D}{s}\left(1+\frac{1}{k}\right)$ = aver time for one round trip.

$W \div \left[l+\frac{D}{s}\left(1+\frac{1}{k}\right)\right]$ = aver number of trips per day.

$Ww \div \left[l+\frac{D}{s}\left(1+\frac{1}{k}\right)\right]$ = aver total amount transported per day.

$$C = E\,\frac{\frac{D}{s}\left(1+\frac{1}{k}\right)+l}{Ww} = \text{cost of transportation, per lb or cu yd.}$$

$$C = E\,\frac{\frac{D}{s}\left(1+\frac{1}{k}\right)+l}{Ww} + L = \text{cost of picking, loading, and transporting.}$$

"Average earth" costs. The costs in following six paragraphs are for earth [illegible]erate toughness, in which a 2-horse team can plow 35 cu yd per hr (Table 7).

Wheelbarrow work. Wheelbarrows hold from 0.07 to 0.12 cu yd (175 to [illegible] lb, the aver load being about 0.1 cu yd of earth (250 lb) for work on level [illegible]runways, and about 0.07 cu yd on steep grades. One man will wheel a barrow [illegible] 15 miles per day. While dumping, about 0.25 min per trip is lost, or [illegible] min per cu yd, when barrows hold ⅛ cu yd. About the same time is [illegible] while changing barrows and in delays. With wages at 15¢ per man-hr, [illegible]st of picking, loading, and lost time is 17¢ per cu yd. Cost of hauling is [illegible]¢ per cu yd per 100-ft haul for level runways, and 5¢ per cu yd per 100-ft [illegible] on steep grades.

Cost of wheelbarrow work (wages, 15¢ per hr): To a fixed cost of cu yd add 5¢ per cu yd per 100-ft haul, on steep grades, and 3.5¢ on le ways (1).

Cart work. One-horse, two-wheeled, dump carts hold 0.3 to 0.5 On ordinary road loads of earth seldom exceed 0.4 cu yd (place measu loads of 0.48 cu yd gravel and 0.52 cu yd sand are common, and 0.59 cu 0.63 cu yd may be carried. Double carts and wagons hold twice as m single, the loads being limited by condition of the road.

With hauls of 300 ft or less, 1 driver can attend 2 carts by taking one to the dun the other is being loaded; on long hauls, and where the relative positions of pit an are fairly constant, horses may be trained to travel without a driver.

Cost of cart work. A cart can be loaded with 0.4 cu yd in 3 min by of 4 or 5 men, and should be dumped in 1 min. If cost of a horse and $1 per 10-hr day, and the driver's wages $1.50, then the cost of a cart a a driver is $1.75 per day; hence, cost of lost time is 3¢ per cu yd. Co dumpman is 1¢ per cu yd, if 150 cu yd are delivered to him in 10 hr. (picking and loading is about 15¢ per cu yd. To determine total cost, ad per cu yd per 100-ft haul to the fixed cost of 19¢ per cu yd, as above. Fo easily plowed and shoveled, the fixed cost may reduce to 14¢ per cu yd. a driver for each cart, the cost of hauling is 1¢ per cu yd per 100-ft ha carts carry 0.67 cu yd the cost of hauling is 0.5 ct per cu yd (1).

Wagon work. There are 3 types of 4-wheeled wagons in common us tom-dump; end or rear-dump or double cart; and slat-bottom. The las is not now often used, and the end-dump wagon is employed chiefly wh is dumped from piers into scows. Double carts and wagons hold about as much as single, their loads being limited by the condition of the road 13). Two-horse teams are the rule, but 3 to 6 horses are often very econo as almost 50% greater load can be hauled by 3 horses and a proportio greater load by a greater number.

Cost of wagon transport depends upon time lost in loading and dur and upon speed of the horses. Loading time should not exceed 4 or Having extra wagons to be loaded while the teams are traveling to and the dump materially reduces lost time. A team can be changed fro wagon to another in 1 to 1.5 min. Dumping time: 0 to 0.5 min for bo dump, 1 min for end-dump, and 1.5 to 3 min for slat-bottom wagon (1). suming cost of loading earth at 13¢ per cu yd, and cost of lost time for a and driver at 5¢ per cu yd (with wages of team and driver at 35¢ per h fixed cost of wagon transport is 18¢ per cu yd.

At speed of 2.5 miles per hr, the cost of wagon work per cu yd is found by addi following rates per 100-ft haul per cu yd to a fixed cost of 18¢: For 0.8 cu yd loa 0.66¢; 1.0 cu yd, 0.53¢; 1.6 cu yd, 0.33¢; 2.0 cu yd, 0.26¢; 2.4 cu yd, 0.22¢.

Cost for traction engines and large wagons. For 3-yd wagons hau trains (11), use the following factors in the transportation formulas given a

E = total expense per day, = 2 100 cents, made up as follows: Engineman, $3; fir $2; coal for 10 miles (1.25 ton @ $2.25), $2.82; repairs (traction engine and 12 wa $4.30; deprec (traction engine and 12 wagons), $3.53; int (traction engine and 12 wa $1.72; misc and superintendence, $3.63; total, $21.

w = 25 tons, plus or minus (3 to 8 wagons of 3 cu yd or 4 to 5 tons each).

s = 242 ft; $\frac{s}{t}$ = 266 ft per min.

t = 30 min (including getting coal and water). W = 600 min.

L depends on method of loading.

(Interest is based on working season of 7 months, of 25 working days, or 175 wo days per year.)

...st with automobile truck (11).

... = wt of live load, ton; m = dead load of vehicle, ton; s = speed loaded, ft per ... = speed empty, ft per min; D = distance of haul, ft (one way); l = lost time per ..., F = fixed charges per working day, as: int, insurance, taxes, deprec, storage. ...portion of fixed annual charges should be based on number of actual working ...r year, with due allowance for bad weather, break downs, etc); O = operating ...es per working day, cents, as: fuel, waste, oil, chauffeur and other labor, repairs; ...mber of minutes in working day; R = transportation cost per ton, cents; N = ... round trips per day of W min.

... = time for loaded trip, min. $D + ks$ = time for empty trip, min.

$l + (D + ks)$ = actual non-productive time per trip, min.

$\frac{1}{s}\left(1 + \frac{1}{k}\right)$ = total time per trip, min.

$W \div \left[l + \frac{D}{s}\left(1 + \frac{1}{k}\right)\right]$ = number of trips per day.

$Ww \div l - \left[\frac{D}{s}\left(1 + \frac{1}{k}\right)\right]$ = aver amount transported per day.

$O + F \div \left[l + \frac{D}{s}\left(1 + \frac{1}{k}\right)\right] \div Ww = R$ = cost of transport per ton for distance D.

$w \div (m + w)$ = carrying efficiency of vehicle, percentage.

...ctors and bottom-dump wagons. Trailer wagons drawn by auto- ... trucks or tractors are economical for long hauls. An auto truck can ... its rated load, and under proper conditions haul in trailers excess loads ... to 300%. Tractors are usually 3 or 4-wheeled automobiles, with a ... fifth wheel and king bolt for hauling a 2-wheeled trailer. Trailers are ... dump type, of 3-ton (76 cu ft) and 5-ton (120 cu ft) capacities. They ... time lost by the tractor when loading and dumping; front of the trailer ... jacked up and its body loaded while the tractor is hauling another ...

... vantages of tractor and trailer over ordinary type of auto-truck are: simplicity of ... and consequent reduction in first and maintenance costs; reduction in tire ... maintenance, due to use of springs; lessened shocks in starting and stopping; ... of parts. Disadvantages: difficulty of turning in small space; can not back ... distance, tractive power is insufficient on very bad roads.

...ars. There are 6 methods of moving cars in excavation work: by hand; ... or mule; by locomotive; by cable; by self-contained motor and electric ... by gravity, combined with one of the other methods.

... of 6 classes: small ½ to 2-yd light, steel, narrow-gage, side or end-dump cars, ... moved by hand or horse; small 3 to 8-yd wooden, narrow-gage, side-dump cars, ... pulled by contractor's locomotive; standard gage, wooden or steel-side dump ... to 12-yd capacity, generally pulled by large contractor's locomotive; standard- ... cars and standard locomotives; small, narrow-gage side or end-dump cars, ... mules, cable or small air-electric industrial locomotives; special quarry or in- ... cars, equipped with motors.

...ransportation by small cars. Double side-dump, V-shaped, steel cars, ... to 54 cu ft, weigh 700 to 1 800 lb, and cost $50 to $100 each. When the ... excavated lies higher than the dump, these cars traveling under grav- ... pulled back by horses or cable are economical.

... ...ation of a 14-ft bank of hardpan was completed in 1907 with an outfit of this ... Plant: 12 36-cu ft cars, 600 ft of 36-in gage track with turnout and switches, ... $1 345. Labor cost per day: 1 foreman, $3; 6 pick and bar men, @ $1.50, $9; ... men, @ $1.50, $18; 1 horse and driver, $3.50; 2 dump and track men, @ $1.50, ... mouth (proportion), $1.75; Total, $38.25. The soil was undermined by picks

and bars, loaded into 6-car trains, and run by gravity down a 4% grade to the The empty cars were pulled by 1 horse. 198 cars (about the same number of cu y measure) were dumped in 10 hr, giving a labor cost of 19.3¢ per cu yd.

Cars and locomotives. Dump cars or standard railway cars are ge used for steam-shovel work. Common sizes, for contractor's locomoti "dinkeys," are 4, 6, or 8-yd; track gage, usually 3 ft. Track is common with 30 to 50-lb rail, few ties, and unballasted. ROLLING RESISTANCE on such roughly laid rails is far greater than on straight, well-ballasted R R while commonly assumed as 15 to 40 lb per ton, it is seldom less than 40 often 50 lb or more. STARTING RESISTANCE of dirt cars on rough track is ably double the rolling resistance.

R. V. Norris (*Trans* Am Inst Min Engs, Vol 18, p 508) gives result of several h traction tests on mine cars. To start empty and loaded cars required a pull of 8 lb per ton; to keep them moving, 26 to 66 lb per ton. A. W. Wright, from tests on drawn street cars in Chicago, foun the pull to start cars on dirty rails av 134.5 lb per ton, and 32.3 lb to keep t motion. C. E. Emory, from experim N Y, found a resistance of 11.8 lb. Tl probably on clean center-bearing rail, P. Holt found 15 to 46 lb per ton on straight track. H. Hughes found 26 lb, and M. Treaca, 22.4 lb.

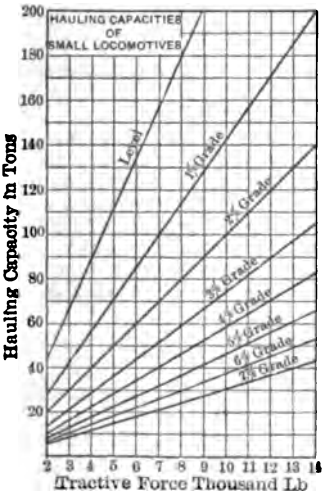


Fig 1. Hauling Capacity of Locomotive on Grades

Rule for calculating hauling c ity (H. K. Porter). Divide tractive of locomotive by resistance per ton gravity and rolling friction; then d wt of locomotive (and tender, if This gives the wt in tons (includi of cars) that the locomotive can Tractive force of a locomotive: $(T = + D$, where T = tractive force, lb; d = of one piston, in; L = length of strok P = press of steam in cylinders, lb D = diam of driving wheels, in

Hauling capacity of construction lo tives is given in Fig 1. Rolling fricti assumed at 40 lb per ton; resistance d gravity is approx 20 lb per ton for eac cent of grade.

Struck- or water-measure capacities and weights of dump cars are g in Table 14. Actual capacities, earth measured "in place," is 75 to 80 the water-measure capacity. See also Table 3, p 119.

Table 14. Dump Cars, Capacities and Prices

Water measure, cu yd...	0.67*	1.0*	1.5*	2.0*	4†	6†	8†	1
Weight of car, lb.....	900	950	1 350	1 700	6 000	11 000	16 500	28
Price, $.....	54	58	70	100	180	300	450	

* Steel: V-shaped, double-side dump cars. † Wood: diamond-frame, double-side dump

Speed of "dinkey" locomotives is from 5 miles per hr when hauling lo to 9 miles per hr empty or on a down grade. Grades should never exceed **if avoidable.**

... of hauling cars depends mainly upon output of the steam shovel loading ... On hauls up to 1 000 ft 1 dinkey can often serve a 1.5-yd shovel. About ... min is required for 1 man to dump and clear a 3-yd car. Under favor- ... conditions, 1.5 to 2 min are lost in switching a train.

... of contractor's track depends upon the prices of rails, ties, and fastenings, and the ... and efficiency of the men employed. Weight of rail for contracting work is usually ... per yd, although 16 and 20-lb rail is common in mines and quarries where ... semi-permanent. For 3-ft gage track, a 6 by 6-in by 5-ft tie is best. About 375 ft ... these ties, and 1 ton of 30-lb rail, are required per 100 ft. Cost of laying construc- ... after grading road bed and delivering the material, is from $2 to $6 per 100 ft.

... of daily operation of 2 dinkeys, assuming 132 working days per year (2):

... men (on 2 dinkeys) @ $3; 2 trainmen @ $2; 6 dumpmen @ $1.75	$20.50
... for 2 dinkeys, 0.6 ton @ $4	2.40
... at $1.50; oil and waste, $0.50	2.00
... on $6 000 (2 dinkeys, 24 cars), @ 6% per year + 132 days	3.65
... on $6 000, @ 1.5% per month + 22 days	5.45
... on $8 000 @ 8% per year + 132 days	4.85
... & housing locomotives & cars once during year, say $500 + 132 days	4.00
Total charges: train crew, fuel, repairs, care of locomotives and cars	$42.85
... grading and track shifting @ $1.75	$10.50
... on $2 250 (35-lb rails for 1 mile of track), @ 6% + 132 days	1.00
... on $2 250, @ 12% + 132 days	2.00
... on $750 (ties, at 30¢, 2 miles track), @ 6% + 132 days	0.35
... on $750 (ties), @ 10% per month + 22 days	3.50
Total track crew and track	$17.35

Hauling engine and cable are often used to pull cars on grades when the ... not over 1 000 ft or less. Cars will coast down a grade of about 3%, and may ... hauled back by cable (Sec 11).

6. Scraper Work

Drag or slip scraper is an open-front steel box, or scoop, usually provided with run- ... a false sheet of steel, to take wear. Wooden handles at the rear are for loading and ... it. The bail, for the whiffle-trees, is pivoted at the sides near the front. Two ... are usually used. Common sizes of drag scrapers, No 1, 2, 3, are listed as holding ... 1 cu ft respectively. Actual capacity (place measure) is about 0.5 list capacity. ... $4; to $5; wt, 75 to 90 lb each.

Cost of work. Drag scrapers are economical only for short hauls, or where ... excavation is in confined places. The minimum "run-around" or total ... for any kind of scraper work is about 150 ft. LEAD is the aver straight ... between center of cut and center of fill; it is shorter than the actual ... by the distance required for turning and handling the team. About 0.33 ... min per cu yd is lost while loading and dumping. With wages of team ... driver at 35¢ per hr, cost of loading and dumping is about 2.75¢ per cu yd, ... turning 1.5¢, and for lost time, turning, etc, 2¢, making total fixed cost in ... of 6.25¢ per cu yd. To this add 4.5¢ per cu yd per 100 ft of lead for a ... scraper. Cost of foreman's wages adds about 0.75¢, and wear and tear ... scrapers about 0.25¢ per cu yd. Work in stiff clay may cost 30% more (1).

Buck and Fresno scrapers. Simplest form of buck scraper is an upright board, 8 ft ... with a tongue for the team and a rear platform for the driver. From this the ... graders and levelers have been developed. Fresno scraper is a long pan, open ... top, and without a pole. It is handled like a drag scraper, except that 4 horses ... instead of 2 are used. Common sizes are No 1, 2, 3, having a listed capacity of ... and 12 cu ft, and an actual aver capacity of 9 54, 7.42, and 6.36 cu ft, respectively. ... $5 to $6; wt, 250 to 300 lb.

Cost of work of 4-horse Fresno scraper, holding 0.5 cu yd, with
driver and horses at $6 per day, is found as follows: To a fixed cost
cu yd, add 1.75¢ per 100 ft of lead. Fixed cost includes extra distance
while loading and dumping, shifting of the gang, and 1¢ for plowing.
is 9 cu ft, the fixed cost is 5¢ per cu yd and the traveling charge per 10
is 2.67¢ per cu yd. To this add 0.5 to 1¢ per cu yd for foreman's wages.
scrapers are usually more economical than Fresnos for hauls exceeding

Wheel scraper is a steel pan hung hinged between 2 wheels, with a pole for
and a lever in the rear for lowering the pan when loading and for upsetting it wh
ing. Sizes are No 1, 2, 2½, and 3, with listed capacities of 9, 12, 14, and 16 c
actual aver capacities of 8, 8.75, 12 and 15.5 cu ft, respectively. Costs, $2
wt, 330 to 750 lb.

Cost of work with wheel scrapers is found by adding a traveling
100 ft of lead to a fixed cost per cu yd. Costs in Table 15 are based o
@ 15¢ per hr for men, or 35¢ for team and driver (2).

Table 15. Costs of Wheel-scraper Work

Number or size of scraper	1	2
Load hauled, cu yd	0.2	0.25
Loading and dumping cost, ¢ per cu yd	1.5	1.2
Man for loading, ¢ per cu yd	0.8	0.8
Plowing, " " " "	1.5	1.5
Extra team travel, " " " "	1.5	1.2
Snatch team, " " " "	.....	1.5
Man for dumping " " " "	.....	.....
Total fixed cost, ¢ per cu yd	5.3	6.2
Traveling cost per 100 ft lead, ¢ per cu yd	2.75	2.2

Cost of foreman and repairs adds about 1¢ per cu yd.

Maney 4-wheeled scrapers, hauled by 2 or 4 horses or a traction engine, ar
tially self-loading 4-wheel wagons. Loading and dumping are controlled by s
and chains from the rear wheels. Capacity is 31 cu ft struck measure, but th
measure capacity is little more than 50% of this. Cost of field labor and team is
per cu yd, for 250 ft lead.

Levelers and graders. TONGUE SCRAPER is a wooden platform drawn at an
60° with the surface of the ground by 2 horses. It is economical for spreading
earth, and filling shallow ditches; cost, about $7. ROAD SCRAPERS and DRAGS, for
ing ruts in roads, cost $12 to $20. Standard ROAD MACHINES weigh 1 500 to 2
cost, $125 to $260. They are drawn by 2 or more (generally 4 or 6) horses or by
tion engine, and are used for spreading earth on embankments, grading roads, a
ging shallow ditches. Small, 2-horse, 1-man road machines, of several types, we
to 900 lb and cost $50 to $150. They are economical for light grading and spre

Elevating graders for loading wagons. Best conditions: ample room for turni
maneuvering the teams, and ground fairly level, with few boulders or roots. The m
is a 4-wheeled truck, carrying a heavy plow which turns its furrows upon a lateral, u
sloping belt conveyor, whence the material is discharged into wagons or falls
ground. It is propelled by a traction engine, or 10 to 12 horses (8 in front and 4 b
The conveyor is driven by gearing, or a gasoline or steam engine. Cost, $1 000 to
weight, 5 to 9 tons. Capacity under favorable conditions, 400 to 1 000 cu yd in

Cost of grader work, at an output of 500 cu yd per 10 hr, is as follows
wages at $1.50 for driver, and $1 for a horse, the fixed cost of 10 horse
4 men on the grader is 3.5¢ per cu yd; of 5 men spreading on the dump,
and of int, repairs, and deprec, $5 per day, or 1¢ per cu yd. To these is add
cost of the waste labor of the teams following the grader to receive their lo

...ges of a driver and 3 horses on a dump wagon holding 1.25 cu yd are $4.50 per ...day. the cost of hauling is 0.6¢ per cu yd per 100 ft of lead, or 2.5¢ per cu yd for the ... 100 ft of waste lead traveled by the wagons. Hence, to find the cost for a given ... 0.6¢ per cu yd per 100-ft lead to a fixed cost of 8.5¢. With a traction engine ...ixed cost is reduced by 0.8¢ per yd. If water for the engine must be pumped and ...cost may be increased 0.5¢ or more per yd.

...son or power scrapers vary in shape from a horse-drawn drag scraper to a bot... box-like affair.

Bottomless power scraper (Fig 2) holds from 0.5 to 7 cu yd and costs $250 to $600. ...able cutter edge is fastened on a runner frame adjustably connected to the back-... This machine is especially economical in sticky material, as the load is not dumped

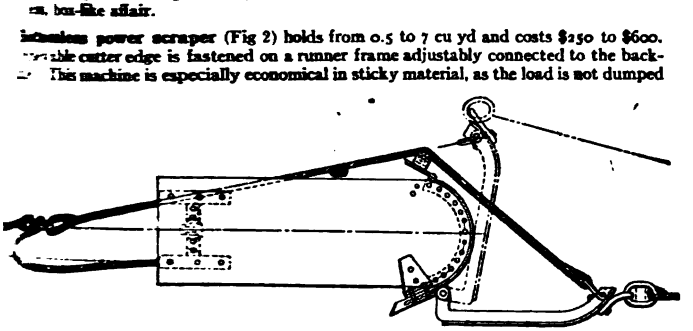


Fig 2. Bagley Grader (Bottomless)

... load rests on the ground and is dragged to the point of deposition. It requires a ...-hp engine, ⅞ to 1-in haul-back line and 1 to 1.25-in pulling line. Approx oper... cost is $28 per day. With a 600-ft haul, about 15 trips per hr should be made; ... 300-ft haul, about 6 trips. An output of 500 cu yd per 9-hr day can be averaged ...favorable conditions.

Drag-line scraper excavators (Fig 3) are operated by locomotive cranes, field ...rs or derricks, with cables, or by dredges. Fig 4 shows a drag-line excavator

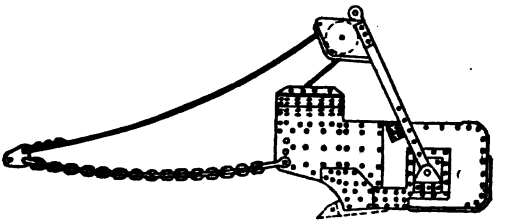


Fig 3. Page Drag-line Scraper

... boom derrick type. Scraper buckets are of various shapes, so arranged ...lt forward or backward in dumping. They hold ¾ to 5 cu yd, and cost ... $500 per cu yd of capacity, not including rigging nor power plant.

...st of operation of drag-line excavators varies with type of bucket, operating ...ng, kind of earth, and local conditions. Sauerman excavator, wherein the ...is drawn in one direction by a pulling line, and allowed to slide back by gravity ...ad cableway (Art 7), will, under favorable conditions, excavate and deliver earth ...ost of 300 ft for 5.5 to 10¢ per cu yd. A scraper excavator machine of the full ...ng or "whirly" type, has an upper platform rotating on a lower frame. The ...gine and long boom for operating the scraper are carried on the platform. These ...ines cost from $5 000 to $10 000 complete. Cost of operating in well compacted ... and gravely soil is 5 to 7¢ per cu yd (12).

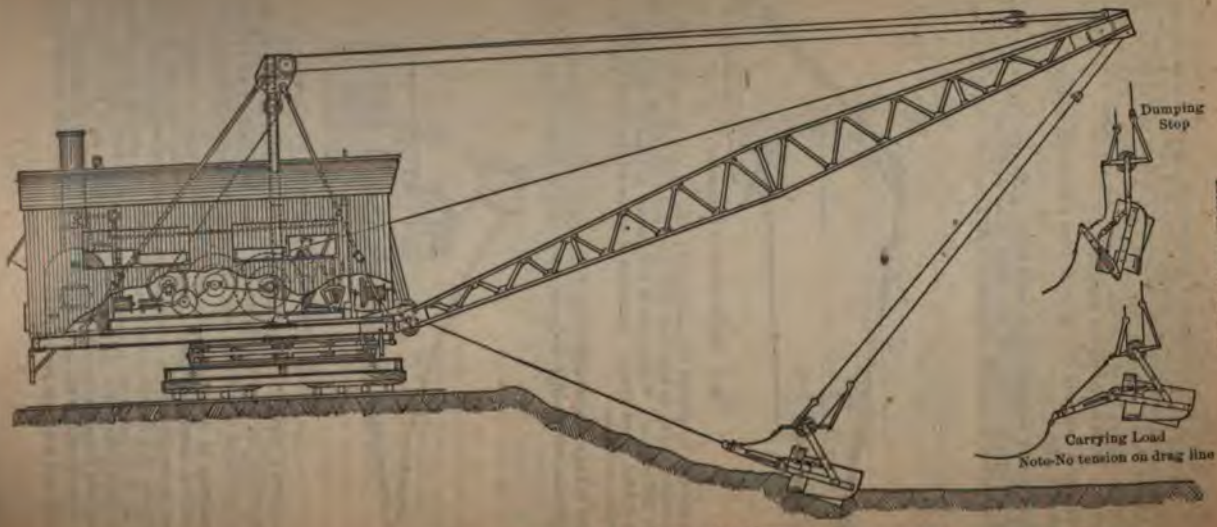


Fig 4. Boom-derrick Type of Drag-line Scraper

7. Cableways and Belt Conveyors

ays. For design and construction, see Sec 26. When the track cable is
d that the sag may be varied, at will of the operator, thereby directly
raising the load, the machine is a SLACK CABLEWAY. For operating a
avator bucket, one end of cableway is set higher than the other, and
oaded or empty bucket moves by gravity. When loads are all carried
ction it often pays to have the dump end lower than loading end.

e use of cableways requires: (*a*) work on a scale large enough to warrant
rst installation, plus costs of ensuing removals and reinstallations; (*b*) enough
the length of span, and within economic reaching distance each side of the
o repay cost of one installation and removal. These conditions are often
h and canal excavation and dam construction.

pends on length of span, height and type of towers, and amount and
er required. In general, cableways for operating, digging, or convey-
cost, installed complete, from $8 to $15 per ft of span, for spans of 400
d from $6 to $12 per ft for spans of 1 000 to 2 000 ft. Duplex cable-
2 complete cables 15 to 20 ft apart on common towers) cost about $12
pans of 2 000 ft, with towers 75 to 130 ft high (see also Sec 26).

operation (see Sec 5, Art 8, 11; Sec 26; and Bibliog 1, 2, 7, 8, 9, 11).

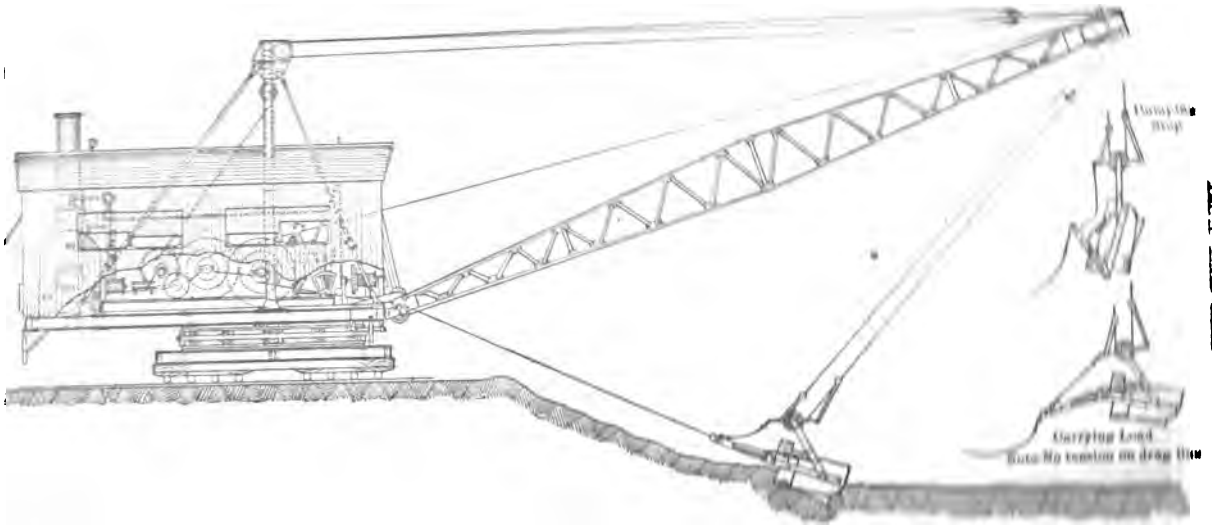
nveyors. For design and construction, see Sec 27. In earth exca-
nveyors find their most important employment as accessories to
hines, as dredges, trench excavators, and elevating graders. But
metimes used in long lengths for excavating cellars, and transport-
l from boats (1, 2, 7, 8, 9, 11).

. A cellar excavation for a power house was made in New York (1899) by
r. A trench 7 ft deep was first excavated; across it were erected 3 bridges
above the conveyor, and the earth was moved to the hatches by wheel
With a 30-in belt the daily output exceeded 1 200 cu yd (7). In 1908 a flat
r 2 100 ft long was built in the South for conveying shale from a quarry to a
. The belt was 24-in, 8-ply, in 2 sections, one being 1 000 ft between centers,
00 ft. It was carried by rollers, the top rollers being 4 ft and the return idlers
Guide rollers were placed 40 ft apart. Contraction and expansion were
tension carriages. Since all loads were carried down grade, more power was
operate the belt empty than when loaded. Power was furnished by two
es. Cost of installation was $9 106, or $4.37 per lin ft. Cost of operation
nance (including labor, power, supplies, int, deprec, and renewals, estimated
cost) was $4.04 per 10-hr day, or 2¢ per ton. Cost per unit of operation is
a capacity of 200 tons per 10 hr (E. C. Soper, Am Soc Mech Engs).

8. Steam-shovel Work

ovels. USUAL DESIGN. The dipper and its arm or handle move in a verti-
pported by a derrick and boom, rotating horizontally. By a rack and pinion,
rice, the dipper arm runs in and out with respect to the boom, so that a load
and dumped at any point within the reach of the machine. SPECIAL DESIGNS.
rican milway ditcher, the dipper arm is hinged to the boom, the cutting
adjusted for each case. In the Thew shovel, the length of dipper arm is
required by bolts, and its inner end is attached to a carriage traveling back
orizontally on the boom. In a broad sense, the term steam shovel includes
land dredges, operating clam-shell and orange-peel buckets.

vels usually run on standard-gage track; lighter machines on narrow-gage
ounted on a truck with broad-tire wheels for road traction. For excavat-
they may be mounted on rollers on a platform spanning the trench. Land
ally run on rollers or on very broad-gage wheels, or, in marshy ground, on
wheels." Ditchers, for R R work, usually travel on track laid on the
flat cars being loaded. (1, 2, 4, 5, 7, 8, 9, 10.)



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... depends on length of span, height and type of towers, and amount and ... required. In general, cableways for operating, digging, or convey- ... cost, installed complete, from $8 to $15 per ft of span, for spans of 400 ... and from $6 to $12 per ft for spans of 1 000 to 2 000 ft. Duplex cable- ... 2 complete cables 15 to 20 ft apart on common towers) cost about $12 ... spans of 2 000 ft, with towers 75 to 130 ft high (see also Sec 26).

... of operation (see Sec 5, Art 8, 11; Sec 26; and Bibliog 1, 2, 7, 8, 9, 11).

... conveyors. For design and construction, see Sec 27. In earth exca- ... conveyors find their most important employment as accessories to ... ines, as dredges, trench excavators, and elevating graders. But ... metimes used in long lengths for excavating cellars, and transport- ... rial from boats (1, 2, 7, 8, 9, 11).

...ples. A cellar excavation for a power house was made in New York (1899) by ... A trench 7 ft deep was first excavated; across it were erected 3 bridges ... above the conveyor, and the earth was moved to the hatches by wheel ... with a 30-in belt the daily output exceeded 1 200 cu yd (7). In 1908 a flat ... 2 100 ft long was built in the South for conveying shale from a quarry to a ... The belt was 24-in, 8-ply, in 2 sections, one being 1 000 ft between centers, ... 100 ft. It was carried by rollers, the top rollers being 4 ft and the return idlers ... Guide rollers were placed 40 ft apart. Contraction and expansion were ... tension carriages. Since all loads were carried down grade, more power was ... operate the belt empty than when loaded. Power was furnished by two ... Cost of installation was $9 106, or $4.37 per lin ft. Cost of operation ... including labor, power, supplies, int, deprec, and renewals, estimated ... cost was $4.04 per 10-hr day, or 2¢ per ton. Cost per unit of operation is ... a capacity of 200 tons per 10 hr (E. C. Soper, Am Soc Mech Engs).

8. Steam-shovel Work

...shovels. USUAL DESIGN. The dipper and its arm or handle move in a verti- ... mounted by a derrick and boom, rotating horizontally. By a rack and pinion, ... the dipper arm runs in and out with respect to the boom, so that a load ... and dumped at any point within the reach of the machine. SPECIAL DESIGNS. ... American railway ditcher, the dipper arm is hinged to the boom, the cutting ... adjusted for each case. In the Thew shovel, the length of dipper arm is ... secured by bolts, and its inner end is attached to a carriage traveling back ... vertically on the boom. In a broad sense, the term steam shovel includes ... and dredges, operating clam-shell and orange-peel buckets.

... els usually run on standard-gage track; lighter machines on narrow-gage ... mounted on a truck with broad-tire wheels for road traction. For excavat- ... they may be mounted on rollers on a platform spanning the trench. Land ... ly run on rollers or on very broad-gage wheels, or, in marshy ground, on ... wheels." Ditchers, for R R work, usually travel on track laid on the ... flat cars being loaded. (1, 2, 4, 5, 7, 8, 9, 10.)

9. Dredging

Types of dredges: dipper; grapple or grab-bucket; ladder or bucket-elevat
draulic or suction. Those having bunkers or hoppers for carrying the dredged m
are "hopper dredges" (1, 2, 6, 8, 9, 11).

Dipper dredge is essentially a steam shovel mounted on a scow. There are 3 clas
drainage and irrigation ditches; for deep water and harbor improvements; and fo
work. Ditching dredges are small, with narrow hulls and telescopic bank spuds
dredges have narrow hulls and side floats; deep-water dredges, for depths to 50
generally of large size, with spuds operated by independent engines. Wooden h
common, but steel hulls are now favored.

Grapple dredge is a floating derrick, with clam-shell, orange-peel, or other
grab bucket. It serves for very deep water or in confined places.

Ladder dredge, popular abroad, is especially adapted to gold placer mining
construction and use, see Sec 10.)

Hydraulic dredge is a scow, carrying a centrifugal pump, with a suction pipe r
to the bottom to be excavated, and a discharge pipe to place of deposit. In all exc
easiest materials, a revolving cutter at the mouth of the suction pipe is requi
loosening the soil. Special advantage of this dredge is its ability to convey ex
material long distances.

10. Hydraulic Excavation

This method originated in California for excavating gold-bearing gravel. I
DRAULIC MINING, pipe, monitors, ground sluices, etc, see Sec 10. Hydraulic m
are here considered for moving material in ordinary excavation, as for hydra
dams, railway embankments, and ordinary grading (1, 3, 11).

Hydraulic-fill dams. With enough water, a sufficiently high working fa
grade to convey the mixed earth and water, dams can be built more cheaply
well) by hydraulicking than by ordinary methods of embankment with rolli
tamping. Water is delivered by pump or gravity to a HYDRAULIC GIANT or
TOR; pressure at nozzle, 100 to 150 lb per sq in; velocity, 100 to 200 ft per se
ume, 10 to 15 cu ft per sec. When water is scarce the waste may be led to a si
clarifying basin, and used again. The soil, after being loosened by the mon
carried by sluices to pipes or flumes and thence to the dam. The bank from
the earth is washed must be at a higher elevation than crest of the dam, grad
dam being at least 2% for fine materials, and 6 to 8% for coarse, heavy stu

The solid material hydraulicked may be retained by the SUMMIT POOL or SHEA
method. The first is the commoner, consisting in keeping the sides of emba
higher than the center by discharging the mixed water and solids along the sides.
water runs off through drains, the entrances of which are kept above the center
finished part of the fill, thus forming a summit pool or settlement basin. This
only successful method for clay or other slow-draining soil. In the shearboard
the water and its contained solids run lengthwise of the bank, the stream being
from place to place. The slopes are formed by "shearboards," consisting of p
laid edgewise and held in place by vertical stakes. The embankment is built fr
ends instead of from the bottom up, as with the summit-pool. This method
applicable in grading sand and gravel, or other fast draining soils, where terraces
steeper slopes than the natural surface are desired.

The pipes and flumes discharge directly at the sides of the dam, or th
laterals that can be shut off at any time. When any one material begins t
a pervious layer through which water can percolate, the soil must be kn
and mixed by men working with long boards or poles from a raft. Perc
of solids carried by the water in dam construction averages 20 to 25%, r
from 5% to 50%; it depends on hardness and other characteristics of th
quantity of water, and grade of flumes (16).

Cost of the Concully dam (15). This dam is part of the Okanogan project
U S Reclamation Service, at Salmon Creek, Washington. It was made of a talus

formed from disintegration of granite, mixed with angular rock fragments. Water ...med by a diversion from West Fork of the creek, through a 3-mile flume. Creek flow ...50 ft per sec and more during April, May, and June, to 10 ft per sec in Aug and ...ater supply flume, dirt flume, spill-way, etc, were built in 1907. In 1908, 97 000 cu ...aced, in 1909, 188 000 cu yd; and in 1910 the remaining 64 000 cu yd. In 3 years ...percentage of material carried by the water was 2.85, 2.42, and 1.87 respectively. ...was $324 325, or a cost per acre-ft of $24.95 for 13 000 acre-ft. The hydraulick- ...$155 637, or 45.79¢ per cu yd, place measure; unusually high for such work. Table ...itemized costs. Wages per 8-hr shift: Common labor, $2.25 to $2.50; pitmen, $, giantmen, $3; powdermen, $3; carpenters, $4 to $4.50; foreman, $5.

Table 20. Cost of Concully Dam (built by Hydraulic Method)

...pplies, and labor	Total costs, $	¢ per cu yd, place meas	Plant, supplies, and labor	Total costs, $	¢ per cu yd, place meas
Plant:			*Labor:*		
...ary dams & flumes	17 140	5.04	Foreman	4 274	1.25
...and trestles, except ...ing	20 128	5.94	Building road to pit	571	0.17
...ining	14 964	4.40	Clearing borrow-pit	547	0.16
...giants and hose	3 725	1.09	Feed-supply flume tenders	7 261	2.14
...light plant	1 250	0.37	Giantmen	2 365	0.70
...share of camp bldgs	1 116	0.33	Pitmen	13 232	3.89
...ndence	5 076	1.49	Clearing pit of rock	8 358	2.46
...n, eng'g, gen exp	7 805	2.29	Building lateral flume in pit	963	0.28
	71 204	20.95	Hauling & laying pipe in pit	2 707	0.79
...n of plant on hand	2 105	0.62	Dirt flume tenders	7 748	2.28
	69 099	20.33	Labor, steel lining	4 975	1.46
Supplies:			Spr'd'g soil & puddl'g in dam	1 354	0.40
...	2 150	0.63	Carp'ters on dam & flumes	3 915	1.15
...boots & clothing	2 700	0.80	Blacksmith	944	0.27
...and explosives	4 793	1.41	Operating light plant	767	0.24
...share of camp bldgs	170	0.05	Transporting laborers	1 308	0.38
...ndence	776	0.23	Dismantling plant	936	0.28
...n, eng'g, gen exp	1 194	0.36	Propor charge for camp bldgs	998	0.29
	11 783	3.48	Superintendence	4 543	1.34
			Administ'n, eng'g, gen exp	6 989	2.06
				74 755	21.99

...ing Mississippi River banks by water jet. In 1882, a barge 110 by 30 by 6 ft ...ed a Blake compound duplex pump, with 16 by 24-in plungers, and 18 and 36 by ...ain cylinders: capacity, ...per min, at 160 lb per sq ...water was led to shore ...in pipes, and delivered ...nozzles through 2.5-in ...se. In operation the ...moored to the bank, ...se led to within 8 ft of ...ce cut by hand in bank. ...nies, each manned by ...aver of 1 300 cu yd was ...per day at about 4¢ per ...was graded the bank to ...1 to 1, trimming be- ...by hand.

Table 21. Cost of Hydraulic Filling (Nor Pac R R)

	377 000 cu yd (gravity supply), ¢ per cu yd	42 250 cu yd (water pumped), ¢ per cu yd
Sluicing and building side levees	3.85	10.81
Hay used in levees	0.09	0.21
Tools, lumber, nails	0.30	0.26
Labor, building flumes	0.44	0.14
Coal for pumping	0.00	1.87
Eng'g, superintendence	0.11	0.20
Total	4.79	13.50

...ern Pacific R R embankment. In 1896–7, a number of trestles were filled ...t by the hydraulic method. In 8 cases, where there was a gravity supply ...the cost was 4.79¢ per cu yd. In one case, pumping was necessary, making a ...¢, which included clearing of dense forest growth (Table 21). Plant includ

a No 2 giant, costing $95, and 300 to 1 000 ft of light sheet-iron pipe, at 27 Sluice boxes were paved with 3-in blocks, and when removing gravel or rock on grades of 7 or 8% (1).

11. Trenching

Methods: By hand; with scrapers, drawn by horses or cables; with gral loaded buckets on a cableway or overhead track; by trench excavators, stea land dredges, or derricks.

Hand work. Cost depends on hardness or tenacity of the soil, n boulders or other obstructions, presence of water, and depth and (sc on the width of trench. In trenches over 4 ft deep, some of the soil shoveled twice: first, to the surface; then, as the spoil pile grows, b edge of trench. In trenches 6 to 12 ft deep, the soil must first be thr staging about halfway up, thence to the surface, and finally, back-from For depths of 12 to 18 ft, the soil must be handled 4 times.

From studies by the Construction Service Co (4) of sewer, drain, and water-pip tion, in aver soils, 9.23 cu yd per man per 10 hr are excavated in trenches 3 to 7.37 cu yd for depths of 6 to 12 ft, and 6.38 cu yd for 12 to 18 ft. (For dee trenches, machine excavation is better than hand work.) These rates are as hi be expected for aver trenching; when water, quicksand, or boulders are present the men and superintendence are less efficient, costs may reach $1.25 per cu yd of trench affects the cost. In narrow work, men are cramped and can not work e in very wide trenches, cost of removing the spoil from the edges increases tota

Cost of back-filling depends on: nature and condition of soil (whether fro packed, or dry); means employed to back-fill; amount of tamping require back-filling and tamping are done by hand, work per man-hr ranges from 1- aver 1.5 cu yd; most compact tamping (clay excepted) is obtained by casting water; for thorough dry tamping on large-scale work, power tampers should

Stayley power-tamper will strike in a trench from 1 to 4.5 ft wide by 6 ft weighs 950 lb, costs $300, and requires 1 operator and 1.5 gal gasoline per 10 hr. 60 blows per min and, assuming 50% lost time, should cover 900 sq ft per hr; or, in 3 ft wide and 5 ft deep, tamped in 6-in layers, should compact 30 lin ft of trenc

Labor cost of sheeting in small trenches is from 0 to say 25¢ per trench, or 0 to 1.5¢ per sq ft of wall; aver, about 0.4¢.

Trench excavation by scrapers is advantageous in trenches wide for a team, and in soil free from obstructions and not requiring sheeting. cableways, with scraper-excavator buckets, are economical on wid trenches, and may be used even where braces and sheeting are necessar

In building a 7-ft sewer in sand at Gary, Ind (2), where the ground-water at ma was level with the surface, the trench was unwatered to depths of 6 or 8 ft, and Schnable drag scraper was used to dig to water level. This held 2 cu yd and oper 58-ft boom. Excavation amounted to 10 cu yd per lin ft of trench, and the mac 60 lin ft per day. (This was not its full capacity, but the speed was determine rest of the work.) The machine was mounted on rollers traveling on a timber t by dumping some of the soil ahead of itself, formed its own roadbed. The moved itself, using the bucket as an anchor for the drag line to pull on. Aver 400 cu yd in 9 hr, with 1 engineer, 1 foreman, and 4 laborers. Costs were as f

	Per day	Per lin ft
Scraper excavator work (400 cu yd)	$22.25	$0.370
Shovel excavation (300 cu yd)	170.00	2.833
Sheeting and bracing (300 cu yd)	29 50	0.491
Pumping and pipe system (300 cu yd)	94 50	1 575
Back-filling (500 cu yd)	20.25	0.337
Miscellaneous (300 cu yd)	7 00	0.116
Total	$343.50	$5.722

Costs of per cu Scraper cluding cle cu yd) @ trenching, and sheeti yd) @ 0.9 filling (50 @ 0.040

...ableways (Sec 26) can be used to advantage on trenches 6 ft and wider. ...ay on 30-ft towers, 300 to 400 ft apart, handling 1-cu yd tub at a time, ... either soft digging or rock, as no part of the machine is carried on the ... Tubs can be loaded at any point and swung as much as 10 ft to ... Engine and 1 tower stand on a car on rails; the other tower stands ... ground, and must be lowered for removal to a new position. Outfit, ... about 19 tons, can be loaded on 1 R R car. Price of a 300-ft cableway ... is about $3 250; 400-ft cableway, about $3 500; they can be hired ... $200 to $225 per month. Engines are usually 7 by 10-in duplex, mov... buckets at about 440 ft per min. To move the machine to a new posi... road, requires from 4 hr to say 3 days, depending upon method used and ... of the men (1, 2, 7, 8).

... 1 trench 14 ft wide by 18 ft deep, in earth slightly more compact than average, ... 400-ft Lambert cableway, delivering buckets through a chute into R R cars, was (1):

	Per day
... loading buckets, @ $1.50	$45.00
...signalman (signaling to engineman) @ $1.75	1.75
... hooking on baskets, @ $1.75; 1 man dumping baskets @ $1.75	3.50
... driving sheet plank and bracing, @ $1.50	6.00
... spreading earth in cars and moving cars, @ $1.50	7.50
...engineman $3; 1 fireman $1.75; 1 water boy $1; 1 foreman $4	9.75
Total	$73.50

... in above work was 260 buckets in 10 hr, each holding 1.3 cu yd loose earth, or ... more than 1 cu yd in place. Wages and coal, $76 a day. Hence, not including ...ering, nor hauling and unloading of cars, the cost of excavation was about 30¢ ... There was no back-filling, as the trench was for a retaining wall.

...ench machines are usually earth conveyors rather than excavators, ...h grab buckets operated by a cableway are included. In general, they ... a cableway or an over-head track, carried longitudinally directly ... the trench. The Adams trench machine comprises a series of wrought-... bents, supported on wheels. Under the top of the bents is a ... the bucket carriage. In the Moore conveyor, the rail for the bucket ... is supported by A-bents. Potter machine has a track on top of the ... which travels a car carrying a hoisting drum. Two men stand on the ...erate the buckets. Carson machine comprises a rail supported on rec... A-frames, with a rail for a train of 6 connected carriages and buckets, ...ing 0.2 cu yd. Its working length is about 288 ft; total length, 336 ft. ...ds on a T-rail track, and may be pulled ahead to a new position by its ...gine. Cost, about $3 400; capacity, about 250 cu yd in 19 hr.

...rking force for a Carson machine, in a 4 000-ft section of the Metropolitan sewer ... boston, comprised: 1 engineman, $3; 1 lockman, $2; 1 dumper, $1.50; 8 shovelers, ... $14; 2 bracers @ $2.50, $5; 2 tenders @ $2, $4; 4 plank drivers @ $2, $8; 2 men ...ing planks @ $2, $4; 8 men pulling planks, etc, @ $1.75, $14; total, $55.50. ...trench was 9 ft wide by 20 to 30 ft deep; aver advance per week, 64 lineal ft in ...y sand, 192 ft in gravel and coarse sand, at a cost of from 25¢ to 80¢ per cu yd. A ... compounding $20 per day was required. Fuel consumption, 0.5 ton coal per day (1).

...raveling trench excavators comprise: the wheel type, in which exca... buckets are carried on the perimeter of a wheel; ladder type, in which ...ng buckets or scrapers are carried on a belt traveling on a ladder frame. ... power plant and excavating mechanism are mounted on tracks, platform ..."crawler" wheels. The earth is scraped into the buckets and carried to ... above the ground surface where it is dumped on a belt conveyor and ... to one side of the machine. These machines do not work efficiently ... containing boulders or other obstructions.

Buckeye machine is of the wheel type. It weighs 5.5 to 51 tons, costs $200 to
ton, and excavates to depths of 4.5 to 12 ft. At Greeley, Colo, a 36-mile wa
trench (excepting 2 000 ft in rock) was dug by a 17-ton machine, 8 miles being in
gravel, the rest in clay. A trench, 30 in wide by 4 ft deep, was dug at an ave
627 ft in 10 hr, at a cost of 14¢ per lin ft. Machines of the belt type are: the M
Parsons (Fig 5), Austin, and the Insley. In laying 7.5 miles of pipe sewers at Ma

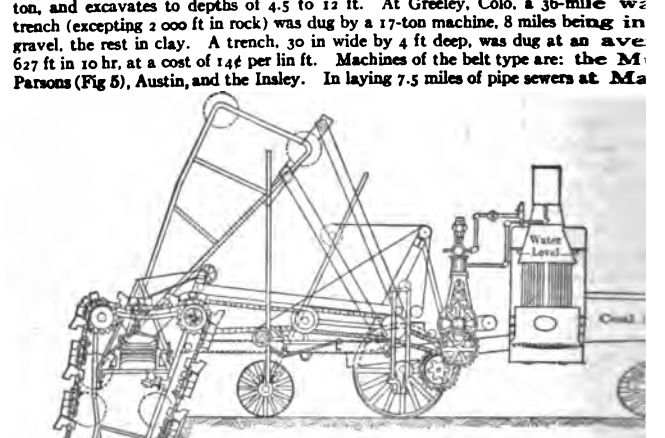

Fig 5. Parsons Trench Excavator

Wis, the daily cost of operating a Municipal machine and lab
was $35. The trench was 27 in wide by 7 ft aver depth, an
500 lin ft, or 300 cu yd, were excavated per day.

Steam shovels for trench excavation are suppo
timbers spanning the trench. As their weight comes
on the banks, they can not be used in soft ground
the walls are well sheeted and braced. The shove
usually be stopped while the trench is being sheeted,
consequent delays materially decrease the output.
excavators, though not subject to this delay, can not
so difficult soil as steam shovels.

From observations by the Construction Service Co, the fo
data are compiled (4). A 25-ton revolving-type shovel, m
on traction wheels, dug 3-ft wide trenches in sandy, cavi
It was supported on a platform in 3 sections, which, as the
moved forward, were carried ahead by hooking them to the dipper. A 70-ton
supported on rollers on a 38 by 24-ft platform, excavated trenches 16 ft wide i
firm clay, loading into dump cars. The shovel was moved forward by a cable a
to a deadman. The excavation was made in 2 benches, the first 10 ft and the sec
ft, in order to obtain fine material for back-filling. Costs of this shovel work
follows. 25-ton shovel: cost per day, $19.21; cu yd per day, 80; cost per cu yd,
actually digging, 35.8% of working time. 70-ton shovel: cost per day, $38.10;
per day, 569; cost per cu yd, 6.7¢; actually digging, 44.9% of working time.

Traveling derricks, land dredges, and locomotive cranes, usually
ahead of the trench or along one side, though they may straddle it.

A 1-cu yd orange-peel bucket, on a full-swing roller-mounted derrick, used in
by city forces, cost $20.30 per day for labor of 6 men. Ordinarily it excavated a
cu yd per 8 hr day, all dumped on a spoil bank at one side; hence, labor cost was
cu yd. 50 lb dynamite @ 15¢, and 0.75 ton coal @ $3, were used per day, mak
field cost of 6.6¢ per cu yd, excluding wear and tear. For this work the swinging
with orange-peel bucket was better than a steam shovel, since it could work

... These were 11 ft apart. With the derrick the soil could be deposited far ... from the trench to make rehandling unnecessary. A steam shovel would have ... temporary bracing, followed by permanent bracing, and its boom would be too ... to deposit the soil the necessary distance from edge of trench.

Ditching by explosives, if properly done, will excavate and spread the ... over a distance, and is economical in dry or wet ground, or soil under ... The flow of water is depended upon to clean out the bottom. In stiff ... hardpan, holes should be 26 in apart, in loose mucky soil, 30 in apart, ... are punched or bored to within 6 in of desired depth of ditch. Strongly ... soil is cut with a spade along side lines. For methods of charging and ... dynamite, see Sec 5, Art 7.

... are best blasted simultaneously with a magneto; or placed 18 to 24 in apart and ... by concussion from a middle hole, detonated by fuse and cap. Table 22 gives ... charges. 20% dynamite is ordinarily used, or 40% in stiff, tenacious soil. For soils ... at top and hard at bottom, use 40% dynamite in bottom of charge and 20% above.

Table 22. Charges for Ditch Blasting (Du Pont Powder Co)

Without magneto				With magneto					
...	Approx number cartridges per hole	Rows of holes	Dist bet rows, in	Top width of ditch, ft	Approx number cartridges per hole			Rows of holes	Dist bet rows, in.
					2.5-3ft	4 ft	5-6 ft		
...	for 2.5 to 3-ft holes 4-ft holes 5-ft 6-ft	1	....	3	0.5	....	....	1	....
		2	30	6	1	2	3	1	....
		2	36	8	1	2	3	2	20
		2	42	10	1	2	3	2	28
		2	48	12	1	2	3	2	36
		3	36	14	1	2	3	2	42
		3	42	16	1	2	3	3	42
	1 2 2.5 3	3	48	Length No 6 Victor fuses	4 ft	6 ft	6-8 ft		

12. Excavating Frozen Earth and Hardpan

Bank blasting. In hard, cemented deposits, dynamite is better than black ... in soft ground, the latter is more economical. With very high banks ... toes should be blown out and the top allowed to drop. Charges should ... so that the line of least resistance is horizontal. With banks 50 to ... high in cemented gravel, the length of main drift should generally equal ... height of bank. Cross drifts are driven parallel to the bank face, their ... depending on length of face to be blasted. Powder charge for aver con- ... is about 4 lb per cu yd of ground to be broken. (For further details, ... 5 Art 7, Table 14, Cases 12–16.)

... at the property of the Milton Mining and Water Co, Sweetland, Cal, during ... required an aver of 0.382 lb Judson powder per cu yd (3). The top gravel had ... washed off, leaving banks (usually hard and cemented for 50 ft, but soft above) from ... to 85 ft high. In some parts, 8.3 to 8.5 cu yd were shattered per lb of powder. ... clay, loam, conglomerates, marl, etc, difficult to pick, blasting may be economical. ... ate by picks until a face is formed, and make vertical 1.5 to 2.25-in holes in a ... of the face with pointed bar, churn drill, or auger. Depth of holes should ... less than the height of bank, distance between them being 1.5 times line of least ... (Sec 5, Art 5). In the formula $B = CR^3$, B = charge in oz, and R = line of ... resistance in ft; the "rock coeff" C should be determined by trial. For loam, con- ... and ordinary soil, using 30% dynamite, C is usually nearly 0.6. Holes in ... ground should be "chambered" (2).

Thawing frozen ground may be done by burning gasoline or coal oil, or
of lime, steam jets, or wood fires. In northern Minnesota, where earth fre
a depth of 4 or 5 ft, frozen earth is softened, where pole holes are to be d
pouring in a cupful of gasoline or coal oil and igniting it. Use of 1 gal of oil p
reduces cost of digging 15%. C. P. Chase states that ground frozen to 4
hard to be excavated by a trench machine, was softened sufficiently by spre
small pieces of lime along the line of proposed trench, covering them with m
or straw, and pouring on hot water to slack the lime and liberate the hea

An economical mode of thawing earth for digging trenches is to use open-end w
boxes 12 by 24 in by 16 ft; 4 of them are placed end to end, with joints wrappe
gunny sacking or canvas, and are covered with earth. 2 rows of boxes are set side
over the site of the trench and filled with steam from a boiler. Thawing has thu
done for 1.75¢ per lin ft of trench. For digging post or pole holes, a steam jet pip
be forced down, 2 men being able to thaw about 30 holes in 10 hr.

Cost of trimming and dressing frozen ground on R R construction is thus
About 1 500 ft of roadbed had to be dressed for laying track, 2 300 sq yd of surface
in cut and 900 in embankment. The cut was within a few in of grade throughc
aver of 1 in being trimmed off the surface, amounting to 100 cu yd. 133 cu yd w
cavated by ditches. Only a few places on the fill had to be cut down, the low
being raised with stuff from the cuts. The ground was frozen to about 1 ft deep.
was as follows: foreman, 9.5 days ($31.67), 1¢ per sq yd; labor, 145.5 days ($218.25)
carts, 15 days ($52.50), 1.6¢; total, 9.4¢ per sq yd. Usual cost of trimming on R R
is about 1¢ per sq yd. On this work each man trimmed and dressed 22 sq yd pe
while under favorable conditions he can do about 6 times as much. Each man lo
and shoveled 1.5 cu yd per day. Total cost per cu yd was $1.30. Picks could not b
to advantage except in ditches, and most of the soil was cut with mattocks, 10
pickers being required to keep 3 or 4 shovels busy loading (1).

13. Embankment

Dumping. Earth dams and dikes should be built in horiz layers, but
embankments are usually made by dumping from cars traveling on temp
trestles, and are built up from ends and sides. When filling in old trestl
may be necessary to spread the soil evenly when dumped, to prevent the t
from being thrown out of line.

Unloader plows (ballast unloaders) used to scrape or plow the earth from flat
They are built of steel in 3 general types: center, right-hand, and left-hand plows.
are usually drawn over the tops of cars (after the wheels have been blocked) by a
attached to the locomotive. The Lidgerwood unloader uses a cable from a 10 by
hoisting engine mounted on the first car. Price of unloaders, $300 to $650. A 1
train can be unloaded in 10 to 30 min on straight track. On curves the cable mu
guided by snatch blocks, and unloading requires more time.

Spreading. A man can spread 4 to 10 cu yd (aver, 7.5 cu yd) of earth
hr, 5 to 8 cu yd of gravel or hardpan, and about 2.5 cu yd of mud. A
grader, such as a Shuart or Twentieth Century, with team and driver,
spread up to 50 cu yd per hr, but must have room for the team to operate.
road machine, with 3 teams, driver and helper, will spread 90 cu yd per h
6 in layers, provided there is plenty of room. Under certain conditions e
can be spread efficiently by hydraulicking (Art 10).

Railroad grader is a machine, mounted on standard-gage trucks, having wings a
sides which spread and grade the earth after it has been dumped from cars. A 2
25-ton locomotive is required to pull the grader. Weight, 6 500 lb; cost, $3 000 (11)
train load of 150 to 200 cu yd can be leveled in 8 min sufficiently to clear passing tr
and leveled down to 2 ft below the rail in 10 to 15 min. From 1 000 to 2 000 cu yd
day can be handled. Upkeep cost per day: Depree on $5 000 machine (15-yr life,
days per yr), $1.11; int @ 5% on $1; repairs, $50 per year, 20¢; 1 operator, $2.50;
waste, etc, 10¢; locomotive and crew, $5.; total $10.13.

Sprinkling and rolling embankments for reservoirs is done with grooved rollers. The material must be thoroughly compacted, to avoid future settlement and obtain max- it is usually spread in 6-in layers, soaked with water, and well rolled. 3-horse ... holding 150 cu ft of water, can sprinkle 333 cu yd in 10 hr. Cost of sprinkling ... embankment employing 3 wagons was 2.25¢ per cu yd, 5 cu ft water being used ... Men with hose will usually be more expensive, but this depends on thorough- ... the sprinkling. On a reservoir embankment, made by dump wagons loaded by ... shovel, 12 men were employed on the dump (mostly for picking out stones) and ... and 2 men on a light Austin leveler. Sprinkling was done from a line of 1-in ... with valves every 50 ft. Daily labor cost is given as follows (1): 1 craneman, $4; ... fireman, $3; 1 fireman, $2; 1 pit boss, $2.50; 4 pitmen @ $1.50, $6; 19 teams on ... @ $3.50, $66.50; 2 teams on leveler, $7; 12 men on dump @ $1.50, $18; 1 man ... $1.50; 1 water boy, 50¢; 1 foreman on dump, $3.00; total, $114. At 800 cu ... output, cost of labor and fuel was about 15¢ per cu yd of excavation. To this ... added int and deprec of plant, and cost of moving shovel. This cost is unusually ... reservoir work.

Puddle is a mixture of clay and gravel, wetted and rammed or rolled into a dense mass, ... percolation of water.

... 1 will make 1.3 cu yd, or 1.25 cu yd when thoroughly rolled; formula 2 will ... 16 cu yd, and 1.2 cu yd when rammed. When clay can not be obtained, very fine ... some loam may be used to fill the voids ... Clay alone is sometimes preferred. ... puddle is used in confined places it must be ... concrete and rammed by hand. With ... at $1.50, the cost of mixing and placing ... 10 to 60¢ per cu yd. When covering a ... the gravel should be spread 3 in thick, ... spread over this, and sand on top. The ... mixed by harrow and team, then watered, ... with a 2-ton sectional roller. With labor ... and teams at $3.50, spreading by hand ... per cu yd, harrowing 5¢, sprinkling 2¢, and ... 6¢, total, 20¢.

Table 23. Proportions for Puddle
(Ingredients measured loose)

	For-mula 1	For-mula 2
Coarse gravel..	1.00	1.00
Fine gravel....	0.35	
Sand.....	0.15	0.35
Clay.....	0.20	0.25
Total.....	1.70	1.60

Trestles. Except as an approach to a higher trestle it rarely pays to build ... for construction work under 16 ft in height. The track should be built up ... required height by jacking and filling in the dumped material. Cost per cu yd ... material dumped is nearly constant for trestles between 20 and 60 ft high.

Costs. Two trestles (Table 24) were designed to carry a train of 5-cu yd dump cars ... trestle was filled, and locomotive only after filling. Each bent consisted of 2 soft-wood piles, bracing, and cap. The timber, except bracing, was second-hand. There were 2 8 by 16-in stringers for each 13-ft span; stringers were recovered; all other material was buried in embankment.

Table 24. Cost of 2 Trestles

	No 1	No 2
...th, ft	2961	2142
... height, ft.....	40	35
... cost	$9 907	$5 853
... cost, lin ft ...	1.30	1.22
... cost, lin ft.	1.74	1.51
... cost, lin ft.....	3.04	2.73

A pile trestle for building a concrete pier at Superior Entry, Wis, cost $16 per M for lumber and $3.25 per M for labor of framing and erection, exclusive of foundation piles. It was built each side of the proposed pier, 3 023 ft long.

Table-supported track and dumping platforms may be used where the ground to ... is too soft and yielding to carry a trestle, or where the required height of a trestle ... make the cost prohibitive. Two of these were used on the Lake Erie & Pittsburg ... One was 1.25 mile long, for a 30-ft embankment on a marsh. First attempt at ... was by laying crossed timbers on the muck surface, piling brush upon this grill- ... dumping trains from this. After being extended 600 ft, the entire work sank ... ft of water. A trestle built for the same distance became badly deformed after ... use. The third method was as follows: Two 1⅝-in steel cables, at 5-ft centers, ... stretched from the fill already made over the nearest remaining trestle bents to the

...ber bents. Th... ...ese cables ties and nails were laid. Cable towers eventually w... ...ad the trestle ... Trains of 7 cars (4-cu yd) were run on the cables, loading ... ...00 cu yd per ... ... of 2 shifts, most of the material being dumped by day shift. ... near Cleveland, O. ... method was used for making a fill across a 400-ft gorge, 95 ft deep. ... was 2 small ... ...e bent on each bank and a tail bent in the center. Two 1 1/16-in ... ...anized cables, 7 L... ...part, provided with turn buckles, were stretched over the bents and ... ...ored to dead-me... ...of buried logs. Ties for the rails were fastened to the cable by ... ...bolts. Small trest... ...bents were put in as the fill advanced. Table 25 shows the cost.

Table 25. Comparative Costs of Cable Roadway and Trestle

Actual cost of series cable roadway:		
2.25-in galvanized bridge cable, 1 000 ft		$600 00
Eyebolts, 2 5 in diam, with clevises for both ends		108 30
Turnbuckles (2) at 00		120.00
Chains (2) at north end		67 40
Cast washers (4), 8-in d		1 46
Timber for A-frame (a	tained on ground):	
upper 42 ft, 24 ft by	g and cross ties, 3 800	
ft, at $34 per M		108 80
Lower 50 ft of A-fra	ng, rough in tree	32.00
Team work for hauli	ring timber for erection	65.00
Carpenter labor on	bank	231 40
Time of supt, getting on	work in general	60.00
Common labor, for digging trenches,	and erecting cableway	112.00
Nails and iron in A-frame and bents		29.40
Total cost of cableway		$1 531.76
Estimated cost of timber trestle:		
Timber (all uprights, planks for bracing, stringers, etc), 98 000 ft @ $26 per M		$2 548.00
Labor, @ $6 per M, $588; spikes, $98; drift bolts, $40		726.00
Total cost of trestle		$3 274.00
Balance in favor of cableway		$1 742.24

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SECTION 4

EXPLOSIVES

BY

J. G. HASKELL, E.M., FLETCHER B. HOLMES, A.B., ARTHUR LA MOTTE, AND F. J. LEMAISTRE, PH.G., B.Sc.

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Note.—Numbers in parentheses in text refer to Bibliography at end of this section.

1. Chemistry of Explosives

Underlying principles. The power of an explosive to do work depends upon the facts: (*a*) that a small volume of explosive is capable, under certain conditions, of changing into a large volume of gas at high temperature, and (*b*) that this change takes place almost instantaneously, resulting in the development of great expansive force at the moment of explosion. In the case of black blasting powder, a mixture of sulphur, charcoal, and sodium or potassium nitrate, the nitrate supplies oxygen for combustion of the sulphur and charcoal. The decomposition of black powder, once started, therefore proceeds without need of oxygen from the air. The case is somewhat different with nitroglycerin, a compound of carbon, hydrogen, and nitrogen, which is explosive in itself without requiring admixture with other substances. When detonated, it is decomposed into CO_2, nitrogen, and water, which at the high temperature of explosion and at atmospheric pressure occupy about 1 000 times the volume of the original nitroglycerin.

Ingredients and their properties. The common ingredients of high explosives are given in Table 1. The term "explosive base" in column 3 covers, besides compounds explosive in themselves, certain compounds which are not explosive alone, but become so when sensitized by some such substance as nitroglycerin.

Note.—Throughout this section, nitroglycerin will generally be designated by N G, and other ingredients of explosives by their chemical symbols.

Table 1. Ingredients of High Explosives

Ingredient	Chemical symbol	Function	Remarks
Nitroglycerin	$C_3H_5(NO_3)_3$	Explosive base	Liquid, highly explosiv
Nitrocellulose (guncotton)	$(C_6H_7(NO_3)_3O_2)_x$	Explosive base and gelatinizing agent	Solid, highly inflamma and explosive when
Nitrostarch	$(C_6H_7(NO_3)_3O_2)_x$	Explosive base	White powder, highly flammable and explo when dry
Organic nitrocompounds	.....	Explosive base, but used primarily to reduce freezing point	Some solid, others liq the higher nitro-c pounds explosive, lower non-explosive themselves
Ammonium nitrate	NH_4NO_3	Explosive bases and oxygen carriers	Solid, not explosive alo very soluble in water
Potassium chlorate	$KClO_3$		Soluble in water, hig explosive when mi with combustible mat
Potassium perchlorate	$KClO_4$		Difficultly soluble in wat highly explosive wh mixed with combustil matter
Sodium nitrate	$NaNO_3$	Oxygen carrier	Soluble in water, not exp sive alone, deliquesce
Potassium nitrate	KNO_3	Oxygen carrier	Soluble in water, not exp sive alone, not deliqu cent
Wood pulp	.....	Absorbent and combustible	Best combustible absor ent; in highest grad equal to kieselguhr i absorbent capacity
Wood meal	.....	Absorbent and combustible	Fairly high absorbent capacity
Ground coal	.....	Combustible	.....
Charcoal	.....	Combustible	.....
Flour	.....	Combustible	.....
Sulphur	S	Combustible	.....
Chalk	$CaCO_3$	Antacid	.....
Zinc oxide	ZnO	Antacid	.....
Kieselguhr	SiO_2	Absorbent	Has no value except as absorbent

Reactions. When carbon burns in presence of an excess of oxygen CO_2 is ... if there be insufficient oxygen for complete combustion, CO also is ... When carbon in lumps burns in air, combustion is slow; but if in ... the reaction is very rapid, and may result in explosion. The in... of black blasting powder (S, charcoal and niter) are finely ground, and ... ghly incorporated, to bring all parts of the combustibles into close con... with the oxidizing ingredient, thus favoring rapid and complete combustion.

When black powder explodes, the reaction is:

$$2\,KNO_3 + 3\,C + S = K_2CO_3 + K_2SO_4 + 3\,K_2S_2 + 14\,CO_2 + 10\,CO + 10\,N_2$$

	Charcoal	Sulphur	Potass carbonate	Potass sulphate	Potass trisulphide	Carbon dioxide	Carbon monoxide	Nitrogen
	Solid	Solid	Solid	Solid	Solid	Gas	Gas	Gas

...plosion is accompanied by evolution of heat, which expands the gases to ... large volume, resulting in high pressure.

When nitroglycerin explodes, the reaction is:

$$4\,C_3H_5(NO_3)_3 = 12\,CO_2 + 10\,H_2O + 6\,N_2 + O_2$$

	N G	Carbon dioxide	Water	Nitrogen	Oxygen
	Liquid	Gas	Vapor	Gas	Gas

...ense rapidity of this change is illustrated by the fact that, if a pipe 5 ... long were filled with N G, and a blasting cap were detonated at one end, ... column would be converted into gas within about one second.

Dynamite consists essentially of a mixture of $NaNO_3$, wood meal, and N G. $NaNO_3$ may be replaced by KNO_3, the wood meal by flour or sawdust, and ... a portion of the N G by other organic compounds or by NH_4NO_3. The ... ingredients are not so finely divided as those of black powder, nor so thor... incorporated; hence, the mixture would not burn so rapidly except for ... N G, the extremely rapid explosion of which so accelerates combustion of ...her ingredients that the whole mixture explodes much faster than black ...der. Taking a dynamite of the composition: N G, 40%; $NaNO_3$, 46%; ... meal, 14%; and assuming that wood meal has same ultimate composition ...s cellulose $(C_6H_{10}O_5)_n$, the reaction of explosion is:

$$4\,C_3H_5(NO_3)_3 + 6\,NaNO_3 + C_6H_{10}O_5 = 9\,CO_2 + 6\,N_2 + 10\,H_2O + 2\,O_2 + 3\,Na_2CO_3.$$

Gaseous products of explosion. When explosives detonate, they usually ... a mixture of solid, liquid, and gaseous products. The solid products may ... sodium or potassium carbonate, sodium or potassium sulphate or ... where sulphur is present in the explosive, and calcium carbonate, etc. ... all explosives, except black powder, form large quantities of water, be... vapor at moment of detonation. Smoke consists of the solid products ... divided state. Gaseous products are of most importance to the miner, ... they determine character of fumes after a blast, and provide the ruptive ... For products of different explosives, see Sec 23, Art 3, Table 3.)

With the Bichel pressure gage (2), it is possible to detonate an explosive in a closed ...er, to withdraw a sample of the gases formed by the explosion, and to determine ... composition. The gases produced by a 40% straight gelatin under these conditions ... approximately the following composition, as determined experimentally: CO_2, ... 12%; O_2, 1%. The composition of the gases varies with different explosives. ...me cases there is a small amount of free O_2, as in example cited above; in others, ... O_2 but varying amounts of CO and H_2. CO is poisonous, and serious or even fatal ...urances may result from the use of explosives which produce large amounts of this ... places where ventilation is poor.

Character of fumes from an explosive is affected by conditions under explosive is used. When dynamite BURNS instead of detonating the fum entirely different from those formed by detonation, and contain large an of oxides of nitrogen and CO, both poisonous. Burning of dynamite in a hole may result from improper mode of charging; for example, if the f passed through a cartridge, the dynamite may be ignited by side-spit from fuse. Blown-out shots are apt to produce noxious fumes; well-t shots are least apt to yield them. When dynamite is so charged that ma amount of useful work is done, the fumes are least harmful.

2. High Explosives

General classification of explosives. There are two general classes: (different types of black blasting powder, and (*b*) high explosives. powder is a mixture of combustible and oxidizing ingredients, no one of is explosive alone; high explosives always contain an ingredient which plosive in itself, at least when sensitized by proper means. Because o difference in composition, high explosive detonates with much greater ra than black powder; hence, the great rending and shattering effect of explosive, even when unconfined. It is a common idea that high expl "shoot down," while black powder "shoots up." This fallacy arises fro fact that the slow black powder, when exploded unconfined on top of a r other object, does no damage to the rock, but dissipates into the air; whi quicker high explosive, under same conditions, may break the rock be it, even without confinement other than that of the atmosphere. All expl exert equal pressure in all directions.

Properties. The different types of high explosives vary widely in properties. Some are exceedingly quick, others relatively slow, still o intermediate in quickness. Different types of explosive also vary in de from the heavy gelatins to some of the coal-mine powders, which are light. High explosives are graded according to their strength compared straight dynamite, which is the only type containing actual percentages of The other types make up their strength by such explosive substances as substitution compounds, explosive salts, and guncotton. High explosives taining no N G are graded the same way as "40% strength."

Straight dynamite, containing only N G, $NaNO_3$, wood meal, and an an (Table 1) is taken as standard because it is the simplest and best known ty high explosive in the U S. It is more or less pulpy, easily crumbled wrapper is removed, obtainable in different strengths up to 60%, very q fairly waterproof, and the most sensitive of the dynamites. A 50-lb case tains 100 cartridges 8 in sticks. Straight dynamites are suitable for work requ strength and quickness where water conditions are not too severe; not re mended where ventilation is poor. Freezing point, usually from 42° to 4

Ammonia dynamites have explosive base consisting of N G and NH_4 They are of same strengths as straight dynamites, but slightly slower and sensitive, do not easily ignited by flame, and, hence, not liable to be lighte side out of fuse. Because of the solubility of NH_4NO_3 in water, they req more care in wet work than straight dynamites.

Ammonia gelatins are similar to straight gelatins in composition and properties. Their explosion products are especially free from noxious fumes; hence, they are recommended for confined work.

Table 2. Classification of High Explosives

Principal types		Essential ingredients
Straight dynamite		Nitroglycerin, sodium nitrate, and wood pulp or other combustible material
Ammonia dynamite		Like 1, with addition of ammonium nitrate
Straight gelatin		Nitroglycerin, nitrocotton, sodium nitrate, and wood pulp or other combustible material
Ammonia gelatin		Like 3, with addition of ammonium nitrate
Blasting gelatin		Nitroglycerin and nitrocotton
Granulated dynamite		Sodium nitrate, sulphur, and coal, sensitized by nitroglycerin
Permitted explosives for coal mines	High-combustible class	Nitroglycerin, sodium nitrate, and high percentage of wood-pulp or other combustible material
	Ammonium nitrate class	High percentage of ammonium nitrate, with low percentages of nitroglycerin and wood pulp
Explosives not containing nitroglycerin	Ammonium nitrate class	Ammonium nitrate, with small amount of organic nitro-compounds
	Nitrostarch class	Nitrostarch, sodium nitrate, and combustible material
	Chlorate class	Potassium chlorate or perchlorate, with organic substances

Note. — Low-freezing modifications of nearly all dynamites and gelatins are also on the market. They have same essential composition as the above, with additional ingredients which cause them to remain unfrozen for a long time at temperatures far below the freezing point of other nitroglycerin explosives.

Blasting gelatin is a tough, elastic, jelly-like mass which, except for 1% of antiseptic, consists entirely of N G and guncotton. It is the strongest and most powerful of all explosives. When loaded to fill drill-hole completely it is excellent for hard rock, especially where large holes can not be drilled. Owing to its elasticity it is difficult to make it fill the holes completely; whence, a loss of efficiency. Best results are obtained if explosive is charged with wrappers which can then be pressed in to fill the hole, without so much tendency to spring back and leave unfilled spaces. When soft and plastic, blasting gelatin is no more dangerous than other explosives, but when frozen it should be handled very carefully. It is dangerous to break frozen sticks of blasting gelatin.

Granular dynamites are mixtures of $NaNO_3$ and combustible dope in form of hard grains with a small percentage of N G. They are free running, especially the lowest grade, such as R R P, containing 5% N G, which is in grains nearly corresponding in size to FF

blasting powder. R R P dynamite is usually packed in paraffined bags, containin
Granular dynamites are slowest of all dynamites, approaching black powder mo
than other high explosives; not well adapted for wet work, but resist water be
black powder; especially useful for stripping work, and for loosening sand and e

Coal-mining explosives. Special "short-flame" explosives should in collieries containing dangerous amounts of inflammable gas or dust. explosives have been largely developed during the past few years, an now on the market can safely be used under proper restrictions. Thei facture began in Germany about 1886; in U S, 11 300 lb w in 1902; in 1913, 27 685 770 lb, of which 21 804 285 lb were used in col

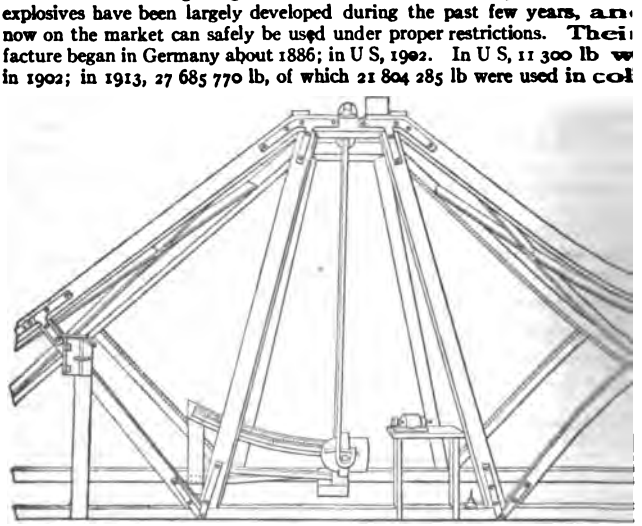


Fig 1. Ballistic Mortar

"Permissibles." At the Pittsburg testing station of U S Bureau of Mines mining explosives are tested to determine whether they meet definite requireme safety in "fiery" mines. Those which pass the prescribed physical and chemica are classed as "permissible" explosives (19), lists of which are published from t time. The tests include the firing of "blown-out" shots into explosive mixtures and air, coal dust and air, or gas and dust with air, in a large steel gallery (Fig 2) plosives which do not cause ignition of such mixtures, and which are also satisfact to chemical composition, stability, sensitiveness, and volume of poisonous gases ev are considered "permissible," when used under prescribed conditions. These expl are recommended by U S Bureau of Mines for use in collieries, and in some states quired by law for dangerous mines. Bureau of Mines bulletins describe meth testing, results, and fees for testing explosives (19).

Classification of permissible explosives in the U S: (*a*) ammonium-nitrate sives, containing NH_4NO_3 as chief ingredient, are insensitive to shock, free from lia to ignition from side-spit of fuse, and produce small amount of noxious fumes; (*b* drated explosives, in which the desired reduction of temperature results chiefly fro water of crystallization of salts included in their composition; (*c*) explosives of th ganic-nitrate (other than N G) class include nitrostarch explosives; (*d*) nitroglyc class comprises those containing N G which are not included in the other cl These are deficient in oxygen, thus producing much CO, instead of CO_2, but are prefe in wet mines. In each class are explosives of widely varying properties, and the sele of a suitable "permissible" depends largely upon local conditions.

High explosives not containing N G usually contain no liquid ingred which can freeze, a decided advantage in cold climates. They are usually l

plasticity, are often somewhat dusty, and have the disadvantages of low low strength, and low sensitiveness. They can not be used indiscriminplace of N G explosives, but are useful for special purposes.

Low-freezing explosives are similar in properties to other high explosives, resist freezing at temperatures considerably below freezing point of N G; remain unfrozen for days or weeks at temperatures as low as 0° F. More and skill are required in making low-freezing dynamites than regular dyna- and since many grades are on the market, some of doubtful merit, their can requires special attention.

Testing high explosives. While laboratory tests, however elaborate, can not entirely practical field tests, in determining relative value of different explosives for any

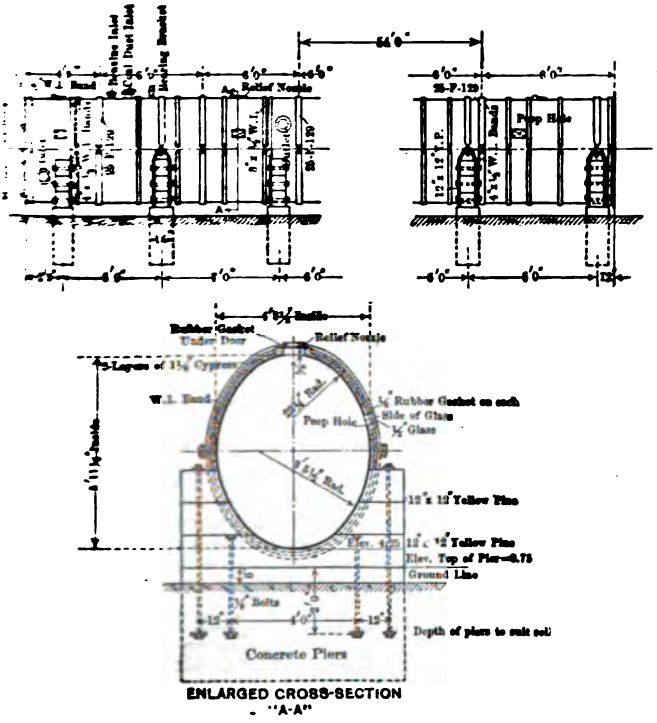
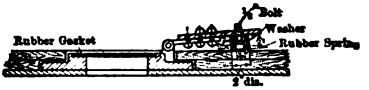


Fig 2. Testing Gallery

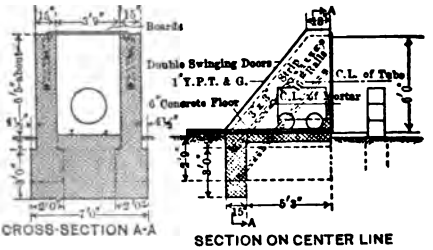
similar work, they can aid in selection, if carried out with suitable apparatus and by expert persons. No single apparatus or test is sufficient, since the practical value of explosive depends upon many factors. Some of the more important factors, determined in a well-equipped laboratory, are:

1. **Power**: determined in the ballistic mortar (31) (Fig 1) or in the Druckmesser.
2. **Brisance, or velocity of detonation**: determined by the Bichel (2) or the Dautriche (35).

STRENGTH AND QUICKNESS combined: determined by the Trauzl lead block t and 34). (Note. For a discussion of the Druckmesser, Trauzl, and ballistic morta with comparison of results, see Bibliog 32.)



SECTION SHOWING RUBBER SPRING OF RELIEF NOZZLE.



CROSS-SECTION A-A

SECTION ON CENTER LINE

Fig 3. Detail of Mortar Housing for Testing Gallery

SAFETY in gaseous and dusty mines: determined by means of the testing galle (Fig 2 and 3).

OTHER FACTORS are: propagating power, density, resistance to water, resista freezing, stability, and sensitiveness to impact.

3. Black Blasting Powder

"A" blasting powder (saltpeter) is made from KNO_3, charcoal, an phur, in the approximate proportions of 75, 15, and 10. It is used mai quarrying, for blasting hard dimension stone, and for work in damp clima

"B" blasting powder (soda) is made from $NaNO_3$, charcoal, and su in the approximate proportions of 72, 16, and 12. Because of its lowe "B" powder is more commonly used than "A" powder, and is sufficiently for most of the purposes for which black powders are used. Owing to quescent property of soda niter, "B" powder is less desirable for use in climates, and for long transportation or storage.

Important properties. Black powder is not made in different strengt dynamite, but varies in quickness, depending upon size of grain. Classes and "B" are of different granulations. For "A" powder the common are C, F, FF, and FFF; for "B" powder, CCC, CC, C, F, FF, FFF, The CCC grains, representing largest size, are about ½ in diameter; grains, the sma lest, are about $\frac{1}{16}$ in diameter. Fig 4 shows the different

The finer granulations are quicker than the coarser, and are used for bl rock, refractory materials, and coking coal; the coarser granulations ar and are used for other coals, shale, and earthwork, or wherever it is des to heave out the material in large pieces, instead of shattering it. (See A Blasting powder is either glazed (polished) or unglazed. Glazed pow brighter, and more free-running than unglazed, and is more generally Glazing does not increase efficiency, and produces more smoke. The s

of black powder varies from 1.5 to 1.9, usually about 1.8. High specifi results from compressing the powder to smaller bulk, with consequen

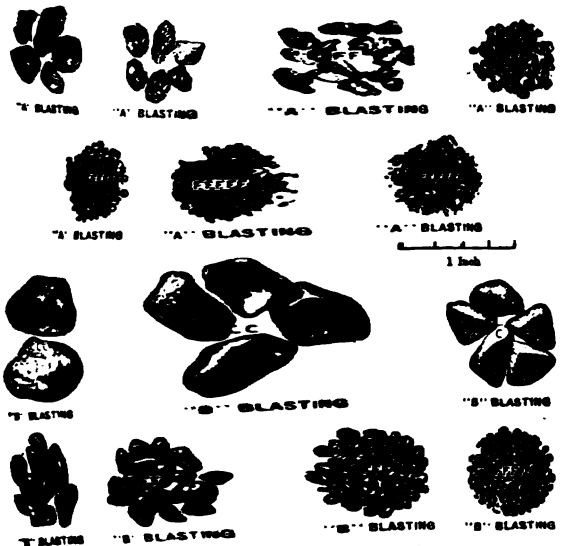


Fig 4. Standard Sizes of Black Powder Grains

action of air-spaces in the grain. Black powder is unaffected by cold, bu little resistance to water, since niter is readily soluble.

4. Transportation of Explosives and Blasting Supplies

Transport by rail. A shipper of explosives should be familiar with loca ordinances, state and federal laws, and the regulations of the Interstate Com merce Commission. By Act of Congress, March 4, 1909, effective Jan 1, 1910 the Interstate Commerce Commission has power to regulate interstate trans of explosives. These regulations specify that explosives to be shipped by rail must pass certain tests for stability and sensitiveness, that container shall stand specified tests for strength, and that cases and contents be packed in a prescribed way. Hence, nearly all manufacturers of explosives doing rail business make and pack all their product to comply with the regulations Printed copies of regulations in pamphlet form are obtainable on request fro the Bureau of Explosives.

Explosives which can not be shipped by rail include:

1. Liquid nitroglycerin.
2. Dynamite containing over 60% N G (except gelatins); see Table Nos 3, 4, and 5.
3. Dynamite having an unsatisfactory absorbent, or showing signs o leakage of N G.

4. Nitro-cellulose in a dry condition, in quantities over 10 lb, in o side package.
5. Dry fulminates in bulk.

The matter of forbidden explosives is of interest to the user mainly in con with the condition of his stock, in case reshipment is necessary; then above becomes important. Dynamite stored for a great length of time, o adverse temperature conditions, occasionally exudes N G, and becom for rail transport. With proper storage, reasonably rapid movement o and care to use old stocks first, this condition should not arise. If ne to ship by rail explosives not acceptable under Interstate Commerce Com regulations, these explosives may be repacked only when authorized by of Explosives. No explosives in broken or damaged packages should be for rail shipment. The aforementioned Act of Congress makes it a c offense to ship explosives on common carriers carrying passengers for hir offer for shipment any explosive under deceptive markings.

Explosives which must not be shipped together. The Bureau plosives publishes a chart showing the explosives and other inflammable which must not be shipped together. A specially important regulation blasting caps must not be shipped or stored with high explosives.

Condition of cars. R R cars in which explosives are shipped must b fully inspected, and must comply with certain specifications. They must certified and placarded in uniform manner, as well as loaded and brace specified way.

Carload shipments. The Interstate Commerce Commission regul permit shipment in one car of not more than 70 000 lb gross weight of expl The minimum quantity taken at carload prices varies with different ra and in different parts of the country, ranging from 17 500 to 30 0 Consignee must remove shipment of explosives from carriers' property 48 hr after notice of arrival at destination; many railroads allow only

Shipment by boat. Navigation laws must be complied with, also all regulations in regard to authorized docks and quantities which may be unl Regulations prohibiting transport of caps with dynamite should apply wh the vessel is under Interstate Commerce Commission jurisdiction or not it is permissible on large vessels, in certain cases, to carry caps in special partments entirely separate from the cargo of high explosives and at saf [illegible] thereon.

Shipment by wagons. Special care should be taken that equipment [illegible] of explosives is in good condition and not likely to break [illegible] Never overload wagons. Wagons intended for regular tran [illegible] explosives should have springs and sides or body should be lined with b [illegible] Explosives may be t [illegible] body is covered with boar [illegible] explosives.

4. Shipping Containers

Black blasting powder [illegible] shipped in kegs of two sizes, k [illegible] powder, or in 5-lb cans [illegible] anthracite coal fie [illegible] powder. Two "sl

... of powder) are packed together in a long can. Gross weights of more ... packages of black powder are:

... lb keg with contents	Approx	gross	wt	27¼	lb
... lb " " "	"	"	"	13¾	"
... lb cans, with contents and shipping box	"	"	"	135	"
... lb can, with contents in two cartridges	"	"	"	29	"

High explosives are shipped in cylindrical cartridges, about 8 in long and ... in diameter. Cartridges are usually packed in wooden boxes or cases, ... 50 lb to the case. Character of cases, as to strength, thickness of wood, ... construction, is regulated by Interstate Commerce Commission. The ... of cartridges of each size in a 50-lb case is fairly regular for any given ... of dynamite.

Size cartridge	Approx number cartridges per 50-lb case.		
	Straight dynamite	Gelatin dynamite	Colliery powders
... by 8 in	195	174	240
1 " 8 "	150	132	200
1¼ " 8 "	100	88	135
1½ " 8 "	68	63	100
2 " 8 "	42	34	57

Approximate weight per cubic inch:

Gelatin (straight and ammonia)	0.97 oz
...namite (straight, ammonia, and granulated)	0.85 "
...llery powders	0.63 "

Note. — There are some exceptions. Also, there are a few explosives, such as the low ... ited dynamites, which are packed in 12½-lb paper bags, four bags to a case.)

Blasting supplies. Caps are packed in tin boxes, containing 100 caps, and ... boxes in wooden cases of 500, 1 000, 2 000, 3 000, or 5 000 caps. Electric ... and electric squibs are packed in cardboard boxes containing 25 or 50 each, ... these are packed in wooden cases of from 250 to 500 electric caps or squibs. ... fuse is in rolls of two 50-ft lengths; shipped in wooden boxes containing ... 1 000 to 6 000 ft.

6. Storage of Explosives and Blasting Supplies

Explosives should be stored in well-ventilated buildings, erected for the ... Buildings for storage of black powder or blasting supplies should be ... proof; those for dynamite, both bullet-proof and fireproof.

Location of magazine. In selecting a magazine site the local topography ... be considered, and advantage taken of such natural protection as is ... by hills and areas of timber. The magazine should be far enough from ... buildings to minimize danger to life or property through an accidental ... Table 3 gives distances, depending upon quantity of explosives, to ... tained between magazines and inhabited buildings, public railways, and ... highways:

Table 3. Distances for Magazines, American Practice

Quantity of explosive stored, lb	Minimum distance, ft			Quantity of explosive stored, lb	Minimum dist. ft		
	Inhabited building screened*	Public railway screened*	Public highway screened*		Inhabited building screened*	Public railway screened*	
50	120	70	35	10 000	890	535	
100	180	110	55	20 000	1 055	635	
200	260	155	75	30 000	1 205	725	
300	320	190	95	40 000	1 340	805	
400	360	215	110	50 000	1 460	875	
500	400	240	120	60 000	1 565	940	
600	430	260	130	70 000	1 655	995	
700	460	275	140	80 000	1 730	1 040	
800	490	295	150	90 000	1 790	1 075	
900	510	305	155	100 000	1 835	1 100	
1 000	530	320	160	150 000	1 965	1 180	
1 500	600	360	180	200 000	2 095	1 260	
2 000	650	390	195	250 000	2 215	1 320	
3 000	710	425	215	300 000	2 335	1 400	
4 000	750	450	225	350 000	2 445	1 470	
5 000	780	470	235	400 000	2 555	1 530	
6 000	805	485	245	450 000	2 655	1 590	
7 000	830	500	250	500 000	2 755	1 650	
8 000	850	510	255	.....	.....	.....	
9 000	870	520	260	.....	.....	.....	

* "Screened" as here used signifies that the building containing explosives *is sc* from buildings, railways, etc, by either natural or artificial barriers. *Where such b do not exist, distances should be doubled.*

Table 3 is the result of an investigation of a committee appointed by the expl manufacturers of the U S, and represents conclusions reached after prolonged stu available data. The Bureau of Explosives of American Railway Association has app and applies the distances specified to be maintained between magazines and public ways. When there are specific state laws and local regulations, they must be com with, but if there be none, the table of distances gives the accepted practice. W explosives are distributed among several magazines the distances between maga should comply with the following formula (distances given are for magazines fully tected from each other by natural or artificial barriers; lacking such protection, dist should be doubled): For magazines containing 25 000 lb or under, not less than 10 for magazines containing over 25 000 lb, add 1½ ft for each 1 000 lb of explosives ad When applying Table 3 for location, *if magazines are nearer* than the above distan they should be *classed as one magazine* containing total quantity of explosives store all. Magazines containing blasting caps should never be nearer than 50 ft to *any o* magazine, and if quantity is over 20 000 caps, distance should be at least 100 ft.

Construction of magazines. Dimensions of magazine without aisles:

Capacity	Dimensions	Capacity	Dimensions
5 000 lb	8 ft × 8 ft	25 000 lb	12 ft × 12 ft
10 000	8 × 10	30 000	12 × 14
15 000	8 × 12	40 000	14 × 16
20 000	10 × 12	50 000	14 × 18

...es of these sizes are for temporary use, and should therefore be as small ...e for quantity of explosives to be stored. Such magazines are mainly ... using explosives in 2 or 3 sizes or grades; dimensions are there... for the given quantities, leaving floor space sufficient only to per... to enter magazine when filled to stated capacity.

...ions of magazine with aisle from front to back and cross aisle through ...

...acity	Dimensions	Capacity	Dimensions
... lb	8 ft × 9 ft	25 000 lb	12 ft × 18 ft
...	10 × 12	30 000	12 × 20
...	12 × 12	40 000	14 × 22
...	12 × 16	50 000	14 × 24

...structing a magazine consideration should be given to the permanency ... the variety of brands and materials to be kept, the quantities in which ... to magazine will be made, and the ease with which stock can be ... from mill or distributing magazines.

...struction specifications. Stone and concrete magazines are undesir... ...use of danger from missiles in case of accident. Brick or sand-filled ... may be used for dynamite, black powder, or blasting caps. Wood ... magazines without sand filling are suitable for black powder; other ... may be used, but this is the most inexpensive construction; it is not recom... for dynamite, because it is not bullet-proof. The nature and thickness ... varies with the kind of small arms in general use in region where magazine ... Tests show that it requires 10 in of sand, between walls of 1-in ... to stop the bullet from a U S Govt Springfield rifle. Where ordinary ... rifles, such as 30–30 Winchester, are used, 8 in of sand is sufficient. ... wall is bullet-proof against the strongest small arms in use in U S. ... of doors it is found that ⅜-in boilerplate, backed with 3 thicknesses of ... hardwood, will stop the bullet from a U S Springfield rifle. This com... of iron and wood seems to be the most practicable, as any increase in ... of either the wood or iron, with corresponding reduction of the other, ...aterially to weight of the door.

...zines of various widths may be designed along lines indicated below, ... spacing of material being revised accordingly.

1 **Dynamite magazine of brick.** Fig 5 shows a brick dynamite magazine 14 ft ... Any desired length consistent with this width.

...ions may be of brick, stone, or concrete. They should reach below frost line, ... and bearing material.

... are 9 in thick, laid in cement mortar. Use as soft a brick as possible, consistent ... quality and durability.

... roof consists of ceiling-joists, floored as shown. A box is formed above this ... by a 6-in strip around walls, the box being filled with 4 to 6 in of sand. Bullet... construction also helps to maintain a uniform temperature.

... Rafters covered with rough boards or ship-lap, and then with No 24 corrugated ... iron. The iron should have side lap of not less than two corrugations, and ... not less than 6 in. Tin roof may be used, but is more expensive.

... Brick walls are lined with 2 by 2-in nailing strips, covered with 1 by 6-in ... forming a lattice work. Nails should be countersunk. The purpose of lining is ... stock away from walls and assist ventilation.

... Merely a strip of No 24 flat galvanized iron, bent and fastened over ends of ... All iron should be put on with galvanized nails and lead washers.

Floors. 1⅜-in matched flooring, or a sub-floor of ⅞ in. covered with ⅜-in flooring. Note that floor stops 2 in from brick wall, to provide ventilation fro floor.

Ventilation. Foundation is ventilated as shown in Fig 5. Roof vents should

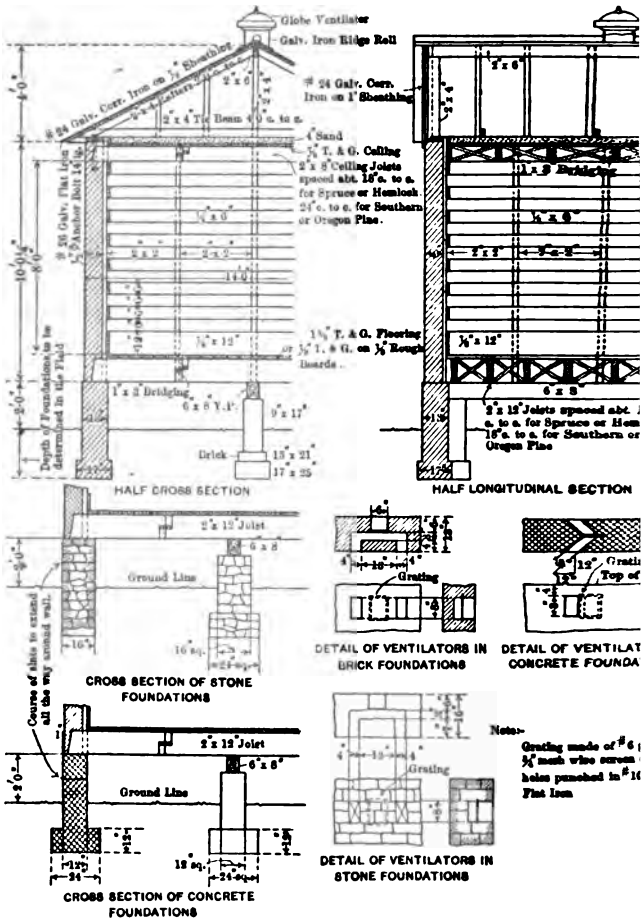

Globe Ventilator
Galv. Iron Ridge Roll
24 Galv. Corr. Iron on 1" Sheathing
2" x 4" Tie Beam 4'0" c. to c.
4" Sand
⅝" T. & G. Ceiling
2" x 8" Ceiling Joists spaced abt. 18" c. to c. for Spruce or Hemlock 24" c. to c. for Southern or Oregon Pine.
20 Galv. Flat Iron ⅝" Anchor Bolt 14'0"
1¼" T. & G. Flooring
⅞" T. & G. on ⅞" Rough Boards
1" x 3" Bridging
6" x 8" Y.P.
9" x 17"
Brick
13" x 21"
17" x 25"
Depth of Foundations to be determined in the Field
HALF CROSS SECTION
HALF LONGITUDINAL SECTION
2" x 12" Joist
Ground Line
CROSS SECTION OF STONE FOUNDATIONS
Course of slate to extend all the way around wall.
Grating
DETAIL OF VENTILATORS IN BRICK FOUNDATIONS
DETAIL OF VENTILAT CONCRETE FOUNDA
Note:-
Flat Iron
DETAIL OF VENTILATORS IN STONE FOUNDATIONS
CROSS SECTION OF CONCRETE FOUNDATIONS

Fig 5. 14-ft Brick Dynamite Magazine

or Globe ventilators, or equivalent, size and number depending on climate, and si building.

Doors. Fig 6 shows details of door for magazine shown in Fig 5.

(b) Wooden dynamite magazine, covered with iron and sand-filled (Fig 7).

Foundation may be of posts, brick, stone, or concrete, as best suited to local conditi

...re of two rows of 2 by 4-in studs, spaced as shown, desired quantity of sand ...ring spacing of studs across wall. Studs are held parallel by nailers at top, ... and intermediate, in number sufficient to prevent spreading under weight of ... studs are covered outside and inside by ⅞-in matched boards, to prevent sand ...ing away. Space between is filled with coarse sand (never use coarse gravel ... stone because of possibility of their becoming missiles). Lower foot of filling to ... a weak mixture of sand and cement, to prevent remainder of sand from leaking ... Outer sheathing is covered with No 24 flat galvanized iron. For other details,

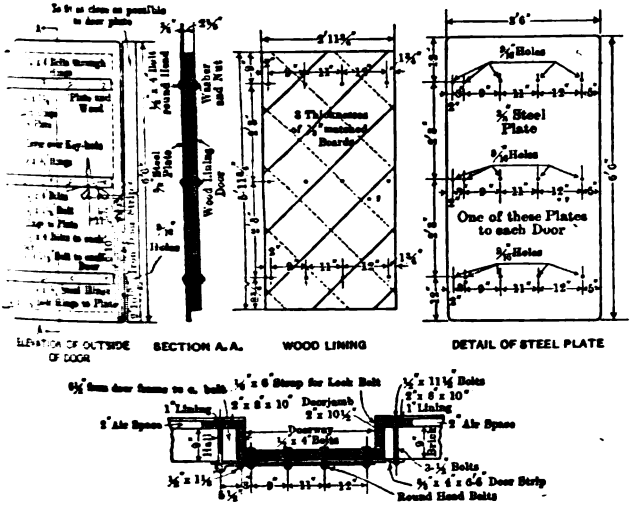


Fig 6. Bullet-proof Door for Dynamite Magazine

...pecifications for brick magazine apply. When post foundations are used the board ... should be ventilated by holes, 8 by 4 in, covered with punched sheet steel.

... Block blasting powder magazine. Fig 8 shows a wood and iron magazine for ... powder or blasting supplies.

... Of 2 by 4-in or 2 by 6-in studding, covered on outside with ⅞-in boards and ... flat galvanized iron; on inside, by 1 by 6-in lattice work.

... *ventilation and floors* (see specifications for brick magazine).

... *ventilation and ventilators* (see specifications for sand-filled magazine).

... Standard door for this type magazine consists of two thicknesses of ⅞-in ... covered with iron of any desired weight, No 22 being considered the lightest ... recently. Either a mortise lock or padlock may be used.

... Iron storage magazine. A "knock-down" iron magazine is manufactured ... is satisfactory for northern or temperate climates. Can be obtained in sizes ... 6 by 6 ft to 11 by 21 ft. When used for dynamite storage it should be made ... proof by lining with 3 or 4 in of hard wood, or with studs and sheathing, sand-filled.

... Small portable magazine, for storage of small quantities of explosives within ... quarry, or at stores, or within city limits. A box may be made of 2-in oak, or ... hard wood and covered with sheet iron. For dynamite, iron should be at least ... thick. Top of box, of like material, should be on hinges. Inside metal should be ... Box should be kept locked and marked to indicate contents.

Care of stock in magazines. Magazines should be so **constru** located that they will not be brought to high temperatures **by rays** N G becomes less viscous at high temperatures, which may **cause the c**

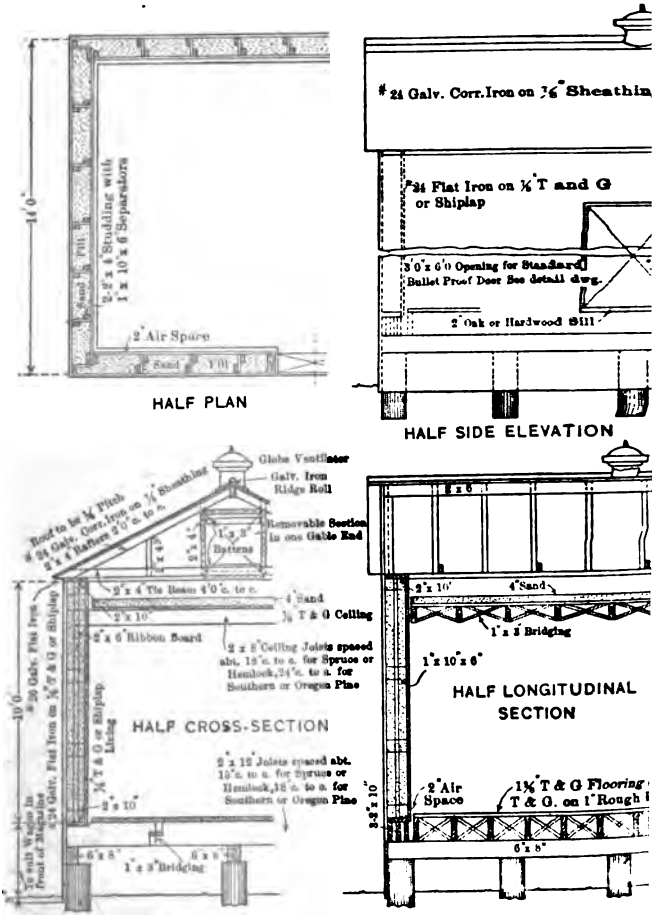

HALF PLAN
HALF SIDE ELEVATION
HALF CROSS-SECTION
HALF LONGITUDINAL SECTION
24 Galv. Corr. Iron on
24 Flat Iron on ⅛" T and G or Shiplap
2" Air Space
2" Oak or Hardwood Sill
Globe Ventilator
Galv. Iron Ridge Roll
Removable Section in one Gable End
Battens
2" x 4" Tie Beam 4'0" c. to c.
4" Sand
⅛" T & G Ceiling
2" x 6" Ribbon Board
1" x 3" Bridging
2" Air Space
6" x 8"

Fig 7. Wooden Dynamite Magazine

to leak. In hot climates metal magazines should be protected by double r and sides, with good ventilation between, so that temperature in magazine w not rise above outside temperature. Take care to keep dynamite dry, especia the ammonia dynamites, as they contain hygroscopic salts, and in humid climat may eventually attract enough moisture to impair their sensitiveness a

When dynamite is shipped in winter at very low temperatures, it ... not be seat down immediately into the mine, as cold dynamite may ... enough moisture in the warm humid mine to impair its efficiency.

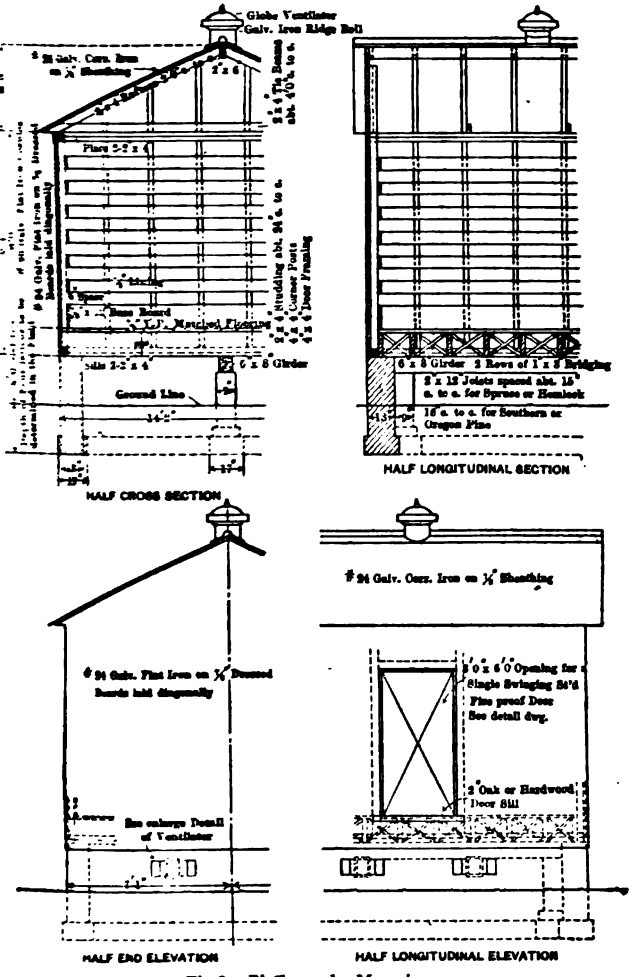

Globe Ventilator
Galv. Iron Ridge Roll
Plate 2-2 x 4
Ground Line
6 x 8 Girder
HALF CROSS SECTION
6 x 8 Girder 2 Rows of 1 x 8 Bridging
2 x 12 Joists spaced abt. 16" c. to c. for Spruce or Hemlock
16 c. to c. for Southern or Oregon Pine
HALF LONGITUDINAL SECTION
Flat Iron on ½ Dressed Boards laid diagonally
See enlarge Detail of Ventilator
HALF END ELEVATION
Galv. Corr. Iron on ½ Sheathing
3'0" x 6'0" Opening for Single Swinging 84'd Fire proof Door See detail dwg.
2" Oak or Hardwood Door Sill
HALF LONGITUDINAL ELEVATION

Fig 8. Black-powder Magazine

... should always be in charge of one person, who can be held respon... condition of magazine and stocks, and for their proper and safe ...ng.

Rules for dynamite and powder magazines.

Explosives must be handled carefully.

Do not throw down boxes of explosives violently, nor drag them along the floo

Do not open boxes of dynamite or powder kegs in or near magazine.

Do not have in or about the magazine loose cartridges, open boxes of dyna loose powder.

Do not make up primers in the magazine.

Do not smoke, have matches, oil-burning lamps or lanterns, fire-arms or cartri or near, magazine. If artificial light be needed, use electric flashlight or electric

Do not store blasting caps nor electric blasting caps in this magazine.

Store dynamite and black powder separately. Store dynamite boxes flat, top grades and brands showing; store powder kegs on sides with seams down, or o bungs down.

Powder kegs should be rolled over and contents shaken every 2 or 3 months.

Always use old stocks first.

Keep magazine floor clean.

Keep the ground immediately around magazine clear of leaves, grass, trees, s and débris, to prevent fire from reaching it.

Do not allow any shooting in neighborhood of magazine.

Keep the door locked.

No unauthorized person should be admitted to magazine.

Do not keep any steel, or metallic tools or other implements, in the magazine.

See that good ventilation is maintained during all seasons of year.

When repairs have to be made to interior of magazine, all stocks of explosives be removed to safe distance and carefully protected from weather during prog repairs. Before starting repairs in a black-powder magazine scrub floor with If dynamite has been stored in a magazine, any stains on floor should be carefully scr with solution consisting of: ½ gal water, 1 gal denatured alcohol, ¼ gal aceton sodium sulphide (fused) or potassium sulphide.

Rules for blasting-supply magazines.

Store blasting supplies only in this magazine, i e, blasting caps, electric blasting and fuse.

Do not store powder or dynamite in this magazine.

Do not have loose blasting caps, electric blasting caps, nor coils of fuse lying a magazine, nor take them out of original packages until required for use. Keep pac closed.

Open boxes with a wooden mallet, except when lids are screwed on; then use a driver. Do not keep any other metallic tools in magazine.

7. Handling of Explosives and Blasting Supplies

The Interstate Commerce Commission in matters of transportation, and majority of states in framing their laws, recognize that explosives are a mercial necessity, and furthermore that they can be handled with reason safety. Nevertheless, one must always recognize their nature; their fun is to explode. All owners of explosives should require employes to obs rigidly the rules and regulations which experience has shown will best cons safety of the men themselves, as well as of the public.

From cars or boats to magazine. Use only wooden or non-sparking metal too breaking the bracing in cars. Wooden wedges and mallets answer all practical purp Damaged or broken cases or kegs found in shipment should be set aside, and not take magazine with undamaged stock. If damage is slight, the cases or kegs should be t to a safe distance from magazine, or from car, and repaired. If damage is too grea repair, take explosives to point of consumption and use immediately. If broken ridges or loose grains of powder are scattered in the car, they should be carefully s up and removed before proceeding with unloading, and afterwards destroyed. If t

...ms screws. D-shaped strips or runners, not more than 6 in apart and running ...w of chute, should be fastened by wooden pegs to upper surface of bottom ...en dynamite packages are being handled, wipe down chutes with waste ...with machine oil. A mattress, 4 by 6 ft, and not less than 4 in thick, or a heavy ...eep mat of like dimensions, should be placed under discharge end of chute. ...st not be so steep that packages slide too rapidly. With a long chute, station ...ment intervals along it, to check speed of packages and prevent bumping ... Do not rehandle or switch the car to other points after the bracing has been ... in case part of shipment is to go to another point, the part remaining in car ...rebraced (according to Interstate Commerce Commission regulations) before ...ered for shipment. The load on any vehicle should be braced before transfer ... Always protect explosives from weather.

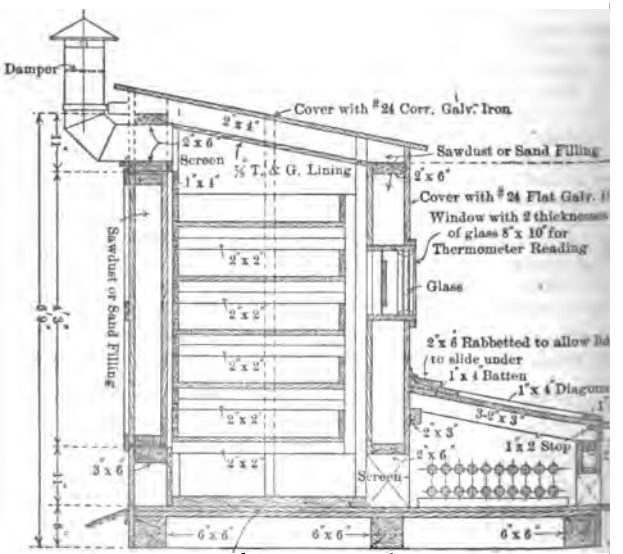
...m mine or quarry. Same rules and regulations should be adopted as ...ound magazines and elsewhere above ground. Never bring exposed ...ose to explosives. Only the smallest possible quantity for economic ... operation should be taken underground at one time. In transport- ...the mine by cage or tram-car, only the man in charge should be per- ...t ride in same cage or car with explosives. Primers should be made up ... entirely separated from regular stock of dynamite. If made up above ... they should be taken into mine at a separate time and in separate car ... explosives. In opening dynamite cases, use no metal tools other ...e made of non-sparking materials; wooden wedges and mallets are ... opening black-powder kegs use wooden wedge. The rather common ... driving a pick or other tool through the keg is dangerous. If possible, ...ing dynamite or powder in mine over night. If this can not be avoided, ...ve should be left in a place set aside for that purpose, protected from ..., and posted so that all persons will know nature of material stored.

...wing frozen dynamite. If dynamite is frozen it must be thawed before ... freezing point of pure N G is about 55° F, it follows that dynamite

Fig 9. Dynamite Thawing Kettles

...t at any higher temperature. In thawing, it is usually necessary to ...peration somewhat; hence, higher temperatures are used. But ... is readily decomposed by exposure to high temperature. Thawing ...n hot ashes, on top of a boiler, or around a bonfire, is dangerous, and ... to explosion, with possible loss of life. Small quantities of dynamite ...be thawed in a double-jacketed kettle (Fig 9), the dynamite being ...inner compartment, while jacket is filled with water at temperature ... 130° F. Water hot enough to scald the hand is too hot to use. Al- ...l water in separate vessel, and then pour into jacketed kettle. Heat ...ever be applied directly to kettle containing the dynamite. Thawing ... these types are made in two sizes, holding 30 and 60 lb of dynamite.

Dynamite thawing-house heated by exhaust steam. Where large of dynamite are used, a thawing-house should be provided. Fig 10 good type of permanent thawing-house, for 500 lb dynamite at a time, of 50 lb; same construction for larger capacity, size being increased units, with corresponding increase of radiating surface of steam coils. steam is usually the most economical and convenient source of hea steam (steam under pressure) is dangerous, because of possibility of h perature. The area of radiating surface necessary to insure complete varies with climatic conditions. In installing the steam coil, give pipes slope to drain properly. If preferred, same type of coil may be used as designed for hot-water heating (Fig 11). The packing in the walls is eit



SECTION A-A

Fig 10. Dynamite Thawing-house Heated by Steam

or sawdust. Dry sand is good insulating material, but makes ship-lap o and groove boards necessary on outside walls to hold it in place. N coarse gravel for packing on account of missiles which would be throwr of accident to thawing-house. Dry sawdust is probably the best cheap Mineral wool or asbestos fiber gives more perfect insulation, besides be proof, but cost is much higher. A thermometer is placed in back wall of t house, behind a double glass window, so that inside temperature may b without opening doors and admitting cold air. A damper is also place stack to permit regulating the circulation of air through the house. ature of 80° F (27° C) is good for thawing. This house can be built materials at approximate cost of $100. At mines or other works there is enough pipe lying around to make the heating coil, and scrap lumber

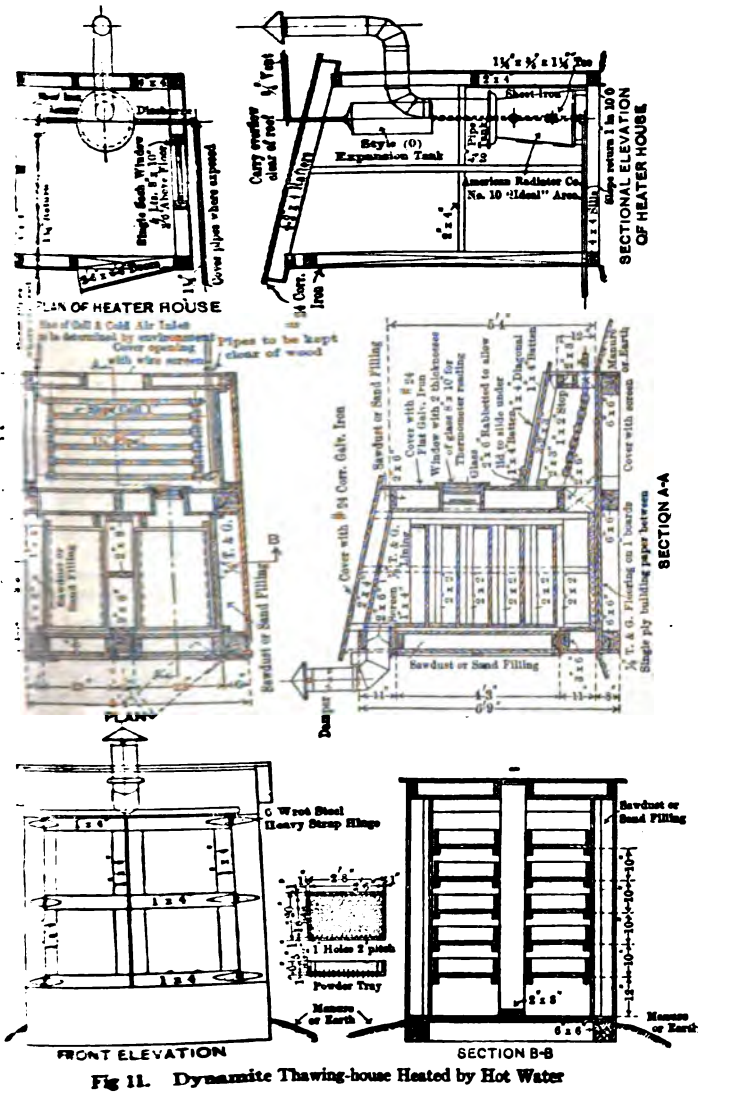


Fig 11. Dynamite Thawing-house Heated by Hot Water

construction, which will materially reduce cost. Note that there is no this house to permit a person to enter, so that it is impossible to use it to prime cartridges. If desired, a small box on wheels can be made

of a size just to hold a certain number of trays from thawing-house. ber of trays needed at one time are placed in the box, covered, and tal work without exposing the dynamite to cold air until ready for chargi

Dynamite thawing-house heated by hot water (Fig 11). This is d thaw 500 lb dynamite at a time, in trays of 50 lb each. For greater capacit size of house in units of 250 lb, with corresponding increase of radiating surfa and size of heater. Construction details are same as of the steam-heated thaw The heater house and standard heating unit for small hot-water systems s Heater house must be enough lower than thawing-house to permit good gr in return pipe to heater. Heater house should be 30 to 50 ft from thawing-hous distance increases construction cost and diminishes economy of operation, whil proximity to thawing-house increases risk. Cover the pipes between the buil standard magnesia pipe-covering, or place them in a sawdust-filled box. Ap cost, when of new materials, $175.

Disposition of damaged explosives. Dynamite to be destroyed should b from magazine in quantities not exceeding 100 lb, to a safe place 400 to 500 from any magazine, and 1 000 ft or more from any dwelling, building, publi railroad; where, in event of its exploding while burning, no damage will be d of boxes should be carefully removed with wooden wedge and mallet, each slit, and the opened cartridges spread upon the ground over as large a spac ticable. To insure proper burning of the dynamite, spread a quantity of str shavings, or excelsior on ground first, on which dynamite is placed. A chain paper, or other material, is then led away from the dynamite to such a distan may be lighted without danger of flame reaching the dynamite before operator position of safety, which should be 400 or 500 ft distant. Explosions sometin even with care, and operator should never remain near the burning explosiv powder may be destroyed by pouring it into a stream or large body of water; t part quickly dissolves and remainder becomes harmless. Cases which have dynamite are dangerous; they should not be used again for any purpose, but burned, using same precautions as described above for destroying damaged dy

To destroy damaged blasting caps, place them, not more than 500 at a hole 2 or 3 ft deep, preferably in sand. Prime a small dynamite cartridge wi electric blasting cap, and place it in contact with the damaged detonators. Fil with sand, and fire with a blasting machine from distance of about 200 ft. D ordinary caps and fuse, as there is danger of premature explosion from spark great care is taken. Before being destroyed, electric blasting caps should h wires cut off an inch or two from capsule, as the wires are liable to cushion the s prevent complete explosion of all the caps. Observe utmost caution in handling caps, as they are extremely sensitive to shock, friction, heat, and sparks.

Precautionary Rules:

Don't forget the nature of explosives; but remember that with proper care be handled with comparative safety.

Don't smoke while handling explosives, and don't handle them near an ope

Don't leave explosives in a field where cattle can get at them. Cattle like tast and saltpetre in explosives, but the other ingredients may make them sick or kill

Don't carry loose caps in the clothing. Keep them in their boxes.

Don't tap or attempt to open a blasting cap or electric blasting cap.

Don't try to withdraw wires from an electric blasting cap.

Don't attempt to take caps from the box by inserting a wire, nail, or oth metallic instrument.

Storing

Don't store or transport blasting caps or electric blasting caps with high explo

Don't store fuse in a hot place, as this may dry it out so that uncoiling will bre

Don't allow priming (the placing of the detonator in dynamite) to be done in house or magazine.

Don't leave explosives, caps, or blasting machines in a wet or damp place. K suitable, dry place, under lock and key, and where children or irresponsible can not get at them.

Thawing

...e frozen or chilled explosives; it is dangerous and wasteful.
...e any arrangement for thawing dynamite other than those recommended by ... manufacturers.
...w dynamite on heated stoves, rocks, sand, bricks, or metal, nor in an oven; ...dynamite in front of, near, or over, a steam boiler, forge, or fire of any kind.
...eat thawing-house with pipes containing steam under pressure; high tem... dangerous and escaping steam may spoil the explosive.
...ace a hot-water thawer over a fire; never put dynamite into hot water, nor ... come in contact with steam.

8. Charging and Firing Explosives

(See also Sec 5, Art 5, and Sec 6, Arts 8 to 11.)

...ng holes. A drill-hole may be "straight" or "sprung." A straight ... one which is loaded and fired without enlarging. A sprung hole is en... the bottom by exploding in it one, two, or more, successive charges. ...harges are usually not tamped (stemmed). The first charge usually ... of one or two cartridges of dynamite, increased in subsequent charges ...mber is large enough to hold required quantity of explosive. This ... is known as "springing," "chambering," "squibbing," or "burn... Except in soft rocks, slow-acting dynamites are better for springing than ...-acting, as more of the rock is thrown out of the hole and there is less ... to cave and choke up. If sides of hole are very rough, final charge may ... through a tube of brass, tin, or galvanized iron, about 2 ft longer than ... and as large as will fit into it. This prevents cartridges from being ... or smeared along sides of hole. Where a loading tube is not avail... cartridges are usually attached to a sharpened stick, lowered to bottom ... and shaken off.

...ing is the placing of the detonator (ordinary or electric blasting cap) in ...rite cartridge; or placing electric squib or electric cap in a cartridge of ... powder. In priming high explosives, place detonator so that its closed ... its towards bulk of the explosive; that is, if primer is inserted last, the ... should be so placed that its closed end points toward bottom of hole. ...ctor should be so secured that it will not change its position, nor jam ...s of hole, nor come in contact with tamping stick. The wires of an ...ting cap should not be secured in a half hitch; for, when tension is ... current may be short-circuited where wires cross each other. In ... dynamite with cap and fuse, the fuse should never be "laced" or run ... cartridge, because the "side spit" of fuse will often ignite dynamite, ... which will burn, decreasing efficiency of charge and producing noxious ... Top of cap should be imbedded ½ in deep in the dynamite, to cushion ... tamping stick. This small length of fuse will not side-spit before cap ...

...ging. Eliminate all air spaces by slitting cartridges lengthwise with a ...fe and pressing them firmly home, so that they expand and entirely ...ole. Exceptions to this are: (*a*) Blasting gelatin should not be slit, ... plastic that it can not be rammed solidly like other dynamites; (*b*) in ... of coal an air space is purposely left to cushion action of explosive ... prevent undue shattering. Dynamite cartridges, especially gelatins, ...ever be broken, if there is a possibility that some portion of the ex... frozen. They may appear soft on outside and yet have a frozen core. ... such a cartridge may explode it; accidents due to this practice have ...gulation in many foreign countries.

Tamping is necessary for all explosives to develop their full power, required in practically all work except springing a hole (see above). sand, and loam make best tamping. Broken rock, screenings, and o serve fairly well, but are liable to break or cut the fuse or wires.

Wiring for electric blasting. There are three general methods: parallel, and parallel-series connection.

In series connection (Fig 12) one of the wires from first drill-hole is connecte leading or firing line. The other wire is then connected to one of the wires from hole and the other wire of tha of the wires of third hole, and the last hole; the remaining f from that is connected to th leading or firing line. Series tion is necessary when firi ordinary blasting machine. T rent required for series conne at least 1.5 amperes, and the sufficient to overcome resist electric caps. Resistance var length of wires. About one

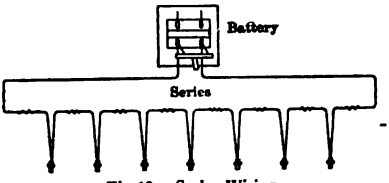


Fig 12. Series Wiring

required for each cap connected in series, although an excess, up to about 440 volt harmful. Too high voltage may cause misfires from short circuits across the ca especially when more than one cap is used in a hole. Direct current is general but alternating current is equally efficient when of a frequency of 60 cycles o and can be used down to 25 cycles. Alternating currents of lower frequency may cause trouble from misfires of the less sensitive caps in the circuit.

In parallel wiring (Fig 13) one wire from each cap is connected to one leading line, and the second cap wire to the other leading line. This method can be used only where a power or lighting current is available, having 1.5 amperes for each

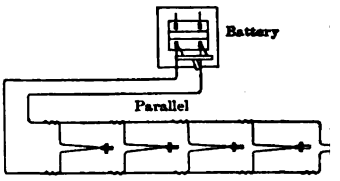


Fig 13. Parallel Wiring

cap so connected. Thus, with 20 caps, firing circuit must have at least 30 a The voltage required is very low. The greater the number of caps in circuit, th the resistance. Assuming resistance of leading wire at 3 ohms and resistance c cap at 1 ohm, the resistance caps in parallel is $(1 + 20) + 3$ or 3.05 ohms. Parallel conn of ordinary electric caps a often used, on account of lar ume of current required.

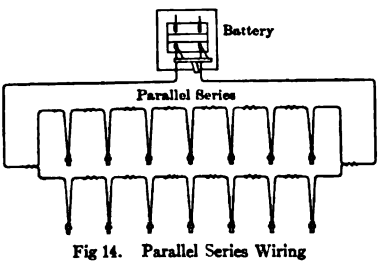


Fig 14. Parallel Series Wiring

In parallel-series wiring (the caps are first connected in of say from 4 to 10, and each thus connected is in turn at at its two free ends to the l wires. The current requi found by multiplying the nun series by 1.5, which gives the c in amperes. To determine re voltage, multiply resistance of each cap by number of caps in each series, and divid by total number of series. This system is used frequently for firing a large numl charges, where power or lighting current is available.

Electric firing may be done with a blasting machine, or a power or lig current. For firing *with machine*, the caps *must be connected in series*. E

should be scraped bright and clean and twisted tightly together, and if water be present, covered with insulating tape. The two remaining free from the two caps at ends of series are then connected to the leading wires, should be bent or hooked at end, to prevent the smaller wire from slipping

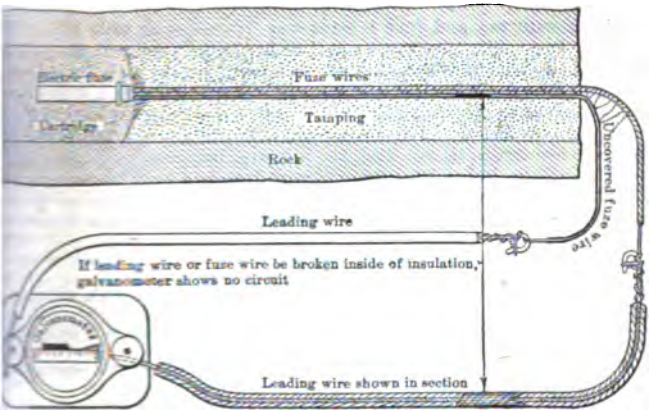

Fuse wires
Tamping
Rock
Leading wire
Uncovered fuse wire
If leading wire or fuse wire be broken inside of insulation, galvanometer shows no circuit
Leading wire shown in section

Fig 15. Poor Contacts in Electric Circuit

wires are subjected to strain. Test circuit with a circuit tester (gal- at ends of leading wires which are to be attached to battery (Fig 15). to be sure there are no broken connections or short circuits. Poor con- connections give abnormally *high resistance* readings. Looped connec-

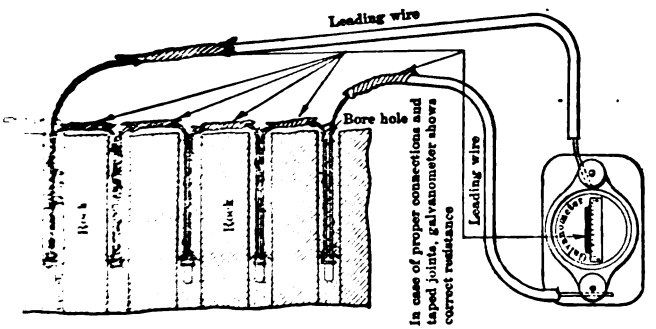


Fig 16. Testing Circuit with Galvanometer

wires may show *no circuit* one moment and *normal resistance* the next. spliced wires touch each other and make a short circuit, *no resistance* is Fig 16). The ends of leading wires are inserted into the binding posts of machine and firmly secured by thumb nuts. Place blasting machine spot (a dry board or plank is best), to prevent its tipping over, and handle with both hands and full force.

If firing is by means of a power or lighting circuit, use a special switch design as to show at a glance whether circuit is open or closed. Avoid cated switches, especially those having springs. The switch should be structed that it can be locked in the open position. The cut-outs on should be of ample capacity.

Firing with cap and fuse is often done by taking an extra piece 2 or 3 ft long, and cutting notches in it with a knife at intervals of abo The end of this fuse is then lighted, and, when powder train burns t notched place, the flame spits out vigorously. By directing each of thes against end of the fuse to be lighted a round of shots may be lighted w tainty, in a few seconds. This method may inflict disagreeable burns on t hands unless care is taken. Lighting fuse with hat lamp or candle is as the spit of the ignited fuse may extinguish flame and leave blaster in th The surest and safest method is to slit the fuse at the end to expose powde and light it with the glowing end of a burning cotton rope, such as windo or clothes line.

Charging

Don't tamp with iron or steel bars. Use only a wooden tamping stick, with n parts.

Don't force a primed cartridge into a drill-hole. Have hole drilled of ample cartridge.

Don't cut or break a dynamite cartridge while frozen, and don't rub a cartridg hands to complete thawing.

Don't prime dynamite cartridges, nor charge nor connect drill-holes for electri during immediate approach or progress of a thunderstorm.

Don't fasten a blasting cap to the fuse with the teeth, nor by flattening it with use a cap crimper.

Don't attempt to use electric blasting caps with ordinary insulation in very we For this purpose secure waterproof caps.

Don't handle fuse carelessly in cold weather; when cold it is stiff and cracks ea

Don't "lace" fuse through dynamite cartridges. This practice is frequently resp for burning the charge.

Don't cut fuse short to save blasting time. It is dangerous economy.

Don't use fuse that has been injured by falling rock or in other manner.

Firing

Don't explode a charge before every one is well beyond danger zone and protect flying débris. Protect supply of explosives also from this source of danger.

Don't explode a charge to chamber a drill-hole and then immediately reload it, hole will be hot and second charge may explode prematurely.

Don't use a "permissible" powder in same drill-hole with another explosive.

Don't hurry in seeking explanation for a misfire.

Don't drill, bore, nor pick out a charge which has failed to explode. Drill and another hole at least 2 ft from the missed one.

Don't expect high explosives to do good work if you try to explode them with a ator weaker than No 6.

9. Special Uses for Explosives

High explosives in coal mining. Use of the "permissible explosive defined by testing station of the Bureau of Mines (19), is increasing in anthracite and bituminous fields. They were used at first in gaseous and mines solely as a safety precaution, because of their comparative freedom liability to ignite gas and dust mixtures. It is now recognized that, by gent use of these explosives, the softest coal is shattered as slightly as with est grades of black powder (23). Permissible explosives are used for rock in hard-coal mines, and in bituminous mines for coking coals, where gas p

encountered. Different kinds of permissible explosives should never be in same hole.

Ore mining. The best type of explosive for a given case can be determined by experience. It depends not only on character of ore, but on ventilation and cost of labor. It is often feasible to substitute with satisfactory results a less expensive explosive for a more costly one, in mines where the miners may be unwilling to adopt methods of drilling, charging, and firing other than those to which they are accustomed. Thus, in soft hematite, a slow-acting ammonia powder properly primed, loaded, and confined, will often break more ore per shift than a more expensive dynamite handled unintelligently. The use of short fuse, lacing fuse through cartridge, and insufficient tamping are dangerous extravagant practices, often difficult to eradicate (18, 20).

Cutting or tunneling. It is desirable to "pull the cut" at the first shot; hence it often pays to use a stronger explosive for firing the cut than is required for the relief, rib, and lifter holes. It is impossible to increase effectiveness of a round by loading the cut-holes by increasing amount of explosive beyond a certain point because a definite amount of tamping is required to prevent charge from blowing out. Generally, cut-holes require an explosive having strength and density of 60% gelatin. Blasting gelatin is sometimes the most economical for this purpose, in spite of its relatively high cost. In refractory rock, or one with irregular stratification, cut-holes are often blasted more satisfactorily by electricity than with cap and fuse. Delay electric caps (Art 10) are advantageous where cut-holes bottom up well; but they should not be used in same round as cut-hole shots, if latter require to be loaded and fired a second time.

Stoping. Every mine is an individual problem. The principal point regarding the explosive is the size to which it is desired to break the material. For coarse, ammonia dynamite breaks fine enough for easy handling without undue shattering. Where ventilation is poor, gelatin or ammonia dynamites are necessary regardless of their other properties, and with much water, gelatin is best. Ammonia dynamite has added advantage that it will "stick" in "uppers." If it is to blast out dry timbers in old stopes, to allow the top to cave and fill the worked-out spaces, a permissible explosive should be used, to avoid possibility of gas.

When firing with cap and fuse, the LEAD (difference in length of fuse in holes intended to fire in succession) should never be less than 10%. For instance, if the last of such a round of holes has a 5-ft fuse, the hole to be fired next should have a fuse at least 6 in longer. With less "lead" than this, variation in burning speed of even the best fuse may cause holes to fire out of order, generally spoiling the shot. When fuses are not lighted in their proper order, the necessity for increased "lead" is apparent. When firing holes in rotation the priming cartridge is sometimes placed at or near bottom of hole, so as to have all fuses lighted at a safe distance inside the holes, when first shot explodes; then, if the fuse in a hole is shot off, its fuse will not be cut, causing misfire. This practice is used with ammonia dynamite or gelatin dynamite, but never with straight nitrates, which are readily ignited by the least side-spit from a fuse and are more liable to burn in the hole. EFFICIENT TAMPING SHOULD BE INSISTED on. It produces greater effect, and the more complete the detonation, the fewer will be the fumes. Ready-made paper tamping bags are a convenience; their use leads miners to exercise more care in tamping.

Tunnel driving. The most satisfactory explosives are the gelatins, which have maximum density and water resistance, and produce minimum of fumes. During the heading, the charge must be concentrated in the bottom of the

holes, to leave room for sufficient tamping to insure a clean break. Th
are usually wet, especially in the bench, and in dry holes or uppers, g
desirable because it can be relied upon to "stick." Ventilation in tu
generally poor and the fumes from gelatins are the least noxious of
explosives. For heading cut-holes, 60 or 75% gelatin or blasting g
generally used; for relief, rib, roof, floor, and bench shots, 40% gelatin is
strong enough. In soft rock, and small tunnels or drifts, a good meth
drill a hole straight into center of face, spring or chamber it once or tw
load for final shot with a fairly large charge of blasting gelatin, well tamp
moist sand.

Quarrying dimension stone, including stone for fills, rip-rapping, and crit
well as for building. Use slow-acting explosives; the more powerful, quick-ac
plosives shatter the stone. Granular, and other slow-acting powders, such as
freezing ammonia grades, may be used in holes where an air space is left for st
line of fracture. For subsequent blasting, black powder is best, fired by electri
ordinary fuse, or electricity. For large charges, blasting powder is fired with ac
by a dynamite primer.

Quarrying small stone (for crushers, cement works, kilns, etc). High e
may be used, the one best adapted to be determined by experience. Quick-ac
plosives are best for rock which readily transmits shock of explosion to a cons
distance. Hard limestone, trap, granite, etc, usually require straight N G dy
of 30% strength and upwards. In very wet holes, where explosive is immersed
hours, gelatin dynamites must be used. In dry work, and in rock which absorl
of shock of explosion, slow-acting explosives, like granular powder and low-
ammonia powders, are usually best. Quick-acting explosives should not be used
like sandstone and marl, nor slow-acting explosives in flint, granite, or the like
freezing explosives of all grades, especially low-freezing gelatins, are preferable
weather, because they do not suffer loss in efficiency as noted with the straight

Stripping. The slowest-acting high explosives are best; granulated dynami
low grades of low freezing ammonia dynamites 20 to 30% being generally us
heavy stripping holes should be sprung and fired with slowest-acting explosiv
dry work, a mixed granulated blasting powder is most economical. For moist
use granulated dynamite. If very wet, a low-grade straight dynamite. High-grad
may be used for stripping charges, but not for final charge
where holes are full of water.

Well sinking: [illegible] Where 3 or 4 holes drilled by ha
[illegible] with vertical holes
[illegible] charges of same explosive. Even if holes ar
[illegible] secure best results.

Sinking Air Shafts. Where work warrants the expense, best explos
[illegible] have as required, stru
[illegible] where no other explosive will
[illegible] straight dy
[illegible] wet day fre
[illegible] made with slight danger
[illegible] be done w
[illegible] when shot is fired.
[illegible] drawing a wheel from a
[illegible] mud with small ch

Road Building. [illegible] manually in rock excav
[illegible] dynamite and in wet
[illegible] ammonia for soft rock
[illegible] their effect e
[illegible] ammonia dyna
[illegible] for the
[illegible] center,

... with not more than two cartridges of 1¼ by 8-in dynamite. This charge will ... deep pot holes and material can be handled with horse scraper without miring ... For blasting boulders, block-holing is most economical, and 40% low- ... dynamite is suitable. If time is more important than economy of explosive, ... can be broken quicker by mud-capping or "adobe shots," with 40 to 60% ... dynamite.

Submarine blasting. Though gelatin dynamites withstand action of ... better than other high explosives (blasting gelatin stands water almost ... tely), straight N G dynamites are usually preferred for submarine ex- ... because of their greater sensitiveness. The temperature of the water ... ly so low that gelatins, less sensitive than straight dynamite under the ... nditions, are very difficult to explode completely. Hence, holes are ... rock is not broken, and unexploded gelatin is found by the dredges. ... dynamite is sufficiently sensitive to explode by concussion from an ... ble, if holes are not more than 4 or 5 ft apart. Consequently, in case ... of one of the electric caps or connections, all the holes will be exploded ... In submarine work, holes are generally untamped, except by ... and the quicker-acting straight dynamite therefore does better work. ... cartridges should just fit the holes; especially necessary for shallow holes.

Miscellaneous uses: clearing land of stumps and boulders; ditching and draining ... breaking log and ice jams; destroying wrecks; cutting off pilings; breaking ... tree planting; hardpan and subsoil blasting; digging holes for posts and poles; ... ing for foundations and cellars; trenching for tiling and pipe lines; breaking ... re and other materials; loosening frozen material in railroad cars; tearing down ... ings; splitting logs for railroad ties, fence rails, etc; cutting off large fires; ... snow slides; breaking old building foundations; blasting old mine timbers; ... ing forest fires.

Black blasting powder in coal mining. Black powder has so long been ... coal mining that, as regards execution only, it is considered best for this ... Most coal miners are so familiar with its use, and good miners can judge ... curately, that excellent results are usually obtained with it (15). The slow ... action of black powder produces a large percentage of lump coal. Be- ... of its bulk, it can be charged advantageously and estimated so closely that

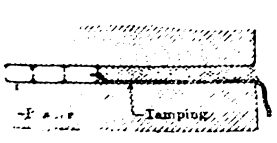


Fig 17. Blasting in Coal with Black Powder and Fuse

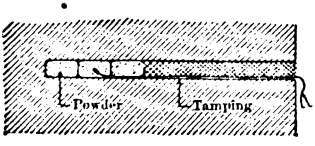


Fig 18. Blasting in Coal with Black Powder Middle Cartridge Primed with Electric Squib

... generally be used more economically than other powders. But, as black ... is loose, considerable care and judgment are required to get best results. ... powder must be made up by the miner in paper cartridges or shells of ... size. In doing this, observe following points:

... cartridge of proper diameter to slip into the hole without too much ... space.

... cartridge of proper length to hold just the quantity of powder necessary.

... the powder down into the shell, to compact it, to minimize air-space ... the full force.

... cartridge is placed in hole it must be well pushed back, unless an air- ... desired to cushion force of explosion.

... electric squib (Fig 17 and 18) is fastened in cartridge to ignite powder,

or a "needle" (Fig 19) or a "blasting barrel" (if hole is wet) is inserted powder, and the tamping compacted firmly around it. The more charge is confined, the greater t developed. When unconfined simply burn (15).

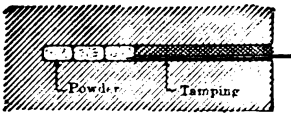


Fig 19. Miner's Needle, for Squib

If a miner's squib (Art 10) pull out needle and insert squi end first, into hole made by need the blasting barrel, and ignite sm of squib. The squib burns for seconds, and then shoots back into the powder, igniting it. Use right tamping, so that needle-hole will be smooth, or, if barrel is used (¼- pipe), see that the hole in it is clear and clean.

When black powder is ignited with squib or fuse, the force seems to through seams of the coal, displacing it forcibly. It does not exert a sharp and therefore does not produce much fines, if proper granulation of powder and correct quantity be used. When very slow action is desired an air-space is left, either between powder and tamping or around cartridge (Fig 20). The proper granulation of black powder is determined only by knowledge of the powder and the coal, and by trial. A test is necessary to determine conclusively which size of grain is best for any particular coal.

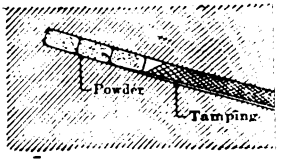


Fig 20. Blasting in Coal. Air S between Charge and Tamping

Earth-work and soft-ore mining. In blasting soft iron ore in op work, holes are drilled by steam drill, or by hand churn drill operated by men. Holes are usually sprung with dynamite and fired with black po Miniature tunnels, gopher holes or coyote holes, as they are sometimes calle used in the stripping operations, and are also fired with black powder underground mining of soft iron ore, the approved method is to fire a c hole, after it has been sprung several times, and then square up with relief a holes. In this method, the lower strengths of low-freezing ammonia dyn are preferred to other explosive. In harder ore the V-cut is used, with ammonia dynamite.

Railroad work. In cutting through fairly solid rock, the holes are usually 18 o deep, spaced 8 ft apart; they are sprung with 50% straight dynamite until each ho hold sufficient explosive to break material small enough to be handled by steam s Roughly, from 25 to 75 lb explosive per hole is used in rock of average hardness it is economical to fire simultaneously as many holes as possible, a large-size bl machine, or a power or lighting circuit, should be used for firing. For soft rock, clay, loam, or sand, it is economical to use a power churn drill. This makes a 4 t hole, of any required depth, usually 40 to 100 ft; holes usually spaced 15 or 20 ft The holes are often sprung with dynamite, and after thorough cooling, usually over are charged with black powder, sometimes several tons in a hole. Black powd granulated dynamites can be used only when holes are dry, and with great care working near steam shovels, locomotives, etc, as many accidents have occurred sparks dropping into black powder. If work is wet and sparks can not be avoided, fairly low-grade low-freezing ammonia dynamite. In firing simultaneously a large ber of holes (50 to 100), use waterproof electric caps, to prevent leakage of current th rock, with attendant chance of misfires.

10. Blasting Supplies

Blasting caps. A cap is a copper cylinder, closed at one end, containing a charge of detonating composition, and is fired by a fuse. Caps are ... according to quantity of detonating composition contained (Table 4).

Table 4. Fulminate of Mercury Blasting Caps

For Fuse Firing

Grade	No 5	No 6	No 7	No 8
... of shell, in	1.250	1.375	1.625	1.875
... of shell "	0.234	0.234	0.234	0.234
... of charge, grains	12.340	15.430	23.150	30.860
" " grams	0.800	1.000	1.500	2.000

For Electric Firing

Grade	No 6	No 7	No 8
... of shell, in	1.562	1.750	2.000
... of shell "	0.273	0.273	0.273
... of charge, grains	15.430	23.150	30.860
" " grams	1.000	1.500	2.000

Note The above tables refer only to caps charged with fulminate of mercury consisting usually of 80% fulminate of mercury and 20% chlorate of potash), ... the standard against which other detonating compounds are graded. Other ... are also used, in charges of such amount that the detonating effect is equal ... of corresponding grade of fulminate composition.

... high explosives, the stronger the cap the better the execution, as a rule. ... detonators should be used for tunneling, shaft-sinking and similar work; ... charges sometimes require No 8. Caps should never be crimped on fuse ... with special crimpers made for purpose; biting them, or nicking them ... knife, is neither efficient nor safe. They should be stored in a dry place, ... nature weakens their force. DO NOT ATTEMPT TO EXTRACT COMPOSITION ... CAP SHELLS; it is exceedingly sensitive and is often detonated if scratched ... out with a pin or similar instrument. Miners should not wear oil or ... hat-lamps when handling caps; many accidents have occurred from ... falling into a box of caps.

Electric blasting caps (fuzes) (Fig 21). An electric cap consists of a ... shell (*A*), 1⅜6 to 2 in long by 0.273 in diameter, closed at one end. It ... a charge of detonating composi- ... *B*, in which is embedded a fine plati- ... wire (*E*) connecting the two copper ... (*D*). Wires (*D*) are held in place and ... insulated from each other by a composition ... or plug (*F*), which, in turn, is retained ... composition (*G*) held by corrugations in shell. Electric caps are used for ... in gaseous and dusty collieries, and for firing charges simultaneously, thus ... ensuring explosive. ONLY ONE KIND OR BRAND OF ELECTRIC CAPS SHOULD BE ... FIRED IN ONE SERIES. Different brands vary in sensitiveness, and if the ... in a series are not uniform, the least sensitive will probably misfire.

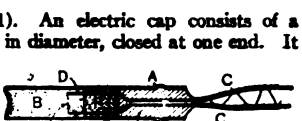


Fig 21. Electric Blasting Cap

Delay electric blasting caps are for firing blasts in 2 or 3 volleys application of electric current. They are used with a special detonator, "No Delay," which is necessary because first and second-period dela caps are less sensitive to ignition from passage of electric current, and can not be used in series with ordinary electric caps. When current mitted, about one second elapses before first-delay caps detonate, a period between these and second-delay. These detonators are useful i driving and are especial mended for shaft-sinki enabling blaster to fi round without returnin

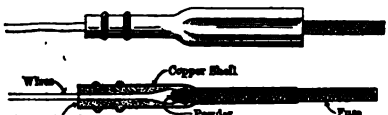


Fig 22. Electric Fuse Igniter

Electric fuse ignit 22) are devices for ig fuse, usually of high-gra proof quality, by electric

When shipped they are not attached to fuses, but are crimped on wh The interval between the operation of blasting machine (or turning on rent) and the firing of the cap, depends on length of the fuse attached.

Special electric blasting caps. For very wet work, and where sludge a possess high conductivity, special insulation of cap wires is necessary to preven leaking from wires at one end of series to wires at other end, thus forming a shur wires at middle of circuit. This condition can be detected by making resistance of firing circuit on a direct-reading ohm-meter. If reading is the same, there is age; if there is a drop in resistance, after loading in wet holes, there is liability c leakage, indicating necessity for special waterproof cap wires. For firing cl deep water, a special, highly waterproof electric cap is made. There are oth fications for various purposes, such as having wires of larger gage than ord decrease resistance in deep-hole blasting. Electric caps are also made with ir used where only 2 or 3 shots are fired simultaneously. These have much higher resistance than caps with copper wires, and are not recommended for lengths o Electric caps with tin-coated copper wires are used in certain mines where it is ol able to have particles of bare copper in material mined, and where number of fired simultaneously makes use of iron wires impracticable.

Electric squibs (Fig 23) are somewhat similar to electric caps, exce the shell is of heavy paper instead of copper, and cap filling is fine-graine powder instead of a detonating compound. They are especially designed for black powder; can not be used for high explosives. They possess advantage of simultaneously firing several charges, and of permitting more perfect confinement of charge than with ordinary powder squibs; also, the charge can be ignited in middle, giving a little quicker and stronger action and ir explosion of entire charge before any portion can be cut off by a fall of sur ing material. Electric squibs are safer than fuse or ordinary miners' squi cause shots are not fired until every one, including blaster, is at a safe di and hang-fires are entirely prevented. They should always be stored place, being readily affected by moisture. Electric squibs are made witl iron and copper wires, those with iron wire being less expensive but req stronger current (see preceding paragraph).

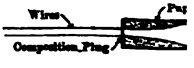


Fig 23. Electric Sc

Miners' squibs are for firing black powder only. The squib consists of a core of p composition tightly rolled in paper; one end terminates in a slow match, made by d twisted end of paper in melted sulphur or other combustible. In using, the squib in mouth of hole formed by withdrawal of needle, or in the blasting barrel (Art 9). other end of hole so formed terminates in the charge. Outer end of squib is lighte

several seconds until its powder core is ignited, whereupon it shoots down into the and ignites that. Time between lighting of fuse and firing of charge can be varied to some extent by position of tail of squib: when turned up it burns more slowly; turned down, much faster. The squib is very cheap, but not so safe as ordinary since it requires an opening through which to travel, the charge is not confined so as with fuse or electric squib.

Safety fuse consists of a train or core of a special kind of powder, tightly wrapped in successive turns of hemp, jute, or cotton yarn, and tape, made more or less waterproof by addition of asphalt or other varnish, or gutta-percha. Usually made to burn, when unconfined, at rate of about 2 ft per min. when tightly tamped, so that gases from burning powder train can not escape, the pressure causes fuse to burn faster. For dry work, hemp fuse is good for powder, but is too small in diameter properly to fit standard caps. The cheaper grades are usually the least waterproof; the more expensive, the better to resist water. For very wet work, or under water, gutta-percha fuse will serve. When extra precautions are necessary, the end of fuse and the cap may be dipped into asphalt paint and dried, or joint between cap and fuse covered with tallow or soft soap. Do not use oil or grease, which is likely to affect powder train by dissolving asphalt varnish. As the powder in a fuse absorbs moisture, always cut off an inch or two from end, before putting in cap. Cut off end square across, push into cap without twisting until fuse touches the cap charge. If fuse is cut at an angle, pointed end may bend over and by covering end of powder train, cause misfire. Nearly every kind of fuse spits out of sides more or less in burning, and therefore should not be placed in the dynamite. When necessary to have priming cartridge at bottom of hole, a fuse should be selected that will spit from the sides as little as possible, as cartridge shells must not be slit; fuse is less likely to ignite dynamite through paper wrapper.

Blasting machines. The usual form is a small dynamo (Fig 24), the armature of which is rotated by a rack-bar and pinion. There are two types, in which rack-

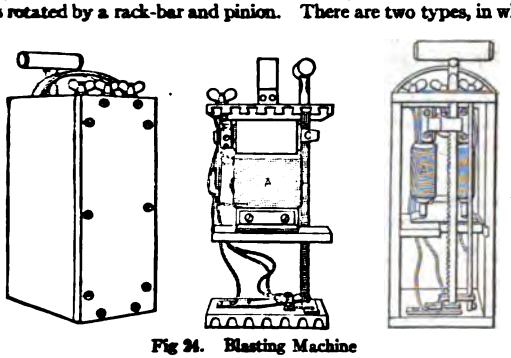


Fig 24. Blasting Machine

bar is either pushed down or pulled up, the dynamo being same in both; the push-down machine is preferred. The current generated during descent of rack-bar is short-circuited through field magnets until end of stroke; then the end of rack-bar strikes spring of circuit breaker, and shunts current through binding posts to external circuit or firing line. Place machine in a firm, level position, and operate with both hands and full force; an attempt to operate it with one hand, or in a half-hearted way, will often result in misfires from in-

sufficient current. Keep blasting machines in dry, cool place. The c
tor, brushes, and circuit-breaking contact points should be kept clear
and free from oil. Oil bearings and gears occasionally.

Circuit testers (galvanometers) are of two general types: one merely
whether circuit is open or closed; the other is essentially a small direct-rea
meter, indicating by movement of a needle across graduated scale the approxim
ance of blasting circuit in ohms. With the latter, it may be determined wheth
blasting circuit is complete, or broken, or short-circuited. By a table, giv
showing resistance of electric caps with different lengths and sizes of connecting
ing wire, the exact condition of blasting circuit at moment of firing may be d
fairly accurately. The circuit-tester is a valuable adjunct, and, where a co
amount of electric blasting is done, should form part of blaster's equipment. I
to breaks and short circuits, it also detects leakage of current through ground,
pipes, steam pipes, and imperfect connections. The instrument is furnished wi
chloride cell, which is constant in its current output. The current thereby gene
weak that the danger of firing an electric cap while testing is remote; but, as a
precaution, tests should be made from a safe distance. The ordinary type
reading ohm-meter and battery tester, containing carbon-zinc dry-cell ba
liable to send sufficient current through a blasting circuit to explode an electric
greater caution is therefore necessary in using them, especially for testing one
time.

Blasting machine testers (rheostats). These are for determining inexpens
capacity and condition of a blasting machine. There are several types. One p
means of sending a current through different resistances and a small lamp, so th
connected to the poles corresponding to type of blasting machine tested, a bri
indicates that machine is up to standard. Another tester has 6 posts, with
resistances so arranged that 20 combinations of varying resistance may be ob
connection with use of an electric blasting cap in series, acting as an indicato
thus possible to determine whether blasting machine is up to strength, and if
how many caps in circuit it is capable of firing. By its use overloading a given
machine is avoided, with consequent danger of insufficient current and misfires.

Connecting and leading wire. Connecting wire is used for connect
electric cap wires of one hole to wires of cap in an adjacent hole. As
dealers in blasting supplies, it is usually No 20 or No 21 B & S gage, wo
1 and 2-lb spools. The use of the larger gage wire is advisable, as it ad
to the resistance of firing circuit. This wire should not be used for con
a line of holes to blasting machine; for that purpose LEADING WIRE sho
used, of No 14 gage or larger. No 14 B & S gage wire, in coils of 50
satisfactory for all kinds of dry work.

Resistance tables. The resistance of copper wire, B & S gage, per 1 000 ft,
usually employed in electric blasting, is:

Gage No	Ohms	
8	0 6271	Power and lighting circuit
10	0 9972	
12	1 586	
14	2 521	Leading wire
16	4 009	Sometimes used for leading wire, but not recommended for firing large circuits
18	6 374	
20	10 14	Connecting wire
21	12 78	
22	16 12	Size attached to electric caps

Wires of electric caps have a resistance of 0.032 ohm per ft (doubled).
platinum bridge of each electric cap has a resistance of 0 859 ohm. Th
distance of iron wire, of size used for iron-wire electric caps, is about 101.0
per 1 000 ft.

Table 5. Resistances of Electric Blasting Caps with their Wires

Length of wire, feet	*Resistance of cap with copper wires, ohms	Resistance of cap with iron wires, ohms	*Length of wire, feet	*Resistance of cap with copper wires, ohms
4	0.987	1.661	20	1.499
5	.....	1.861	22	1.563
6	1.051	2.062	24	1.627
8	1.115	2.463	26	1.691
10	1.179	.....	28	1.755
12	1.243	.....	30	1.819
14	1.307	.....	40	2.139
16	1.371	.....	50	2.459
18	1.435	.....	.....	.....

* These figures, from the E. I. duPont de Nemours Powder Co, apply to products manufactured by that concern. They are probably approximately correct for most other makes of electric caps.

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SECTION 5

ROCK EXCAVATION

BY

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Note. — Numbers in parentheses refer to Bibliography at end of this section.

This section contains data on blasting in general, and surface excavation of rock, as ... mining, quarrying, railroad and highway through-cuts, and side-hill cuts, and ... Related subjects are: Tunneling (Sec 6), Shaft Sinking (Sec 7), Machine ... and Compressors (Sec 15) and Explosives (Sec 4).

1. Cost as Affected by Character and Formation of Rock

Rate of drilling is influenced chiefly by the hardness of rock. In general, ... speed in hard rocks is much lower than in soft, though, in some shales ... other friable rocks, the accumulation of sludge in the hole prevents the ... from striking an effective blow, and retards drilling. This is remedied by ... a water-jet (Art 2). In some soft rocks a heavy blow will drive in the ... that it sticks. Seamy, blocky rock also causes sticking or FITCHERING; ... overcome by withdrawing the bit and dropping a handful of quartz ... cast-iron fragments into the hole. TOUGHNESS AND HARDNESS of the rock ... determine in great measure the kind and amount of explosive required. ... cohesive rocks require more powerful explosives and in greater quantities ... soft, brittle rocks. DIP OR SLOPE of the rock strata and presence or absence ... seams also affect cost of excavation. Rock strata may lie at such an angle, ... seams be so located, that the amount of drilling and explosive required is greatly lessened. On the other hand, these conditions may require the excavation of rock outside the desired cross-section.

Overbreakage for 8 months during 1909 in open-cut work on the Livingstone i ment of the Detroit River was 14.7%; 273 750 cu yd of limestone excavation v for, and 314 000 cu yd loosened (11). Overbreakage in open cuts (mostly in g on Grand Trunk Pacific R R, was 10 to 40% (20). Overbreakage in the approac tunnel near Peekskill was 10%, the strata dipping at a high angle (20).

Size into which rock breaks when blasted determines method dling, and influences cost.

Voids in hard rock, when broken by a crusher, amount to about 35% if all s mixed and the stone slightly shaken, but, if screened, each size has 45 to 48% Soft, friable rocks, as shales, break into widely varying sizes and therefore have percentage of voids. Hard rock blasted in large pieces and thrown into cars ha 40 to 45% voids, 1 cu yd of solid rock making 1.67 to 1.82 cu yd broken.

Voids.....	30%	35%	40%	45%	50%
No of cu yd, loose measure, from 1 cu yd of solid rock	1.43	1.54	1.60	1.90	2.00

Swelling in fill. On excavating a mixture of solid and loose rock and earth, in place makes about 1.4 cu yd in fill. If rock be first stripped of earth, and then and dumped by itself, the percentage of voids is larger. At Boulder, Colo, 3 60 of solid rock made a 5 340 cu yd embankment; a ratio of 1: 1.51. In Virginia, cu yd of limestone and mica schist, broken and put in embankment, made 90 000 an increase of 80%. In subaqueous excavation, Ashtabula Harbor, O, 62 869 (place measure) gave 103 537 cu yd measured in scows, an increase of 65%.

2. Drill Steel

Octagon steel is most used. Length of CUTTING EDGE (gage) of bit from 200% of width of shank for soft rock, to say 130% for hard; chise is generally used for hand drilling, and cross, X, Z, and other shapes for m drills. The projecting ends of cutting edge are the WINGS.

Table 1. Weights, Lb per Ft, of Solid and Hollow Drill Steel (Origin

Size, in	Round		Square		Hexagon		Octagon		Crucifo	
	Solid, lb	Hollow, lb	Solid, lb	Hollow, lb	Solid, lb	Hollow, lb	Solid, lb	Hollow, lb	Solid, lb	lo
½	0.67	.....	0.85	.....	0.73	.....	0.70	.....	.....	.
9/16	0.85	.....	1.08	.....	0.93	.....	0.89	.....	.....	.
⅝	1.04	0.87	1.33	1.16	1.15	0.98	1.08	0.91	.....	.
11/16	1.26	1.09	1.61	1.44	1.39	1.22	1.32	1.15	.....	.
¾	1.48	1.31	1.91	1.74	1.66	1.49	1.56	1.39	1.33	1
13/16	1.77	1.60	2.25	2.08	1.95	1.78	1.83	1.66	1.45	1
⅞	2.03	1.86	2.60	2.43	2.25	2.08	2.12	1.95	1.62	1
15/16	2.34	2.17	2.99	2.82	2.61	2.44	2.45	2.28	1.89	1
1	2.65	2.48	3.40	3.23	2.94	2.77	2.78	2.61	2.23	2
1 1/16	3.00	2.83	3.84	3.67	3.31	3.14	3.13	2.96	2.37	2
1⅛	3.38	3.21	4.30	4 13	3.71	3.54	3.53	3.36	2.84	2
1 3/16	3.75	3.58	4.80	4.63	4.15	3.98	3.93	3.76	3.15	2
1¼	4.15	3.98	5.31	5.14	4.59	4.42	4.33	4.16	3.43	3
1 5/16	4.57	4.40	5.86	5.69	5.08	4.91	4.77	4.60	3.85	3
1⅜	5.01	4.84	6.43	6.26	5.52	5.35	5.23	5.06	4.03	3
1 7/16	5.52	5.35	7.03	6.86	6.06	5.89	5.74	5.57	4.57	4
1½	6.00	5.83	7.65	7.48	6.62	6.45	6.27	6.10	4.78	4
1 9/16	7.05	6.88	8.98	8.81	7.76	7.59	7.32	7.15	.....	..
1¾	8.18	8.01	10.41	10.24	9 00	8.83	8 64	8.47	.....	..

Note. — Size of hole in hollow steel is assumed at 0.25 in. Actually it varies size of steel and with specification.

...cial steels (Fig 1), used with Ingersoll-Rand "Jackhamer" and Sullivan "Ro-...ils. (a) Hollow hexagon steel with a 6-point rose bit (Fig 2, *L*) is recommended ...d medium-hard rocks, air being used to eject ...ings. (b) Solid spiral steel, with a 4-point or ...est for hard ground where dust raised by ...g is objectionable. (c) Hollow spiral steel, ...t, Z, or chisel bits, is good in soft or medium ...ground where it is hard to clear out the cut-... (d) Solid twisted steel, with a fishtail or Y-bit, ...ed for use in coal or soft rock, commonly ...as "auger" ground.

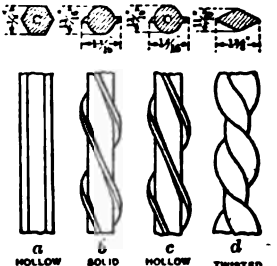


Fig 1. Special Drill Steels

Hand-drilling bits. Steel is usually ¾ to ... For soft rock, ⅝-in steel may have a 1-in ... in steel, a 1.25-in bit. Diam of the "fin-... or last bit of a set for a given hole should ... less than 0.75 in for charging dyna-... For black powder, the hole must be much ...

...el edge of a hand bit is usually convex; its faces should be at 90° for hard rock, ...° to 45° for soft; in sandstone a blunt-edge cuts faster. GAGE of bit decreases as ... shank, or depth of hole, increases. The reduction in gage depends on hardness of the rock. In soft rock, the bit wings wear slowly, and the decrease in gage is less than for hard rock. Reduction in gage is commonly 1/32 to ⅛ in per bit.

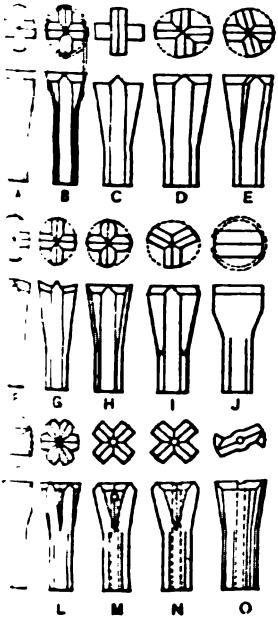


Fig 2. Shapes of Machine-drill Bits

Machine-drill bits. Fig 2 shows different shapes (T. H. Proske, *M & S Press*).

A, standard American cross-bit. *B* has an excess clearance angle (16°) and round edges (see *A*); wears rapidly on the corners, and hole tends to "rifle" (see below). *A* is stronger than *B*, and in soft rocks will remain sharp about 4 times as long. *C*, Simmons bit: 2 wings are solely for reaming and keeping the hole round; a rapid driller in hard rock. *D*, Brunton bit, which combines the easy-sharpening qualities of the cross-bit with advantages of the X. *E* is the X-bit, the wings of which must revolve 180° for the edges to strike twice in same place; this prevents RIFLING. *F* is a high-center bit, favored for hammer-drills, because it starts a hole easily; but it is apt to follow seams and cause crooked holes, and does not MUD (clear out the cuttings) freely. *G* is a fast-cutting bit, but corners dull rapidly and break easily. *H*, round-edge bit, is good in soft rock, but rifles in hard... *I*, Y-bit, muds freely as there is plenty of room for the cuttings to escape; difficult to sharpen by hand. *J*, bull or chisel-bit, formerly used for hard rock in Missouri and still sometimes used in marble quarrying; has been superseded by power-sharpened X-bit; cuts fast in seamy shale with a water jet. *K*, Z-bit, sometimes employed where hand sharpening is in use. *L*, 6-wing rosette, made with power sharpener and used ...mer-drills only. *M*, cross-bit, for hammer-drills. *N* is the same bit, with hole at ...stead of at the side. *O* is a Z-bit widely used in Germany, usually with a ...ted shank.

Table 2. Shapes and Drilling Speeds of Hammer-drill Bits (Sullivan M

Description of bit	Duration of run, min	Diam of hole, in	Aver in drilled per run, 70 lb air press	Aver in drilled per run, 85 lb air press	Remarks
25° diamond-point +	2	1⅝	6⅝	8½	Spots easily. Hole rifles Rotation easy. Gage hc
25° diamond-point ×	2	1⅝	6	7¾	Spots easily. Drills round h tation easy. Gage fairly
Convex +	2	1⅝	6¼	8¼	Spots easily. Hole rifles Rotation easy. Gage hc
Flat +	2	1⅝	6	7½	Difficult to spot hole. Hc
4° concave +	2	1⅝	5¾	8¼	badly. Rotation hard. fairly durable.
25° pointed Bull −	2	1⅝	5¾	7	Spots easily. Drills round h tation easy. Gage wears
Flat Bull −	2	1⅝	5 15/16	7¼	Difficult to spot hole. Hc badly (3 flutes). Rotati Gage wears rapidly.

Special bits for removing cuttings. A spiral steel (like a wood propels the cuttings from the hole. Cruciform steel, with lugs at inte the angles, performs a like function, when used on reciprocating drills, acting as scrapers in removing the sludge. A steel with a longitudina ber from the bit's point to a side outlet close to the chuck, also keep clear. A portion of sludge enters the chamber on the down stroke and is out by the entrance of more at each succeeding stroke. Locher tracti has large steels, with a hollow chamber and ball valve. COMPRESSED WATER forced through hollow bits aids in removing cuttings; air alone rocks makes much dust (Sec 15, Art 25; Sec 23, Art 5). Water allay and preserves temper of the bit. WATER-JET is effective. A small pip serted in the hole alongside of the bit, and a stream of water forced thr

Quantity of water for jet (20): Let Q = gal of water per min; D = diam of of drill hole, in; D' = diam of drill steel, in; d = diam of largest grain of slu Then, $Q = 7\,(D^2 - D'^2)\,\sqrt{d}$; or, neglecting area of steel, $Q = 7\,D^2\,\sqrt{d}$. Vel the rising current must exceed that at which the particles would fall in still water. chips 1.4-in diam (as in shale), require 22 gal water per min in a 2.5-in hole. quartzite, with drills operating at 60 to 65 lb air press, dry holes were drilled at per min cutting time, while holes with a water-jet under a press of 40 to 80 lb p were drilled at 0.78 to 1.34 in per min.

Drill steel sharpening is done by hand or by machine sharpeners. rocks, a bit must generally be sharpened for every 4 ft of hole; in medium for every 2 ft; in very hard rocks, for every 1.5 ft. A blacksmith and help sharpen about 140 bits per day at a cost of about 4¢ per bit. A machi sharpen up to 60 bits per hr, or say 200 to 400 bits per day. Machine-sha bits cost about 3¢, last longer, and cut better than hand-sharpened bits.

Heating. Drill steel contains 0.8 to 1% carbon; by overheating, steel is BURN becomes softer through loss of C. For sharpening, heating must not be contin long, nor carried above a cherry red. The bit and part of the shank are well be the fuel, which should be charcoal, coke, or blacksmith's coal. If the fuel contains s the steel becomes brittle when hot, or HOT SHORT; if it contains phosphorus the brittle when cold, or COLD SHORT. Oil and electric furnaces are sometimes used, gives too large a flame and electricity requires a special forge.

Bit sharpening. Bits should be constantly turned in the fire, removed ... red and dressed. The edge of a badly-worn chisel bit is first upset ... proper width. The bit is then held on the anvil at a slope of about 1 ... on, with its edge even with edge of the anvil. While hammering ... ed after each half-dozen blows. A file may be used on the hot bit for ... ing. Blows should be light and glancing, to draw the fibers of the ... arts the edge, thus toughening the metal. There are 2 methods of ... g machine-drill bits: SET-HAMMER and FULLER-AND-DOLLY. In the ... et-hammer is placed on the bevels for driving the steel back. After ... sharpened a few times a drill bit must be reformed. In the second method ... el is first drawn sharp at the corners with a fuller and then set back in ... ter with a dolly.

Machine sharpening. Power sharpeners are made by Word Bros, Ingersoll-... Co, Hardsog Drill Co, Fairbanks-Morse Co, and a number of others. ... sist essentially of 2 air-driven hammers for shaping the bit; one hori-... the other vertical. Price is from $800 to $1 200.

Tempering. After being sharpened, the bit is re-... for tempering. When cherry red it is plunged ... water for a moment to cool it partially, and then ... on a stone, or sanded board, to remove scale. ... heat advances again toward the tip, the play of ... colors should be watched in a dark corner of the ... The colors appear in following order: Pale yel-... (430° F.), straw (470°), brown (490°), purple (530°), ... blue (560°). If they do not advance parallel to the ... edge, that side on which the colors are advancing ... rapidly should be held in water. When the edge ... straw color plunge the bit into water, wave it back ... forth to insure rapid and even cooling, until steam ... to form, and then leave it in the quenching bath. ... may be of water, salt and water, rape-seed oil, tallow, or coal-tar; brine ... the drill fastest, coal-tar slowest. Fig 3 shows a bath in which a grate, set ... in below the water surface, supports the bits at the proper depth.

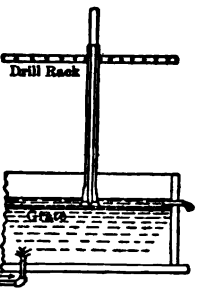


Fig 3. Drill-tempering Tank

3. Methods and Cost of Hand Drilling

Methods. SINGLE-HAND DRILLING is usually economical in the softer rocks ... depth of about 3 ft; wt of hammer, 3.5 to 4.5 lb. DOUBLE-HAND drilling is ... for deep holes or very hard rock; 2 strikers may be employed; wt of ham-... 10 lb. For 6 to 8-ft holes, the starting bit is usually 1.25 to 1.5-in gage. ... DRILL, well handled, is effective for deep vertical holes. It is raised and ... ped by one or more men.

Effect of diam of hole on the speed of drilling has never been fully deter-... ed. In general, doubling the diam divides the speed by from 2 to 4.

Direction of hole. A horiz hole is drilled at about 0.5 the speed of a verti-... down hole; in "uppers" the speed is materially less. Horiz and up holes ... usually dry, and the cuttings prevent the bit from striking the rock effec-... water poured into a down hole keeps the cuttings in suspension, per-...ng the bit to strike against a relatively clean face.

Hardness of rock as affecting speed of drilling. In vertical holes (1.5-in ... ing bit) 1 man holding and 2 men striking can in 10 hr drill 6-ft holes at ... ing rates: granite, 7 ft; trap, 11 ft; limestone, 16 ft.

Table 3. Cost of Hand-hammer Drilling (Original)

Kind of work	Kind of rock	Men per drill	Hr per day	Wages, $	Diam of bit, in: Starting	Diam of bit, in: Finishing	Depth of hole, ft	Ft per hr
Railroad cut	Hard limestone	3	10	....	....	1½	.....	1.7
Open cut	Mica schist	3	..	.....	1⅞	1¼	7.5	1.5
Open cut	Mica schist	3	10	2.00	1⅞	1¼	7.5	1.5
Side-hill cut	Gneiss	3	10	1.50	2	1¼–1½	6–16	1.41
Open cut	Hard porphyry	3	..	.....	....	.....	20 (*b*)	0.6–0.8
Railroad cut	Very hard granite	3	10	2.25	....	.....	.....	1.4
Railroad cut	Dark hornblende	3	10	2.25	1⅜	.....	12	2.9
Railroad cut	Red granite	3	10	2.25	1⅜	.....	12	2.0
Railroad cut	Trap, diabase	3	10	2.25	1⅜	.....	12	1.85
Block holes.	Red granite	3	10	2.25	1⅜	.... { av 1.25 0.5–4 }		1.04
Trench	Limestone	1	8	....	1¾	.....	.....	1.25
Trench	Shale	....	....	....	....	.....	.....	.....
Open cut	.....	....	....	....	....	.....	.....	.....
Tunnel	Gneiss, tough schist	1	10	2.25	1⅜	.....	2.0	0.35
Tunnel	Very hard mica schist	2–3	..	.....	1¼	.....	1.67	1.00
Tunnel	Conglomerate, shale	....	8	2.25	....	.....	.....	.....
Tunnel	Tough sandstone	1	8	....	0⅞	.....	1.5	1.19
Tunnel	Very hard syenite, q'tzite	2	8	....	....	.....	1.0	1.25
Mine	Augite diorite, firm red porphyry	2	10	2.00	1¾	1¼	6.0	1.48
Mine	Chalcopyrite, limestone	2	10	1.35	1¼	.....	.....	1.0
Mine	Medium rock	1	..	.....	....	.....	2.5	0.5
Tunnel	Compact phonolite dike	1	8	3.50	1¼	.....	2.5	1.0
Shaft	Compact phonolite dike	1	8	3.50	1¼	.....	2.5	1.2

(*a*) Including 3.5¢ for "nippers" (carrying drills) and 5¢ for blacksmithing. usual depth for hammer drilling; churn drilling cheaper. After reaching dep ft all 3 men used hammers, the drill turning of itself on the rebound. (*c*) Inclu for sharpening. Machine drilling cost 14¢. (*d*) Steam-drill work cost 18.8¢ pe this work, due to sticking of drills. (*e*) Machine drilling, including sharpening o cost 39¢. (*f*) Including 9¢ for sharpening steel.

Cost of hand churn drilling. 30-ft holes, 2.75 in diam at start and 1.5 in at f in blue sandstone, cost 35¢ per ft. In 10 hr, 3 men put down the first 18 ft and 4 n last 12 ft. In brown sandstone, cost was from 35 to 40¢ (labor, 15¢ per hr). In and sandstones of the coal measures, holes 20 to 24 ft deep cost 40¢ per ft. In the Range, Minn (Art 10), 4 men can drill 40 ft of 1.5-in hole in 10 hr in stripping overb and 96 ft in iron ore. With wages at $2, cost in stripping is 20¢ per ft; in ore,

Hand-power auger drills are sometimes used for prospecting and l blast holes in soft ground, as coal, slate, shale, salt, gypsum, and talc. vary from simple hand augers to elaborate machines with tripod or post m ings. Mounted machines weigh 80 to 100 lb; cost, $30 to $70.

4. Methods and Cost of Open-cut Machine Drilling

Drill mountings for open-cut work are the tripod (Sec 15, Art 29), quarry-bar, g and special carriage for deep holes (Fig 4). QUARRY BAR is a horiz bar supported a end by legs. It is 3 to 6 in diam, 8 to 12 ft long. Cost, about $160; wt, without dril to 1 500 lb. These bars are primarily for drilling in quarries a number of rows of ve holes close together, but may be used for similar work in trenches, etc. GADDER is a q device for drilling a number of parallel holes in a plane at any angle, from horiz to ver as the undercutting holes in a bench partly freed by channeling. A heavy carriage, ru on a track, has hinged to it a standard, adjustable at different angles, on which sli saddle carrying the drill. Cost of gadder frame, $400 to $450; wt, 2 500 to 3 500 lb,

trucks are used to some extent for deep-hole drilling, and for trench etc. Fig 4 shows a truck carrying a derrick and heavy drill similar to subaqueous drill-boat equipment. It is operated from a central compressed-air or steam plant, or may carry its own boiler. The drill may be stationary, or, as shown, mounted on a turntable. The cuttings are removed from the hole by a water jet or by special steels (Art 2). There are other devices for mounting one or more ordinary machine drills on a bar. The electric-air drill (Sec 15) is also furnished on a carriage. A special device for sewer work is described in Art 11. Respecting drill carriages, see also Sec 6, 15 (10, 20).

Cost of machine drilling comprises: (*a*) wages of drill crew; (*b*) proportion of wages of power-plant crew; (*c*) fuel; (*d*) drill sharpening; (*e*) repairs and renewals; (*f*) oil and water; (*g*) interest on plant; (*h*) depreciation of plant; (*i*) proportion of general expense including taxes, and (*j*) setting up, dismantling, and moving plant.

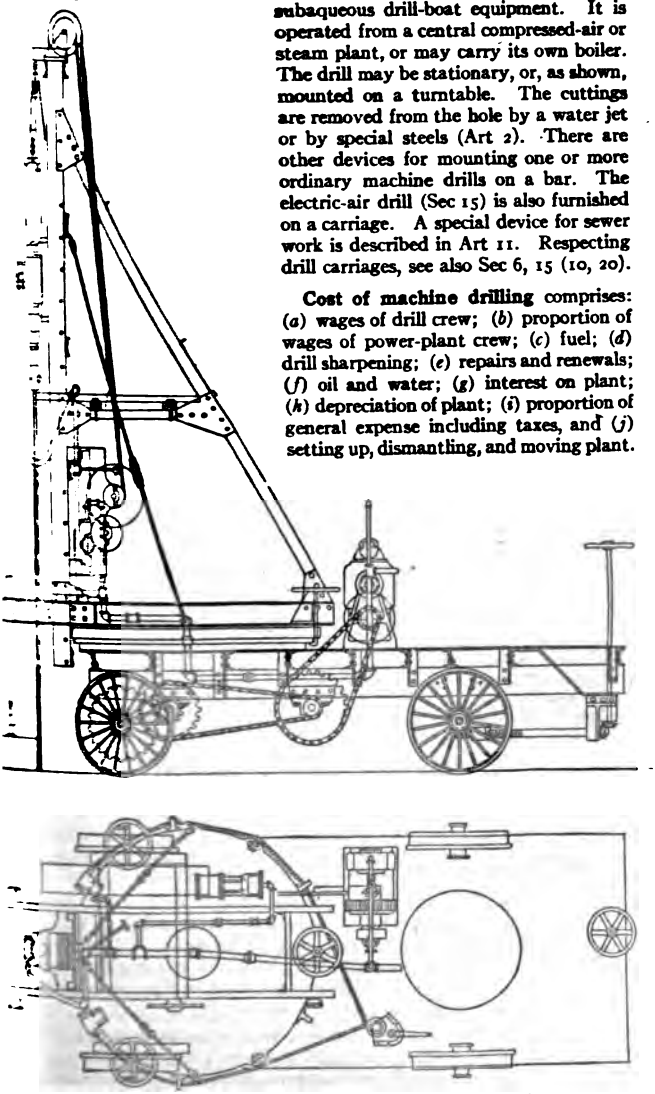


Fig 4. Ingersoll-Rand Deep-hole Drill Wagon (15-ft Turntable)

Factors affecting speed of drilling: (*a*) character of rock, as hardness, seams, sludge, and dust-forming qualities; (*b*) time for changing bits; for taking down, moving, and setting up machine; (*d*) depth of hole; (*e*) d of hole; (*f*) diam of hole; (*g*) use of air or water, or both, in the hole; (*h* of bit; (*i*) quality of blacksmithing; (*j*) percentage of time lost, as by b breakdowns, delays, etc; (*k*) size, weight, and type of drill and mounting or steam press at the drill; (*m*) the energy and skill of crew.

Time occupied by the different operations in drilling may be classed un TING TIME and DELAYS, the sum of which gives total CYCLE TIME for drilling Delays comprise time to (*a*) raise drill, (*b*) loosen bit, (*c*) remove bit, (*d*) get bail, (*e*) get bit, (*f*) insert bit in chuck, (*g*) tighten chuck, and (*h*) get started. the cycle time, there is the time required to move the drill to other holes, set it up, and miscellaneous delays. For time studies on different pieces of work, see (11 drills were mounted on tripods for holes 7 to 24 ft deep and 2.5 to 5.5 in start, in granite, limestone, and slate. From these records, cutting time average of cycle time; cycle time, 74.7% of total time; time for moving and starting 12.5% of total time; time lost in delays, 12.8% of total time. Cutting speed: in 0.18 ft per min; limestone, 0.13 ft per min; slate, 0.17 ft per min. My own recor 4 to 16 min to drill 2 ft (cutting speed 0.125 to 0.5 ft per min), 2 to 10 min to bits and pump out hole, and 7 to 60 min to set up the drill (20).

Table 4. Average Time Drilling Vertical Holes (Tripod Drill) (Origi

Kind of rock	Lm	S	Sd	Gr
Length of shift, hr	10	10	10	10
Air pressure, lb per sq in	70	10	70	80
Diam drill cylinder, in	3.25	3.25	3.25	3.25
Diam starting bit, in	2.5	2.5	2	3.5
Diam finishing bit, in	1.75	1.5	1.25	1.25
Depth of hole, ft	12	6	12	20
Drilling first 2 ft, min	9	10.5	8	12
Cranking out, removing bit, min	1	3.5	1.25	1
Cleaning out hole, min	1	3	1.25	1
Putting in new bit, cranking, min	1	2.5	1	1.5
Drilling second 2 ft, min	13	10.5	1	1.5
Drilling last 2 ft, min	12	10.5	6	11
Moving machine, setting up, min	15	35	12	11
Ft drilled per shift	.....	.....	96	48

Note. — Lm = limestone; S = sandstone (hard); Sd = sandstone (soft); granite; Tr = trap (diabase).

Rate of drilling. Formula for estimating number of ft drilled per shift (20): $\div\left(r+\frac{m}{f}+\frac{s}{D}\right)$; where, N = ft drilled per shift; S = working time per shift, min per 10-hr shift, if no time is lost by blasts, breakdowns, etc; r = actual time 1 ft, min; m = time to crank up, change drills, clean out hole, and crank down = min ordinarily; f = length of feed = 2 ft in ordinary percussion drills; s = t shift machine and set it up = 5 to 60 min, usually 12 to 20 min; D = depth of h

Records of drilling with 3¼-in machines, at 70 lb air or steam press, st bit about 2.75 in, finishing bit 1.5 in, gave following speeds for 1 ft of hole soft sandstone and limestone, 3 min; medium sandstone and limestone, 4 hard granite and sandstone, 5 min; very hard trap and granite, 6–8 min rocks that sludge rapidly, 8–10 min. (For other drilling records, see " pressed Air Plant," Peele, pp 334–342 and 372–376.)

Cost of drilling. Wages of drill crew in open-cut work range from $2 for driller and $1.50 to $4.50 for helper, averaging about $3 and $2 respecti Wages of power-plant crew depend mainly on size and type of plant and numb drills supplied. Drill repairs cost from 25¢ to 75¢ per working day (20) (S

Table 5. Cost of Operating One Drill for 10 Hr (Original)

Single drill from portable boiler		Fifteen-drill air or steam plant	
1 runner	$3.00	1 drill runner	$3.00
helper	2.00	1 " helper	2.00
[illegible]	2.50	1/15 fireman @ $3	0.20
[illegible] @ $2.75 per ton	0.90	300 lb coal @ $3 per ton	0.45
[illegible]ed	0.75	Water, nominal	....
[illegible]ening bits @ 4¢	1.20	Sharpening bits @ 3¢*	0.90
[illegible] drill and hose	0.75	Repairs to drill and hose	0.75
	$11.10		$7.30

[illegible]est, depreciation and general expense are not included. * Machine sharpening.

Hammer vs reciprocating drills (Sec 15). Hammer drills will not stand [illegible]d usage as reciprocating drills; their parts are lighter; the piston and [illegible]cks are subject to crystallization due to the rapidity of the blows; the [illegible] steel often used is high-priced and is more difficult to sharpen. As the [illegible]t of feed of hammer drills is generally shorter than that of reciprocating [illegible], the steels must be changed oftener; but time required for changing is [illegible]. Sharpening hollow steel costs somewhat more. The short feed in- [illegible] danger of the piston cracking the cylinder head, especially when han- [illegible]g an inexperienced man. Hammer drills, while faster in horiz or "dry" [illegible] are slower than reciprocating drills in "down" holes, so common in [illegible]ct work. But, for block-holing, plug-and-feather holes, or holes in trenches or other restricted places, the hammer drill is light and easy to [illegible] and many models require no mounting; it is therefore a valuable ad- [illegible] the heavier machines. It is especially useful for stoping in mines.

Hammer drills. Speed of cutting. A Sullivan, Class D–19 drill, 1.25-in [illegible]er. hollow steel, and air at 100 lb, drilled in granite 1.25-in holes, 1 ft [illegible] an aver of 1.75 min, using 25 cu ft free air per min. A Class D–15 [illegible] same granite, drilled ⅝-in holes at rate of 1 4½-in hole in 10 sec, and a [illegible] hole in 15 sec. In trench work in oölitic limestone, 12 DB–15 drills [illegible]d 90 1.5-ft holes per drill per 10-hr shift for 12 mo work. Best record [illegible] 1.5-ft holes in 10 hr and 36 3.5-ft holes in 7 hr. In dark green granite, [illegible] drill in 16 hr made 47 ft in 25 holes, from 19 to 36 in deep; a DB–19 [illegible]de 19 ft in 5 holes, from 32 to 60 in deep. A DC–19 drill, in soft sand- [illegible] made 20 holes 18 in deep, at rate of 25 sec per hole.

Electric drills. Advantages, as compared with air drills: low first cost of [illegible] ease of installation; economy of power. Disadvantages: very high [illegible]d maintenance; loss of time due to breakdowns. Electric drills have been successful and are little used (See Sec 16) (10, 20).

Air-electric drills (Sec 15) are driven from a self-contained compressor oper- [illegible] an electric motor.

Performance of electric drills (11, 20). 8 Marvin solenoid drills, oper- [illegible]ht and day for 2 yr in hard, seamy limestone, made 25 ft in 10 hr, as [illegible]ed with 45 ft by air drills. They had small pulling power after striking [illegible] and the steel stuck repeatedly. 2 Adams electric drills were tested in [illegible] diabase during 53 consecutive 10-hr shifts, with 2 2.75-in air drills. The [illegible]eraged 4 ft per hr during 317 drilling hr, with no stoppage for repairs; [illegible]tric drills averaged 2.55 ft per hr during 100 drilling hr, with 17 hr lost [illegible]rs. 4 Gardner No 15 drills, used for 2 yr in firm augite-diorite and [illegible]rite, averaged 13 holes, 8 ft deep, per drill per 10 hr. Cost of drill and sharpening (not including overhead expense) was 10¢ per ft of hole. From

2 to 7 Siemans-Halske drills were successfully used in German iron mi
fissured, sticky ground, in U S, this drill has been a failure. Dulles
drill, tested in trap and granitoid gneiss on the Catskill tunnels, made
of 5.3 ft per hr per drill for 5 weeks. Fort Wayne drill, tested in t
place proved strong, but lacked power to cut hard rock. 3 Box drills
Mexico for mining in metamorphosed lime shale, pyroxene, and quartz,
4-ft holes in 24 to 28 hr per drill. Cost of repairs was very high, b
drilling with inefficient labor cost 37 to 50¢ (Mex) per ft and was more ex

Cable, well, or churn drills (Sec 9) are extensively used for deep h
blasting in the softer rocks. Advantages, as compared with machin
Any depth can be drilled, to the possible limit of blasting; no strippin
overlying earth is necessary; holes in high faces are drilled to full depth,
of working in benches; the large-diam holes hold larger charges, hence wid
ing of fewer holes, and consequent saving of time; smaller consumption of
power. To offset these points, machine drills possess the advantages of p
ity and speed of drilling, which make them generally preferable (Sec 15

Size of churn-drill holes. In sandstone, 3-in bits are common,
6-in holes cost little more, they are economical where large, concentrated
are required. In a 16-ft quarry breast, with a 6-ft bottom stratum a
upper strata, 4-in holes were spaced 9 ft apart and 10 ft back. Thoug
was charged with 33 lb of 33% dynamite to within 20 in of the top, and
tamped with clay, the rock broke poorly. 5¾-in holes, spaced 12 by 12
charged with 42 lb of dynamite each, permitted 39 in of tamping, and
well. With 4-in holes, 1 ft of hole broke 8 tons of rock (1 lb dynamite
tons); with 5¾-in holes, 1 ft broke 12 tons of rock (1 lb dynamite to 4.5?

Table 6. Speed of Blast-hole Drilling with Churn or Cable Drills (Or

Kind of material	Ft per hr	Diam hole, in	Depth, ft	Machine	Remar
Clay, soapstone	10	5⅜	18	.....	Railroad w
Limestone	9	5½	30	Cyclone	Cement qu
Limestone	8	5	34	Cyclone	Lime, crus rock quar
Shale	6	5⅜	20	Cable drill	.....
Overburden, porphyry ore	5.7	.....	55	Keystone	Open pit mi
Limestone	5.5	5⅜	.....	Keystone	Crushed sto
Hard, seamy basaltic rock	5.2	3	23	Cyclone	.....
Soil, gravel	5 2	2½	23	Cyclone	.....
Brown sandstone	5 0	3	24	Cyclone	.....
Brown sandstone	5 0	5½	50	.....	Crushed sto
Shale	4 8	5⅜	75	Star	.....
Hard, seamy limestone	4 ?	5	22–30	Loomis	.....
Half earth, half slate	4 4	.....	12	Keystone	Aqueduct
Limestone	4 0	.....	50	Keystone	Lime quarry
Limestone	3 0	5⅜	40–45	.....	Quarry
Copper ore, porphyry	3 5	6½	60	Keystone	Mining
Shale	2 8	4½–6	30–80	Star	.....
Shale	1 8	5⅜	15	.....	.....
Copper ore	1 5 2 5	6	80–85	Star	Mining
Limestone	1 5	5	45–85	Star	Cement quar
Limestone, sandstone	1 0	5⅜	20	Cable drill	.....
Hard limestone	0 5 5	5⅜	15	.....	.....

Cost of operating churn drills (see Sec 9 for tabulated data). For st
power about 10 to 12 hp of water and 500 to 650 lb of coal per day are requi

...soline power (gasoline @ 12¢ per gal), cost is from 70¢ to $1.20 per day; ...ectric power at 2¢ per kw-hr, cost is about $1.25 per day. A compressed-...ated churn drill uses about as much air as a 3.25-in machine drill. About ...sh water are required per ft of hole. If necessary, it can be collected ...mp and used repeatedly. SHARPENING BITS costs much less than for ...e drilling: one bit, drilling 10 to 50 ft of hole, requires 1 hr to dress.

...pensive cost of churn and machine drilling. A cyclone drill, making 3-in ... ft deep, in solid brown sandstone in Ohio, put down 692 ft in 14 days of 10 hr, ... per day (20). A 3.25-in machine drill, making 3.75-in holes, 20 ft deep, put down ...e in 5 days, or 70 ft per day. Daily field costs (exclusive of sharpening) of the above ...ere: Churn drill. Runner, $3; helper (and fireman), $2; water, 60¢; coal @ 10¢ ...uel, 60¢; total, $6.20; cost per ft, 12.5¢. Machine drill. Runner, $3; helper, ...reman, $2; water, 75¢; coal, $1; total, $8.25; cost per ft, 11.8¢. The larger ...e churn-drill holes saved dynamite, as each hole was sprung but 3 times, whereas ...chine-drill holes had to be sprung 4 or 5 times.

...osts of churn drilling for blasting (20) (see Sec 9, Art 8).

5. Theory of Blasting

...ctors influencing results of blasting: size and number of free faces; cohesive ... of the rock; structure of rock (massive, jointed, laminated, stratified, or fissured); ... and nature of the explosive; character of fuse and tamping; whether the shot ... or simultaneously with others; whether the broken rock falls or must be ... the blast; form and size of chamber containing the explosive; proportion of ... of line of least resistance to length of the hole, and to height of free face (3, 10, 20).

...les for blasting. Many have been formulated, but all neglect some of the ... stated above. Also, most published rules are applicable to black powder ... are valueless for high explosives; and practically all ignore the use to which ... rock is to be put. Hence, experience and judgment are more useful in de... proper methods than the theories and rules summarized below. Theory of ... is discussed in detail in (5). According to the CRATER THEORY, a charge in a ... of earth or rock with horiz surface will blow out a funnel-shaped crater, the sides of ... have a slope of 1 to 1 to the FREE FACE (Fig 5). Distance *DB* (more exactly, *DF*) ... LINE OF LEAST RESISTANCE = l; hence, volume of crater ... $l \times w^2 = l^3$ (nearly). Hence, general formula for ... of rock loosened is $V = ml^3$. According to Schoen (5), ... for tough, soft rock, and 0.9 for hard, brittle rock.

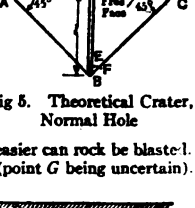


Fig 5. Theoretical Crater, Normal Hole

...rection of hole. If vertical (normal, as in Fig 5), the ... may blow out tamping and fail to break; hence, hole ... be inclined (Fig 6), to reduce chances of a blow-out, as ... to increase area of free face and volume of rock broken. ... inclination of drill hole is 45°.

...ect of free faces. The greater the area of free face, the easier can rock be blasted. ...ows area of volume broken when there are 2 free faces (point *G* being uncertain).

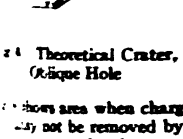


Fig 6. Theoretical Crater, Oblique Hole

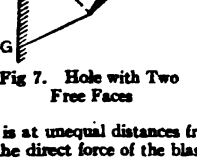


Fig 7. Hole with Two Free Faces

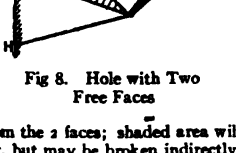


Fig 8. Hole with Two Free Faces

... shown area when charge is at unequal distances from the 2 faces; shaded area will ... not be removed by the direct force of the blast, but may be broken indirectly. ... 3 or more free faces are exposed, the longest line of resistance should not exceed 1.5 *l*. ... the aid of gravity, *l* should be horiz, and the longest line of resistance vertical.

Relation of factors. l should be proportioned to size and diam of the hole. eral, in open-cut work the depth of holes should be 1.5 l.

Table 7. Relations of Diam & Depth of Hole, & Length of Line of Least Re

Diam of hole, in	Depth of hole, ft	l, ft	Diam of hole, in	Depth of hole, ft	l
1.25	3.50	3.50	1.5	12.00	6
1.25	5.75	3.75	1.75	5.00	5
1.25	10.00	5.00	1.75	9.00	6
1.5	4.00	4 00	1.75	14.00	7
1.5	7.50	5.00			

Holes blasted simultaneously. Fig 9 shows the effect. If a and b are blas arately, c would not break out; when blasted together c is broken, if x is not to In ever hard rock, $x = 1.5\,l$ to $2\,l$; in weak rock, be about equal to l.

Fig 9. Effect of Holes Fired Simultaneously

Rock coefficient. To obtain it, select a homo bench 2 ft wide by 3 ft high. In this drill seve tical holes, so spaced that the blasting of one crack nor start the rock around another. Char with different weighed amounts of the explosiv used, beginning with a small quantity. Fire the holes separately. If C = roc P = wt of powder, lb, l = line of least resistance, ft; then $C = P \div P$, and P For 3 free faces, use 0.66 P; for 4 faces, 0.5 P; for 5, 0.4 P; for 6, 0.25 P.

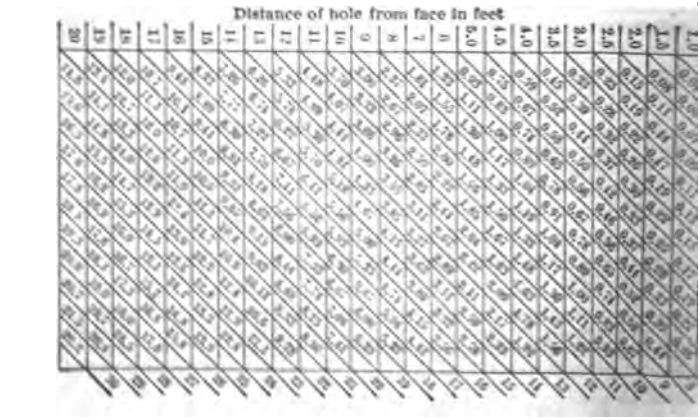
Size of drill hole for charge. If P = wt of explosive, lb; g = sp gr of ex and d = diam of hole, ft; then, $P = 0.34\,gd^3$.

6. Spacing and Pointing Drill Holes

Spacing holes in open-cut work. A common rule is to place a r vertical holes back from the face a distance equal to from 0.75 to 1 time depth, the holes being spaced the same distance apart. This usually hold in stratified rock, with holes of medium depth, but in dense granitic roc spacing must be closer. In shallow excavation the holes should seldom be f apart than 1 to 1.5 times their depth; in deep cuts the spacing may be much Hence, cost of drilling per cu yd is greater for low faces than for high.

Table 8. Spacing of Drill Holes in Open-cut Work

Distance of hole from face in feet



Depth of lift. In deep, open cuts or pits, rock is usually excavated in 2 or more ...s or lifts. Depth for economical drilling, size into which the rock breaks on ...ng, and presence or absence of seams or of horiz drill holes (called toe holes) ...ight assist breaking, all determine economic height of lift. 3½ and 3½-in ... drills are good to depths of 16 to 24 ft. Churn drills are efficient for al- ... depth, and where they are used lifts of 60 ft are common. Max depth ... drilling is about 8 ft with hand hammer, and for machine drills, 18 ft. ...le the higher the bench, the farther back from the face may the holes be ...d, but the farther back the holes, the coarser will the rock break. In deep ... explosive should be separated into several charges with tamping between; ... the entire charge is at the bottom of the hole, the bottom of the bench may ... cut and the top left overhanging. If 2 rows of horiz holes are drilled ... face, besides the vertical holes, the height of the bench may be increased.

...e 9. Spacing of Holes, Charges, and Results of Machine-drill Blasts (Original)

Rock	Kind of work	Aver depth of hole, ft	Aver dist of rows from face, ft	Aver dist apart of holes, ft	Ft of hole per cu yd	Grade of explosive, % Ngl	Kind of explosive	Explosive per cu yd of rock, lb	Diam of hole, in
... ...ze ..	Canal	12	8	8	0.40	40	*A*	0.75	.....
... ...ne	Crushed stone	6	5	6	1.00	40	*A*	0.70	.....
... ...ne	Cement.....	20	.....	.....	....	50	*A*	0.37	.....
... ...nite	R R thro'-cut (*a*)	20	7	7	0.42	60	*A* (*b*)	1.05	4.5
... ...ne ..	.....	15	.....	.....	0.43	50	*A*	0.26 (*c*)	.....
... ...stone	Canal.....	14	4	6	....	40 60	*A* *A*	0.38 0.38	....
... ...stone	Canal (*d*).....	13	8	6	0.56	40 60	*A* *A*	0.38 0.44	..... 4.5
... ...stone	Crushed stone	26 (*e*) 12 (*f*)	9	6.5 6	0.47	60 60	*A* *A*	1.35	5.5 4.5
... ...ne ..	R R side-cut..	20	12–18	12–18	0.10	40	*B* *A*	1.00 0.10	....
... ...stone .	R R thro'-cut.	20	12–18	12–18	0.20	40	*B* *A*	2.00 0.20	....
... ...one. ..	R R cut.....	20	18	14	0.15	....	*B*	0.15 (*g*)	.....
... ...e ..	R R side-cut..	24	12–18	12–18	0.08	40	*B* *A* (*c*)	0.70 0.03	....
... ...ble .	R R thro'-cut.	24	12–18	12–18	0.20	40	*B* *A* (*c*)	1.50 0.10	....
... ...	Rubble.....	16	5	5	1.36	60	*A*	0.20	.....
... ...nite	Crushed, rubble	(*h*)	.....	5	....	75 60	*A* *A*	.....	2.5
... ...	....	12	.....	.....	1.33	40	*A*	0.60	.....
... ...	....	14	.....	.....	0.63	40	*A*	0.50	.....
... ...	Mine.....	12	2.5	6 (*i*)	1.70	40	*A*	0.67	.....
... ...	Mine.....	12.5	.....	.....	0.32	52	*A*	0.44	.....
... ...trap .	Crushed.....	14	.....	.....	0.35	75	*A*	0.20	.....
... ...trap	.....	16	.....	.....	1.00	40	*A*	0.70	.....
... ...slate	R R thro'-cut.	12 (*j*)	10	10	0.27	60	*A*	1.11	4.5
... ...rock	Dam filling...	18	.....	.....	0.13	....	*B*	1.85	.....

... ...namite. *B*. Black powder. (*a*) 35 holes. (*b*) Holes sprung with 2 lb dynamite. ... ...s sprung. (*d*) 45 holes. (*e*) 60 holes; top holes, vertical, 26 ft deep. (*f*) 75 holes; ... ...les, one at 15° and one at 60° with vertical, 10 to 14 ft deep, the former being 6 ft away ... 5 ft in front of latter. (*g*) Sprung 3 times. (*h*) 1st row 6 to 15 ft from face; 2nd row, ... from first row; about 2.5 lb of 75% dynamite & 6.25 lb of 60% per hole. (*i*) Holes ...ical. (*j*) 30 holes; holes at angle of 15° with vertical. Sprung with 3 lb of dynamite.

Table 10. Spacing of Holes, Charges, and Results of Churn-Drill Blasts (C

Kind of rock	Character of work	No of holes	Diam of hole, in	Aver depth, ft	Aver dist holes from face, ft	Aver dist apart of holes, ft	Total explosive used, lb	Grade of explosive, %	Kind of explosive	Rock blasted, cu yd
E(*a*)	Crushed stone	4	5⅛	66	.....	.....	5 500	.....	*A*	20 000
E	Crushed stone	8	5	50	12	12	1 200	40	*A*	.....
E	Cement q'rry	.....	5	36	10	14	60(*b*)	40	*A*	.....
E	Cement q'rry	8	6	65	20	20	4 000	40	*A*	.....
E	Open-pit iron mine	...	6	20	15	15	65(*b*)	40	*A*	.....
E	R R ballast.	8	6	48	19	18	3 300	40	*A*	5 720
E	Cement q'rry	9	6	62	32	20	1 800 2 500	40 60	*A*	12 320
E	Cement q'rry	7	6	60	23	15	500 2 900	60 40	*A*	6 610
E	Hard R R ballast	8	6	95	33	28	2 200 3 350 1 250	..... 60 40	*B* *A* *A*	27 300
E	Lime quarry	3	6	100	24	17	1 200 600	60 40	*A*	5 720
E	Cement q'rry	9	6	52.5	36	20	1 720 2 500	60 40	*A*	13 660
F(*d*)	Crushed sand	4	4	40	13	13	275	.....	*A*	1 387
G	Copper mine.	1	.....	80	.....	.....	4 000(*e*)	40	*A*	12 000
H	R R thro'-cut	578	4	25–40	(*f*)	(*f*)	1 200 27 275	60 35	*C*	35 000
I *J*	Open-pit mine	...	5⅛ 5⅛	60 60	40 15	35 15	100(*g*) 625	40	*A* *D*	3 100 500
K	Placer dredge	...	...	30	20	50	30(*i*)	80	*A*	.....
	R R thro'-cut	8	4 5	32–37	(*h*)	22	50 300	2*F*–3*F*	*D*	16 113
I	Copper mine	12		65–125	....	100	58 050	.....	(*j*)	200 000

A Dynamite *B* Gelatin. *C*. Gelatin dynamite. *D*. Black blasting powd Limestone. *F* Sandstone. *G*. Porphyry. *H*. Basalt. *I*. Copper porphyry. *J*. carbonate *K* Gravel *L*. Ore and capping. (*a*) Cost per day: Runner, $2.50; $e, electric power, $2; oil, sharpening, etc, $1.50; total, $8. Aver ft per 10 (*b*) Per hole (*c*) Cost: drilling and charging, 4 5¢ per cu yd (drilling by c at 10¢ per ft (*d*) Crushing, 12¢ per cu yd, total, 16 5¢ per cu yd. Holes sprun dynamite (*e*) Sprung with 150 lb of 40% dynamite. Cost: drilling @ 58¢ $30 4 150 lb dynamite @ 10¢, $420; fuse and caps, $2; labor charging a tobacco $10, total $460 or 4 1¢ per cu yd (*f*) Holes in 5 parallel lines: 1 line 5 ft from free face, 2 lines 24 ft from center line. Holes staggered, 14 ft apa charged with 60% dynamite. Cost, 11¢ per ft. Loading required 8 days. (*g*) Cost of drilling 4 in holes, 10¢ per cu yd; for 40 by 35-ft spacing. (*h*) Cost, 9¢ per ft. Holes sprung with Cost 10¢ per ft. Charging, after springing, co $1 Per hole cost of blasting about 5¢ per c Material sold at about $2.000

Charging and Firing

Black powder and electric firing

Deep Hole Blasting place the dyna charge having its

If the rock consists of hard and soft layers, the charges should be ... in the hard layers. Contractor's powder and Judson powder (R R P ... are charged like black powder, but are exploded by a dynamite primer. ... grades of Judson and all dynamites are exploded with detonators. If ... powder and dynamite are charged in the same hole, explosion of the ... will detonate the dynamite.

...amite and black powder in one blast. In blue sandstone large charges of black ... and dynamite in alternate rows of holes were found effective (20). In Fig 10, ... holes were made with churn ... in bit. Holes marked "kegs" ... with 25-lb kegs of black pow... those marked "boxes" were ... within 4 ft of the top with 40% ... as shown. Before loading ... black powder, each one was ... below; first with 15 sticks of ... dynamite, second with ... with 80 sticks, and a final ... 30 sticks of 40% dynamite per ... dynamite and powder were fired together, on the theory that the powder ... the rock and the dynamite would shatter it. About 2 700 cu yd of rock were ... one blast, with 800 lb of dynamite for springing and 6 000 lb of black powder ... of dynamite for the final charges.

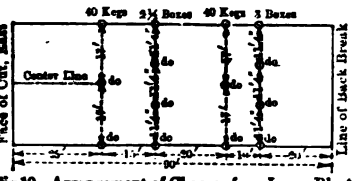


Fig 10. Arrangement of Charges for a Large Blast

...ging holes (see Sec 4, Art 8, 9). In open-cut work in stratified rock, ... black or contractor's powder, it may be advantageous, for widely-... holes, to put the entire charge in the bottom. Sprung holes are then ... In springing 20-ft holes in sandstone the first charge was 2 sticks ... dynamite, the second, 5 sticks and the third, 20 sticks, the last making ... for holding 8 kegs (200 lb) of black powder. But, in trench exca... or in deep churn-drill holes in high breasts, the charges should be dis... each having its own primer. Heavy springing is not advisable in ... shales, as it may fill the chamber with debris. In highly-inclined ... the shock of springing may cause a slip and close up the hole.

...amber blasting is done by driving a small tunnel or sinking a shaft, at the ... h chambers are excavated for the main charges of explosive. For black ... the chamber may be below the floor of the drifts, for convenience in pour... powder into large wooden boxes, built in the chambers. The tramping of the men packs it tightly, and the solid sides of the excavation offer greater resistance to the explosion. COYOTE HOLES are one-man tunnels or drifts, about 2.5 by 3 ft in size, which are charged with large quantities of explosives.

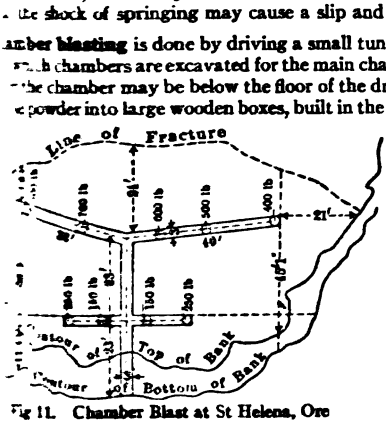


Fig 11. Chamber Blast at St Helens, Ore

Gophering is a mode of blasting used in breaking the overburden in the Mesabi, Minn, mining districts, and elsewhere, in sandy, loose ground, where vertical holes ... be kept open (20). (See *Eng & Min Jour*, Vol 88, p 696.) A hole is ... with a pointed bar, at a down angle of 15° to 20° in the side of the bank. ... cartridges, placed end to end, are pushed into the hole and exploded.

The muck is removed with a long-handle shovel, the hole deepened furt the process repeated until a hole 10 or 12 in diam and deep enough is o A chamber is made at the end by springing with 2 or 3 cartridges. A lon box filled with powder is pushed in and overturned.

Chamber blast, St Helena, Ore (Fig 11). The rock was basaltic, weighing 1 cu ft. Explosive was No 2, MV Trojan powder; charge, 3 500 lb. The tun tamped to the portal with muck. The rock was sufficiently broken to be ha steam shovel, but little bulldozing (sledging) being necessary. Cost of explosiv charging, $58; total cost of blast, exclusive of tunnel driving, $417 or 1.24¢ Volume broken, 14 280 cu yd.

Table 11. Charges Used in Chamber and Coyote Blasts (Origin

Case	Rock	Rock loosened, cu yd	Dynamite, lb	Black powder, lb	Judson powder, lb	Lb per cu yd		
						Dynamite	Black powder	Ju po
I	Granite	110 000	144	35 000	.....	0.001	0.320	..
II	Porphyry	50 000	100	.....	1 200	0.002	.....	0 0.
III	Granite	80 000	5 400	29 050(a)	.....	0.067	0.374 (a)	..
IV	Limestone	.....	....	12 000	.....	....	.....	...
V	Sandstone	220 000	1 250	43 100	.....	0.006	0.195	...
VI	Rock	350 000	11 400	.....	114 000(b)	0.033	.....	0.3
VII	Basalt	14 280	....	.....	3 500	....	.....	0.2
VIII	Black trachyte	250 000	3 000	.....	30 000	0.012	.....	0.1
IX	Limestone	1 700	26	2 775	.....	0.015	1.632	...
X	Trap	150 000	7 000	.....	.....	0.047	.....	...
XI	Granite	440 000	32	18 250	.....	0.0001	0.042	...
XII	Gravel	.....	....	17 500	.....	....	.....	...
XIII	Gravel	150 000	....	50 000	.....	....	0.333	...
XIV	Cemented gravel	500 000	....	.....	36 400	....	.....	0.07
XV	Gravel	200 000	3 500	.....	.....	0.018	.....	...
XVI	Rock	10 000	500	7 500	.....	0.050	0.750	...
XVII	Basalt	40 000	....	35 000	.....	....	0.875	...
XVIII	Basalt	90 000	....	34 625	.....	....	0.385	...
XIX	Basalt	40 600	....	.....	10 000	....	.....	0.24
XX	Basalt	110 000	....	.....	21 000	....	.....	0.19
XXI	Basalt	56 000	....	.....	10 000	....	.....	0.17
XXII	Hard basalt	400 000	20 000	431 000	.....	0.050	1.078	...
XXIII	Basalt	54 800	....	.....	8 000(b)	....	.....	0.14
XXIV	Cemented gravel	180 000	....	30 025	.....	....	0.167	..

Notes on Table 11 (20). I. West Beaver Creek dam, Col. Tunnel 75 ft bel of rock, 135 ft long, with several bends. Cross drifts, 35 ft long, each way fron tunnel. Charges at ends of cross drifts, with 3 000 lb of powder along outer wa mainder of cross drift. Tamping: rock, earth, timber. II. Otay, Cal. Tunn 5.5 ft, 50 ft long. 18-ft Y-branches at end for chambers. Charges: 4 000 lb powder and 50 lb dynamite in one chamber; 8 000 lb powder and 50 lb dynamite i Cost: drifting, $645; powder, $960; charging, $75; total, 3.6¢ per cu yd. Furthe ing by powder in seams made total cost 5¢ per cu yd. III. San Diego, Cal. More Open cut perpendicular to face, with 4 by 5-ft drift, 115 ft long, parallel to anc from cut. Chambers sunk beneath floor at end and 70 ft from face. Face c contained 500 lb 7% Champion powder and 1 500 lb 40% dynamite; end cl 28 550 lb 7 and 9% powder, 1 900 lb 40% and 2 000 lb 60% dynamite. Ta earth, timber. Cost: open-cut, $3 500; drifting and charging, $2 478; exp $3 116; total, 5.05¢ per ton. IV. Northampton, Pa. Quarry. Face, 135 f Drift, 3 ft wide and 238 ft long, along a fault 50 to 100 ft from face. 4 chamber tunnel, 45 ft apart, and 3 crosscuts each way, 25 to 56 ft long. Total cost, $3 8

...Wash. Quarry. 65-ft face. Two 3.5 by 4-ft drifts, 200 ft apart; one 150 ft ... with 3 crosscuts 50 ft apart, each 80 to 100 ft long; the other 180 ft long, with 4 ... each 70 to 100 ft long. 60% dynamite. Tamping: muck, timber, and cement ... VI. Piedra, Cal. Quarry. Aver height of face, 91 ft; aver overburden, ... drifts, each 80 ft long, with 2 crosscuts each side. Crosscuts 40 ft apart, 40 ft ... Pits at ends of cuts. 60% dynamite and Judson R R P. Cost of explosives, ... per cu yd. VII. St Helena, Ore. Quarry. Drift, 3 ft wide by 46 ft long, with ... at end, one 32 ft, the other 40 ft long. Halfway from face, a crosscut in each ... ft long. No 2 Trojan powder in 4 charges of 150 to 250 lb in short cut and ... of 400 to 700 lb in long cut. Cost: explosives, $359; loading, $58. VIII. ... Cal. Quarry. Overburden, 80 ft. Drift, 110 ft long, with side drift 60 ... face, 15 ft long to left and 40 ft to right; diagonal drift 80 ft from face, 40 ft to left, ... the end a diagonal drift 50 ft to left and a straight drift 50 ft long to right. End ... charged with Judson R R P and 60% dynamite. IX. U P R R. 18-ft cut. ... charged with 26 lb dynamite and 2 775 lb of powder. Cost, about $1.10 per cu yd. ... Judson River. 200-ft face. 1 drift at bottom, 65 ft deep; other drift, 60 ft from top ... 50 ft deep. Two 25-ft shafts at top; also drill holes. XI. Long Cove, Me. ... 4 by 4 ft, 64 ft deep, with 2 drifts at bottom, each 27 ft long. Crosscuts from ... drifts, 26 ft long. Explosives in crosscuts. Estimate of 1 000 000 tons broken ... 100 high. XII. Paragon hydraulic mine. Face, 150 ft high. Drift, 110 ft long. ... at right, 70 ft long, with drift at end, parallel to main drift, 55 ft long. Cross- ... 50 ft long, with drift at end, 30 ft long. Much space left untamped for expan- ... chosen. Cost: drifting, $300; explosives, $2 700. XIII. Blue Point hydraulic ... First, 3 by 4 ft, 275 ft long. 6 crosscuts, each 120 ft long on left. 6 on right, ... long. First drift on right, 75 ft from portal, and at end a 15-ft drift, parallel ... drift. XIV. Dardanelles mine. Face, 175 ft high, 1 200 ft long. 5 parallel ... tunnels each of which were 2 or more crosscuts. Total length of drifts, 1 200 ft. ... hydraulic mine. Giant powder No 2. XVI. Colorado. Dam. Coyote or one- ... tunnel, 40 ft long. 2 crosscuts from end, each 12 ft long, with pits at end. Ex- ... FFB powder and 40% dynamite, charged in pits. Tamping: earth. Cost: ... dynamite, $155; powder, $1 140; caps and fuse, $11. Total, 16.9¢ per cu yd. ... Oregon. R R. Coyote hole, 2.5 by 3 ft in hillside, 50 ft deep. Crosscuts at ... and 15 ft. Charges in the 3 openings. XVIII. Crooks Landing. R R. 4 or ... holes, 80 ft long, with Ts 40 to 60 ft long at ends. XIX. Oregon. R R. ... breast. XX and XXI. Oregon. R R. XXII. Snake River, Wash. R R. ... holes, 2.5 by 3 ft, each averaging 89 ft long, run into and then parallel to ... face. 3 500 ft of cliff mined by 6 177 ft of coyote holes. 20 000 lb of dyna- ... used in preparing for main blast of F to 5 F black powder. XXIII. Oregon. Coy- ... No 2 Trojan powder. XXIV. Smartsville, Cal. Hydraulic mine. Shaft, ... with main drift 185 ft long from bottom. 3 crosscuts, 70, 120, and 170 ft from ... long on other side. 10 lifter drifts from crosscuts, each 15 ft long, parallel to ... Total drifts, 570 ft long by 2.5 ft wide by 3.5 ft high. Material moved, 270 ... 100 ft.

Boulder blasting is done by: mudcapping or bulldozing; blockholing and ... or snakeboling. Other methods are: by sledging; by drop- ... or drop weights; by ... with fire and then ... by applying cold ... by a combination of 2 ... of these methods. ... methods: (*a*) sledg- ... drop-hammering; ... ing. Cannot be used ... ers larger than 0.5 ... yd, and it is neces- ... resort to explosives

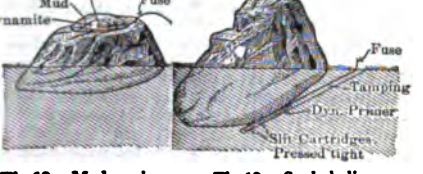


Fig 12. Mudcapping

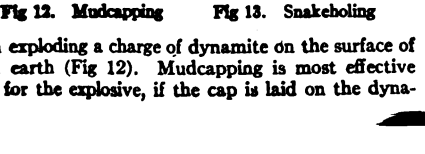


Fig 13. Snakeholing

Mudcapping consists in exploding a charge of dynamite on the surface of ... after covering it with earth (Fig 12). Mudcapping is most effective ... a depression is selected for the explosive, if the cap is laid on the dyna-

mite and not shoved into it, and if wet clay is used as a covering. Sna consists in boring a hole beneath a boulder and firing a charge in it (It is more efficient, but not so rapid as mudcapping. Blockholing co drilling a shallow hole in the boulder for a small charge of dynamite.

Average cost per cu yd of breaking boulders, from a number of recc is: sledging, 4.3¢; drop-hammer, 6.5¢; heating, 14.9¢; blockholing, 16 dermining, 17.5¢; mudcapping, 31¢; mudcapping and sledging, 32.1¢.

Table 12. Charges for Boulder Blasting (du Pont)

Weight of boulder, lb	Approx No of 1.25 by 8-in cartridges (40-60% dynamite)			Weight of boulder, lb	Approx No of 1.25 b cartridges (40-60% dy	
	Mud-capping	Snake-holing	Block-holing		Mud-capping	Snake-holing
500	1.5	1	0.25	4 000	4	3.5
1 000	2	1.5	0.5	5 000	4.5	4
2 000	3	2.5	0.67	7 500	6	5
3 000	3.5	3	1	10 000	8	6

8. Cost of Loading and Hauling

Hand work. One man will load 2 to 20 cu yd of rock (place mea 10 hr, depending mainly on size of pieces and height to be lifted.

On Chicago Drainage Canal the aver per man in 10 hr was about 7 cu yd loa dump cars. Sledging took about 14% of the time. Aver per man loading cableway skips, 10 cu yd; LARGE STONES were rolled into the skips, very little being required. In loading wagons with high sides, 1 man will average 10 cu measure (17 cu yd loose) of easily-lifted stones per 10 hr. Stones handled singl thrown off a wagon twice as fast. Stones can be loaded on wagons having sto at rate of about 13 cu yd per 10 hr, and rolled off at 50 cu yd per hr (20). C STONE can be shoveled from smooth boards or steel sheets at the rate of 13 cu measure (22 cu yd loose) in 10 hr; in shoveling from the ground or hopper-botton man will do only 7 to 8 cu yd solid measure (12 to 14 cu yd loose).

Steam shovel work. Cost in rock excavation varies greatly. In t iron ore of Mesabi range, under fair conditions, a steam shovel easily lo cu yd per hr; but, in poorly drilled and blasted rock, broken in large p shovel may do as little as 17 cu yd per hr.

Table 13. Output of Steam Shovels, Loading Blasted Rock (Origin
Compiled from one 10-hr day's work (12)

Material.....	Iron ore				Limestone			Slate, lime-stone	Por-phyry, granite	Por-phyry
Work.....	In bank		Stock pile		Quarry	R R cut		R R	R R	R R
Conditions..	Good	Fair	Fair	Good	Fair	Low face		Good	Hard	Hard
No of shovels	1	1	av of 5	1	av of 2	1	1	av of 2	av of 2	1
Size, tons...	70	95	70	90	95	65	70	70	70	65
Dipper, cu yd	2.5	2.5	2.5	2.5	2.4	.....	.....	2.5	2.5	2.5
Coal, tons...	1.75	3	2.35	4.5	3.5	2.8	2.3	2.8	2.2	2.2
Oil, gal.....	1.4	3.25	1.45	8.5	.....	.....	.....	.....	.....	.....
Water, gal...	3 000	5 000	3 500	7 700	.....	.....	.....	.....	.....	.....
Cu yd loaded	892	1 350	1 251	2 728	917	358	205	1 126	799	447

Wages of crew: Foreman (per mo), $75 to $175, aver $135; craneman (per m to $125, aver $96; 1 fireman (per mo), $50 to $87, aver $62; 1 coalman (per day) to $1.50, aver $1.47; 4 to 12 pitmen (aver 6), $1.40 to $3.50, aver $1.90.

...NY mica schist, broken large, a 65-ton, 2.25-cu yd dipper shovel averaged for sev... weeks about 280 cu yd solid measure per day into cars; part loaded by the dipper, ...ted by a chain hooked over the dipper teeth (20). On Chicago Drainage Canal, ...ers 55-ton shovels, with broad shallow 2.25-cu yd dippers, loaded limestone on ...ction (Table 13). The rock was in large pieces, much of which had to be lifted ...ram. Combined output of the 2 shovels was 118 650 cu yd (solid measure) dur... ...10-hr shifts, or an aver of 296 cu yd per shovel per shift (4, 20).

...ading with derricks, and bucket or skip. Scows may be unloaded rap... about 25¢ per cu yd loose measure.

...ine-operated derrick, with a crew of 1 foreman ($3), 1 hooker, 6 shovelers, 2 tagmen, ...pman (@ $1.50), water boy ($1) and allowing $3.50 per day for team and driver, ...et 120 cu yd loose measure in 1 day at 19¢ per cu yd. In using an engine-operated ..., with a bullwheel for slewing, tagmen (for slewing the boom) and team costing ...er day are eliminated, and an engineman and coal costing about $4 per day are ...d. A crew of 1 engineer ($2.75), 1 signal man ($2), 1 dumpman ($1.75), and 7 ... $12.25), unloaded from a scow 21.3 cu yd of ¾-in crushed stone per hr, at cost of ...e per yd, not including coal, oil, water, overhead expenses, or depreciation. Clam... ...kets are good for unloading cars and scows. A stiff-leg derrick with bullwheel ...about $400, ¾-yd clam-shell bucket about $400, and double-drum engine with ...ar gear about $1 000.

...bleways (Sec 26) are frequently used in quarry work, canal and trench ...tion, and in open-pit mining.

Table 14. Output of 4 Chicago Drainage Canal Cableways (1 month)

	1	2	3	4
...10-hr shifts	49	35	52	49
...u yd rock	12 633	8 632	16 162	14 535
...k-loads, day shift	5 111	5 327	5 435	4 369
...ck-loads, night shift	4 087	1 201	5 467	4 468
...loaded rock per skip	1.44	1.32	1.48	1.65
...loaded rock per shift	258	247	311	297
...laborers	27	27	32	32
...tormen	2	2	2	2
...s or, hr	12 861	9 608	17 075	15 227
...rock loaded per man per shift	9.82	8.98	9.46	9.54
...s per shift	1.83	1.83	2.28	2.28

...icago Drainage Canal (2, 20) employed 19 cableways with spans of 550 to 725 ft, trav... 73 to 93 ft high, and equipped with aerial dumps. Main cables, 2.25-in, hauling ...ing cables, 0.75-in, but... ...ping cables, ⅝-in. ...der and 10 by 12-in ...e a hoisting speed of ...; a traveling speed of ...er min. A complete ...2 by 7 by 7-ft skips, ...about 225 tons; cost ...,000. Crew: engine... ...d, fireman ($1.80), ...an ($2.70), rigger and ... $1.50) for loading. ...cableways, 300 to 450 ...l measure per 10 hr; ...30¢ per cu yd, incl ...apers, coal, oil, repairs.

Table 15. Cost of Cableway Work Shown in Table 14

	Distributed cost			At 50¢ per cu yd, cost per cu yd, ¢
	Labor (⅓)	Supplies (⅓)	Total	
Drilling	22	10	18	9.0
Explosives	3	58	21	10.5
Loading	46	2	31	15.5
Conveying	15	20	17	8.0
Channeling	4	3	4	2.0
Pumping	4	7	5	2.5
Sup't & gen'l labor	6	.....	4	2.0
Total	100	100	100	50.0

...wrock Dam, Boise irrigation project, Idaho. Two Lidgerwood cableways handled ...u yd of blasted rock, boulders, gravel, and sand. Span, 1 300 ft; aver traveling ...500 ft; aver hoisting distance, 300 ft. Hoisting load, 8 tons, at 300 ft per min;

conveying speed, 1 200 ft per min. Skips, 8 by 8 by 2 ft. In July, 1912, 2-shift w Aug, Sept, and Oct, 3 shifts. Output for 4 mo, 40 624 loads, averaging 2.49 cu yd, cally all handled in 2 shifts. Cost of operation (not including loading), 37¢ per cu follows: labor, 11.1¢; power, 4¢; supplies, 1.5¢; repairs, 4.7¢; deprec, 6.4¢; prop expense, 9.2¢. Wages: laborers, $2.40 per 8 hr; cableway operators, $4 to $5; rang to $4. Power cost, 1.5¢ per kw-hr. Deprec was figured on charging off 75% of fi and all the installation cost (20).

Canal work. On St Mary's Channel Improvement, 4 cableways handled a cu yd of limestone in 2.5 yr. Rock was loaded into skips by 4 60-ton traction-ty steam shovels. Skips held 6 cu yd each, but sometimes 8 cu yd (18 tons) were h 2 cableways had spans of 1 100 ft and 2 of 800 ft. Aver haul, 300 ft. Best 1 record for all, 22 000 cu yd each; best month's record for one cableway, 30 000 cu

Cost of an aver cableway of 800-ft span and for a 5-ton load, complete with 12-in engine, but without boiler or towers, is $6 000 to $7 000. A boiler or a motor-operated plant costs about $1 500. Towers cost erected in place, $500 e Traveling towers cost 3 to 5 times as much (19).

Stone-boats are wooden or sheet-iron platforms, best mounted on runners, fo ing large stones short distances. If the runners are greased, a ton can be pulled by weighing 2 400 lb. A SKID ROAD is formed of partly imbedded round sticks of set like ties on a track 3 to 6 ft apart. A stone-boat holding 0.5 cu yd (solid meas rock can be drawn over such a road. A LIFTER or DEVIL is a wooden stretcher on 2 men can carry as much as 0.5 cu yd. On the Grand Trunk Pac R R (20), ro cheaply hauled by stone-boats on pole tracks in summer for hauls less than 600 ft, winter any distance. Track was of 2 lines of 20 to 30-ft poles, 4 to 8 in diam apart for 2-horse team and 3 ft apart for 3-horse team. Poles were joined by 2-i wood pins. Boats were of 10 or 12 logs, 7 in diam by 8 ft long, fastened togeth 1.25-in rods. In winter the track was iced; in summer, greased (1 gal per 100 ft pe In winter, a team could haul a 3-yd rock; in summer, 1.5 yd; aver load, ¾ cu yd. 500-ft haul a team and 6 men took 40 to 60 loads per 10 hr. Aver yardage p loading, 7.3 cu yd; many rocks were large and had to be blockholed. The exc comprised: 20% shovel dirt, 30% easily lifted stones, and 50% pieces 1 to 5 volume. Cost of loading, 31¢ per cu yd; cost of transport, 17¢. Wages: m $2.00 to $2.25; foremen, $3.75; cost of each horse, 75¢ per day.

Wheelbarrows hold about 0.04 cu yd solid rock; loaded by 1 man in 2 mi wheeled at 180 to 250 ft per min, losing 0.75 min per round trip. COST, at 15¢ per loading, hauling, and dumping: to a fixed cost of 18¢ per cu yd solid rock, add 6. 100 ft lead (20).

Carts and wagons. In aver rock, 1 cu yd solid equals 1.75 cu yd broken, and about 2.2 tons. Over poor dirt roads, with occasional steep rises, 0.5 cu yd (1 ton rock may be hauled by 2 horses; on hard, level road, 1.5 cu yd; aver load on good 1 cu yd (2 ton). Aver speed of haul, 220 ft per min. A 1-horse cart, on short, hill hauls, takes an aver load of 0.25 cu yd solid rock; under favorable conditions yd aver. COST BY CARTS. For short hauls, 1 driver can run 2 carts. With 4 or loading, @ 15¢ per hr, and 2 carts, 2 horses, and 1 driver @ 35¢ per hr, the l cost is about 20¢, cost of lost time 8¢, and cost of dumping 2¢, making a fixed cost per cu yd. For total cost of loading, hauling, and dumping: to a fixed cost of 3 cu yd solid rock, add 0.12¢ per 100 ft lead.

Cost by wagons. 2 men and a driver can load 1 cu yd on a stone rack in 15 mi man and driver can unload it in 7 min, or a total lost time of 22 min. Aver of each 7.5 cu yd in 10 hr. Hence, at 15¢ per hr for laborers and 35¢ for team and driver the lost time of team and driver is 12¢, and for labor loading and unloading, 23¢; hence, fix is 35¢ per cu yd solid rock. For total cost of 1-cu yd loads, add 0.6¢ per 100 ft lea

Cars on track. For tractive power of horses and resistance of ordinary cars see Sec 3, Art 5. On level track, a team will haul 2 cars of 3 cu yd rock; on slight down grade, 1 horse will haul 2 cars holding 1 to 1.5 c On good track, at slight down grade, 1 horse can haul 4 light rocker cars holding 4 cu yd, if assisted by laborers in starting. COST. If ro broken into sizes that 1 or 2 men can lift, 6 to 7.5 cu yd can be loaded pe

hr. About 4 min are lost in changing teams from empty to loaded cars, ed the track arrangement is good. Speed should be 200 ft per min. per hr for laborers and 25¢ for 1 horse and driver, lost time at pit costs of loading 20 to 25¢, and the cost of dumping, 2 to 5¢ per cu yd, hence, fixed cost is say 25¢. For total cost with 1 horse, add 0.5¢ per per 100 ft lead; for 2 horses, add 0.33¢ per 100 ft lead per cu yd, solid.

Steam-shovel loaded. In loading rock by steam shovel, the output of the attendant and locomotive is limited chiefly by the shovel output, not by the speed at which may be handled. Following work was done in a quarry of hard crystalline lime- 2 Bucyrus 95-ton, 2.5-cu yd dipper, shovels (12). SHOVEL A. First day: time, 691.5 min; lost time, 59 min, of which 41 min were for blasting, clearing tightening jacks, 13 min waiting for cars to be spotted, and 5 min idling. Sec- 138 min were lost waiting for cars, of which 87 min were spent in drilling, ling, and preparing to move, and 51 min idling. On 2 days, 197 min, or the time, was lost waiting for cars; while the cars lost on account of shovel: no reason, 7 min; moving forward, 107 min; drilling, 125 min; blasting, 67 min; track, 9 min; total, 315 min, or 25%. SHOVEL B in 2 days worked 1 290 min. 266 min or 20.6% was spent waiting for cars, of which 87 min was in idling. days, 327 min (25.4%) was lost to the trains by the shovel, as follows: oiling, setting up steam, 42; repairs, 5; waiting for cars to be loaded, 52; blasting, ing forward, 113; coaling and miscellaneous, 25 min. There were 5 35-ton motives, 4 working and 1 being overhauled. They hauled 10-car trains. 5 cu yd, weighed 4 tons, and cost $150 each. While moving the engines aver- ft per min; minimum speed 156 ft, maximum, 1 000 ft per min.

8. Quarrying (see also Open-cut Excavation, Art 10)

Kinds of stone. DIMENSION STONE is quarried and split to assigned dimensions ready ing. RUBBLE STONE is in rough slabs or blocks of irregular sizes. For dimen- ne, there must be a good WORKING FACE and usually a CHANNEL at each end, to ves. Then, by wedging or careful blasting, long blocks are secured, which are short blocks for handling by derricks. To get a cushioning effect in blasting stone, several inches of hole above the charge may be filled with hay; called tamping." For rubble or backing stone, but little channeling is done; the taken up by light blasts and large irregular slabs barred and wedged out.

Rifts and cleavage planes must be carefully considered. All sedimentary rocks and ers, as granite, have 3 perpendicular cleavage planes called the GRAIN, RIFT, and Trap rocks, diabase, diorite, porphyry, etc, often have no rift and are unfit for dimen- e. Cost of quarrying depends partly on thickness of beds and their DIP (slope) rystal. With steep dips, both thick and thin-bedded stone must usually be simultaneously; as the quarry deepens, the depth comes too great for profitable work. If joints are a quarry is a BOULDER QUARRY; if vertical and at en, BLOCK QUARRY; where there is practically no nt, but a series of horiz joints, a SHEET QUARRY.

Fig 14. Plug and Feathers

Drilling and feathering consists of splitting rocks by holes, in which 2 FEATHERS or shims (pieces of d iron, the sides of which are curved to fit the forced apart by hammering a wedge plug be- them (Fig 14).

Holes required. A granite block 6 ft thick may be in a row of plug holes 5 in deep and 6 to 8 in for a 3-ft block, holes are 2.5 to 3 in deep. s and sandstones generally require deep holes. For sandstone, holes are in diam, 4 to 16 in apart, depth being ⅔ the thickness of the block.

Drilling methods. Plug holes are drilled by hand, pneumatic hammer reciprocating drill. For shallow holes, the hammer drill is cheapest;

hand hammer next. For deep holes, a reciprocating drill on a quarry *ba* hammer drill, is most economical. HAND PLUG-HOLE DRILLING. In g 1 man can drill in 8 hr 80 ⅝-in holes, 2.5 in deep; total, 17 ft. Wit 24 to 30 in apart, and wages at 30¢ per hr, cost of splitting a block to 3¢ per sq ft. PNEUMATIC HAMMER DRILLS. In granite, 1 man ca in 8 hr 250 ⅝-in holes, 3 in deep, if the driller does not drive the plu sandstone, 4-ft holes have been drilled in 18 min, and 20 holes, 18 i were drilled at rate of 1 hole in 25 sec. BABY DRILL, on quarry bar, will 3 or 4-in hole in 0.75 min, averaging about 100 holes per day.

Cost of pneumatic plug-drilling in a granite quarry (1910): 6 drillers @ $1 air compressor operator, $3.50; 1 ton coal, $3.00; repairs, 60¢; oil, etc, 30¢; on plant, 60¢; total per day, $17. (To this add drill sharpening and plant Holes were as deep as 4 ft, averaging 18 in. Air press, 80 lb; cutting speed ave in per min. This work was formerly done by 18 hand drillers @ $1.50; total, $

Special quarry methods. BROACHING or broach channeling cons drilling a row of holes very close together, and then, with a BROACH or cutting out the rock between them. One drill on a quarry bar (Sec 1 broach per day: in granite, 10–20 sq ft; marble, 20–30; limestone, 15–35; stone, 20–40 sq ft. GADDER is for drilling rows of horiz holes near the quarry or a vertical or inclined row in the face. One drill has made 350 ft of 2-f in marble in 10 hr. TRACK CHANNELER is a self-propelling machine, tra back and forth on a 10 to 30-ft section of track, and cutting a narrow with a single bit, or one or more GANGS of bits. Some channelers cut v grooves, others can be swung at varying angles, or are arranged for unc ting. Ingersoll "Broncho" channeler is mounted on 2 parallel bars, bling a quarry bar. Channelers may carry a boiler, or be operated by or compressed air from an independent plant. Ingersoll-Rand Co bui air-electric channeler, similar in operation to the air-electric drill. Cha cost $1 200 to $3 000. In dime stone quarries (other than granite are economic necessities, becaus 20% of the stone quarried w channeling is lost in subsequent c In granite, broaching or wedging ally cheaper. Cost of running a neler is about the same as of a drill; 2 men and 0.5 ton coal p are required. COST OF CHANN LIMESTONE, N Y State Barge Can 16 consecutive months follows: Su Y-8 channelers, costing $2 800 were used. Operating crew per neler: 0.16 to 0.5 of the time of a man @ $4 per day, 1 runner @ $3.50, 1 fireman @ $2, 1 helper @ $1.75, 1 la @ $1.50. Cost per sq ft for 126 544 sq ft: labor, 22¢; coal, 2.3¢; water, repairs, 0.1¢; int and deprec, 2.4¢; total, 27¢.

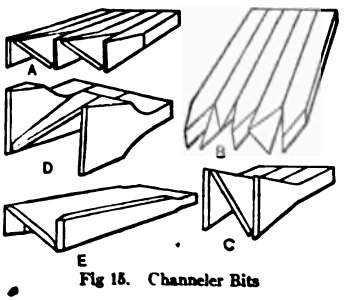


Fig 15. Channeler Bits

Standard or rigid-back channelers cut to depths of 10 to 16 ft; swing and bar channelers, 6 to 12 ft; undercutting channelers, 7 ft. Max inclin of swing-back channeler having boiler or reheater, 24°; other types, 28° bits do not rotate. In Fig 15: *A* shows the gang used in marble or rocks chip freely; *B*, that used for tough rocks which do not chip freely; *C*, for *D* is used in both quarry and contract work for sharp, gritty stones; *E* solid Z-bit, common in contract work in rough broken stone.

Table 16. Rate of Cutting with Sullivan Channelers (from Sullivan bulletins)

Location	Kind of rock	Sq ft cut	Time
Quarry work			
[illegible], Que	*A*	63	10 hr
[illegible]d, Vt	*A*	100	10 hr
[illegible]	*A*	2 509	Aver mo
[illegible]	*A*	80	Aver day
[illegible]	*A*	219	High day
[illegible]	*A*	200	Aver day
[illegible]	*A*	150	Good day
[illegible], Vt	*B*	1 485	Aver mo
[illegible], Vt	*B*	1 677	Aver mo
[illegible]gton, Ill	*C*	6 000	10 days
[illegible], Ark	*D*	130	Aver day
[illegible], Mo	*D*	137 5	Aver day
[illegible], O	*E*	225	Day
[illegible] Keys	*F*	750	Aver 10 hr
Pennsylvania	*G*	60	Aver day
Vermont	*G*	75	Aver day
Virginia	*H*	200	Aver day
Virginia	*H*	262	Aver day
Contract work			
Lockport, Ill	*C*	210	Aver 10 hr
Lockport, Ill	*C*	383	High 10 hr
Sault Ste Marie	*D*	75–100	Aver day
Keokuk, Ia	*D*	80	10 hr
Sault Ste Marie	*E*	70–80	8 hr
Sault Ste Marie	*I*	60–75	8 hr
New York City	*J*	60–75	8 hr
Panama Canal	*K*	120	8 hr

Marble. *B.* Hard marble. *C.* Limestone. *D.* Hard limestone. *E.* Sandstone. [illegible] rock. *G.* Slate. *H.* Soft soapstone. *I.* Tough sandstone. *J.* Gneiss. *K.* [illegible] broken rock.

Knox system of blasting. A number of round holes are drilled, and then reamed by [illegible] the shape shown in Fig 16. In medium sandstone, holes should be 10 to 15 ft [illegible] limestone, about 4 ft apart. Black powder or contractor's [illegible] will split the rock in the direction of the angles. Cost of [illegible] limestone by this system, including placing it on scows for [illegible] breakwater (1900), was 99¢ per cu yd (20).

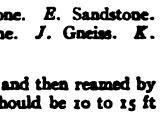


Fig 16. Knox System of Blasting

Quarrying by compressed air is practiced at Mt Airy, N C, [illegible] the granite has few joints, and splits readily in almost any [illegible]. A centrally-located hole, 2 to 3 in diam and 6 to 8 ft deep, is sprung with dynamite. Then, repeated charges of black powder, beginning with a handful and gradually increasing in size, start and extend horiz cleavage cracks. When the cleavage reaches 75 [illegible] ft in all directions, air at 70-lb press is admitted through a pipe cemented into the hole say 0.5 hr, the cleavage reaches the surface 250 ft ± from the drill hole, and the [illegible] slab is split into blocks by plugs and feathers.

Cost of quarrying. A number of detailed examples are given in (20). Cost of 93 500 cu [illegible] (measured in wall) of limestone rubble stone for Chicago Drainage Canal was 74¢ per [illegible]. Cost of thin-bedded sandstone, for dry slope walls and rubble, was 88¢ (measured [illegible] including loading into wagons. Cost of 600 cu yd of sandstone dimension stone [illegible].5 per cu yd, and of 200 cu yd of rubble stone $1 per cu yd. Cutting dimension [illegible] $6.83 per cu yd. Cost of 4 000 cu yd of gneiss dimension stone, ready for cutting, $3.55 per cu yd, and of 3 400 cu yd of rubble stone, $1.80, measured in place in [illegible]. Cost of quarrying granite dimension stone was $4.53 per cu yd.

18. Open-cut Rock Excavation

Side-hill cuts, where rock is wasted directly in front of the excavation, usually the least expensive type of open cut. Ratio of the propulsive [illegible] of black powder to its shattering effect is about 8.6, while that of dynamite is about 1; hence, it is often desirable in side-hill work to use black [illegible], so that when blasted the rock may be thrown as far as possible from [illegible].

High breasts. Quarries, of other than building stone, are most cheaply [illegible] with a high face or breast. Deep, large-diam holes by churn drills are [illegible].

Through cuts occur oftenest in canal and R R work. Depth and wi cut, and mode of removal, determine the plan of attack. Excavation in tl cuts generally costs more than in side-hill cuts.

Cost of excavating 29 295 cu yd of gneiss (mica schist) in street work in N Y pe was (20): foremen and timekeepers, 8¢; derrickmen and helpers, 9.6¢; labor loading hand drillers, 11.7¢; blacksmith and helper, 5.3¢; hauling in wagons, 40.5¢; explosiv coal, coke, oil, etc, 6¢; drill repairs, 1¢; repairs to boilers and derricks, 1.2¢; total, $1 cu yd. Four 3.5-in Ingersoll steam drills made the main holes and 2 baby drills, th holes, and hand drills and sledges broke up the medium-size stone into pieces suit building foundations. The cut ranged from 2 to 63 ft, aver lift, 12 to 15 ft. In sour 20-ft holes were drilled at an aver of 70 ft in 10 hr; in shallow drilling, an aver of n 25 ft per drill per day. There were 18 433 lineal ft of main holes (not including block Wages: 1 machine man @ $3, the others, $2.75 per 10 hr; hand drillers and sledg laborers, $1.50. Drilling plant cost $1 800; boilers, derricks, hoists, etc, $1 080; 40% mite, 20¢ per lb.

Mesabi iron range, Minn (see Sec 10). Comparative cost of open-pit and ground mining. Aver cost of stripping per cu yd: glacial drift and paint roc broken taconite, 75¢; solid taconite, $1. Mining cost in ordinary conditions: shovel, 30¢; underground mining, $1.50 per cu yd. On these bases, the cost of m column of ore 1 sq yd in area by 36 ft deep, with 65 ft of overburden, is:

Underground mining: ore 36 ft thick		$18.
Open-pit mining: Stripping 50 ft glacial drift	$5.00	
" 15 ft solid taconite	5.00	
Steam-shovel mining, ore, 36 ft thick	3.60	13.
In favor of open-pit mining		$4.

Open-pit mining is usually economical when there is not more than 1 cu yd o burden to 1 ton (say 0.5 cu yd) of ore (1 cu yd of hard slate and taconite being e lent to about 3 cu yd of ordinary overburden), and when max stripping depth is le 150 ft. As depth and track grades increase, open-pit mining ceases.

Steam shovels used in Mesabi district are Bucyrus, Marion, and Atlantic, weigh to 110 tons; the smaller for stripping, stock-pile loading, and cleaning-up wor larger for ore excavation. Standard R R cars of 100 000-lb capacity carry the ore 20-cu yd side-dump cars, the overburden. Locomotives, weighing 20 to 100 tons, side-rod and Shay types are used, the latter for steep grades. Pit tracks laid spirals are preferred to switch-backs. Aver output of shovels in stripping is 1 2 200 cu yd per shift for a 60-ton shovel, and 2 000 to 2 500 cu yd for a 90-ton; short periods, 3 000 and 3 800 cu yd respectively. In ore a 90-ton shovel should a 60 cars (960 cu yd) per shift for the season. Table 12, Art 8, gives output of a shovel as 892.5 cu yd; 90-ton shovel, 1 350 cu yd. High records per 10-hr shi 110-ton shovel, 175 cars; 90-ton, 180 cars; 60-ton, 100 cars. Contract stripping 27 to 30¢ per cu yd. Actual unit costs by company account on one property were 22¢. Aver cost of mining is about 15¢ per ton. Wages of steam shovel and crews, to 1913, but since increased, are: shovel runner, $5.97; craneman, $4.04; fir $2.50; locomotive runner, $3.75; fireman, $2.50; brakesman, $2.00 to $2.75; pitmen, laborers, $2.10 per 10 hr.

Blasting preliminary to steam-shovel work is generally done by a special crew. in overburden are by hammer and drill; in ore, by jumper. Gopher holes are dri high banks. Drilling for stripping costs 8 to 15¢ per ft of hole. In ore, 2 to 4 me an aver of 60 ft, and a max of 120 ft per 10-hr, at cost of 7 to 10¢ per ft. Gopher cost 10 to 15¢ per ft in ore, and 7 to 10¢ in stripping; 20 to 25 ft per shift are Holes are sprung with dynamite and blasted with black powder.

As navigation on the Great Lakes ceases in winter, ore is stored through 4 to 5 m in stock piles. Two systems of stock-piling are common, the Michigan and the M sota. In the latter the stock pile is started at end of a short trestle, and is built out end-dump cars (Sec 3, Art 13). The Michigan system has a long trestle. Nega Mine uses a permanent steel trestle. With wooden trestles the timber loss per y 12.5 to 25%. A trestle, 42 ft high, cost $3 per lin ft. Labor cost of stock-piling is power cost, low. Total cost varies from 2 to 7¢ per ton; where 1 000 tons per da

cost is 2 to 3¢; for small output 5 to 7¢ per ton. Stock-pile ore is reloaded by ... shovels (Table 12, Art 8).

Open-pit iron mining, near Rochester, N Y, in 1909 (20). Overburden, 20 ft, consisting of 10 ft of soil and glacial drift and 10 ft of limestone. 2 churn drills drove 6-in holes 6-ft apart, a few inches into the ore. Aver per day per drill, 4 holes (80 ft); cost, per ft, 0.09 ft of hole per cu yd. Cost per day per drill: 1 runner, $2.50; coal, 400 lb, ... supplies, 30¢; plant charges, $1; total, $5.50. Charges, 65 lb of 40% dynamite, ... lb per cu yd, costing 13¢ per lb. Overburden was loaded by a Vulcan shovel, ... a 2.5-cu yd dipper, making a 40-ft cut, 20 ft deep. It excavated an aver of 1 500 ... per day, costing: foreman, $3.50; runner, $2.50; craneman, $2; fireman, $1.75; ... coal, $10.50; oil and supplies, 50¢; 8 laborers, $10.40; plant charges, $6; total, ... Shovels loaded into 5-cu yd skips, running on an inclined conveyor, 129 ft ... and making a waste bank 60 ft high. Cost of operation per day: engineer, $2.50; ... man, $2; topman, $2; fireman, $1.75; 2 tons coal, $7; oil and supplies, 50¢; plant ... $6; total, $21.75. Total cost (not including general expense), per cu yd of ... burden: drilling, 0.6¢; explosive, 4¢; loading, 2.5¢; conveying and dumping, 1.5¢; ... 8.6¢.

... out 8 in of limestone capping adhered to the ore, and was blasted or barred off. ... were drilled through the ore 3 ft apart, and shot with 1.5 to 1.75 sticks of 15% ... size, costing 9¢ per lb. Each cu yd of ore required 4 ft of hole and 1.25 lb dyna... Broken ore was loaded into skips by a Vulcan shovel, 1.25-cu yd dipper, having ... pat of 300 tons (100 cu yd) per day. Skips were handled by a derrick on a car... cost of breaking, loading, and handling: foreman, $3.50; drilling, $12; explosives, ... shovel-runner, $2.50; craneman, $2; fireman, $1.75; coal, $1.25 ton, $4.37; der... engineer, $2.50; winchman, $2; fireman, $1.75; coal, 0.75 ton, $2.62; 8 laborers, ... plant charges, $6; total, $52.16.

... cost per cu yd: foreman, 11.6¢; drilling, 40¢; explosives, 11.2¢; loading, 35.4¢; ... handling, 29.6¢; laborers, 26¢; plant charges, 20¢; total, $1.74.

Excavating rock in open cuts, Grand Trunk Pac R R (20). Cuts, 20 ft wide at ... sloped 3 in to 1 ft. Overbreakage, usually paid for, was 10 to 40%. Rock was ... trap, and diabase. Steam drills used in large cuts, hand drills in small. HAND ... 1-in steel, 1⅜-in bit) made holes as deep as 30 ft. To depths of 6 ft, 2 men struck ... held the drill; below 6 ft, all used hammers, the drill rotating automatically on the ... wages, $2.25 per 10 hr (day's work), or 45¢ per ft, sharpening and nippering ... 3 men, drilling 10 to 14-ft holes, averaged in dark hornblende 29 ft per day, ... granite, 20 ft, and in trap and diabase 18.5 ft. In drilling block holes, 1 gang made ... averaging 15 in each, in 6 days. Drill sharpening for one month, for 5 gangs ... 2 142 ft, cost: blacksmith @ $3.50, $87.50; helper @ $2, $48; nipper @ $2, ... coal, $12; total, $195.50, or 9¢ per ft. Average cost by 5 gangs, each drilling 18 ft ... drilling, 37¢; sharpening, 9¢; total, 46¢ per ft.

... to 30 or 35 ft deep were made by 3.25 and 3.5-in steam drills; holes to 25 ft deep, ... drills. Starting bits, 3.5 in; finishing bits, about 1.25 in. Cost of running 1 drill ... hr day: runner, $3.75; helper, $2.25; fireman, $2.50; 0.5 blacksmith, $1.87; 0.5 ... $1.13; 1 cord wood, $2.25; coal, 30¢; repairs and oil, 38¢; total, $14.43. Aver, ... ft drilled per day, costing 48¢ per ft. When 2 drills were run from 1 boiler, cost was ... 34¢ per ft.

... over 25 ft deep were made in 2 lifts. In bottom benches, 1 ft of hole and in top ... 2 or 3 ft of hole, were chambered. A 26-ft hole, 14 ft from face, was sprung ... 2 sticks 60% dynamite, water-tamped; (*b*) 5 sticks, water-tamped; (*c*) 12 ... water-tamped; (*d*) 30 sticks, sand-tamped; (*e*) 70 sticks, sand-tamped; total, ... sticks. Another similar hole: (*a*) 2 sticks, water-tamped; (*b*) 5 sticks, water-... (*c*) 12 sticks, water-tamped; (*d*) 35 sticks, sand-tamped; (*e*) 100 sticks, sand-... total, 154 sticks. First hole was charged with 275 sticks of 40% dynamite; ... with 150 sticks of 60% and 175 sticks of 40% dynamite. (1 stick = 0.35 lb.) ... holes broke 450 cu yd of rock. Cost: drilling, 4.8¢; springing, 6.3¢; blasting, ... total, 20.4¢ per cu yd. Blaster, 37.5¢, and powder monkey 22.5¢ per hr. Dyna... 13¢ per lb for 40%, 22¢ for 60%, about 0.4 lb of 40% being used per cu yd for the ... shots, and 0.38 lb of 60% for springing. About 75 lb black powder equaled 50 lb ... dynamite. Cost of excavating 7 024 cu yd red granite from a tunnel approach on ... R R was $1.019 per cu yd.

11. Trenching

Overbreakage. Specifications should name a minimum width of t beyond which NEAT LINES the rock removed shall not be paid for. should also be named. Overbreakage in rock sometimes exceeds by 25 o the specified cross-section.

Depth and spacing of holes. Holes in thin-bedded, horizontally str rocks are usually drilled 6 in below specified bottom of trench; in thick-be tough limestones, etc, about 12 in below; in tough granites and traps, below.

For hand drilling in granite, holes are often spaced about 1.5 ft apart. In trench to 3 ft wide, rows, 3 ft apart, of 2 holes each, are common. In a trench 6 ft wide i trap 3 holes per row were drilled, the rows being 3 ft apart. In an 8-ft trench in g there were 3 holes per row, rows 4 ft apart.

Cost of trenching depends primarily on proper spacing and depth of

In the 6-ft trench named above, about 4.5 ft of hole were drilled per cu yd, the going to 1.5 ft below grade. Steam drills made 35 ft of hole per day @ 30¢ per ft. mite, 2 lb of 40% per hole, or 2.6 lb per cu yd of net excavation. Hence, drilling cost and blasting 40¢; total, $2.15 per cu yd. The above 8-ft trench was 12 ft deep; holes to 1 ft below grade, making 2.74 ft of hole per cu yd net. Drills averaged 45 ft in cost per ft, 23¢. Dynamite, about 4 lb of 40% per hole, or 1.1 lb per cu yd. D cost 63¢ and blasting 17¢; total, 80¢ per cu yd.

Cost of trenching in limestone, St Louis, Mo (20). Rock was in horizontal 3-ft strata, the upper 4 or 5 ft being seamy and rotten, the rest hard and difficult to b Rock was excavated 6 in below all pipes of 18-in diam or less and 9 in below larger

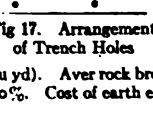


Fig 17. Arrangement of Trench Holes

Excavation was paid for to widths 1 ft greater than diam of of less than 18 in, and 15 in greater than the diam of larger Drill holes were 6 in from side of trench, and staggered 4 ft in top rock (Fig 17) and 2.5 ft apart in hard rock. Projec were sledged or shot off. Holes were drilled in 2 lifts, *top* going halfway through the ledge, bottom holes 0.66 to 0.7 thickness of ledge. Drilling was single-hand, with 1.75-in 10 ft being drilled in 8 hr. Dynamite, about 4 300 lb (2.25 l cu yd). Aver rock broken per 8-hr day per quarry man, 0.96 cu yd. Overbreakage, a 20%. Cost of earth excavation, 50¢ per cu yd. The cost of the rock work was as fol

Table 17. Cost of Trenching in Rock per Cu Yd Net Section, St Louis

Size of pipe, in	Length of pipe, ft	Aver depth in solid rock, ft	Cu yd excavated	Foreman @ $5, ¢	Quarrymen @ $3	Laborers @ $2, ¢	Total direct labor	Blacksmith, ¢	Dynamite, ¢	Total cost, breaking rock	Cost of removal	Backfilling cost, ¢	
21	370	14	600	56	$3.50	24	$4.30	20	40	$4.90	$1.40	15	$6
18	287	12	317	85	3.24	34	4 43	20	40	5.03	1.40	15	6
18	314	13	380	84	2.76	44	4 04	20	40	4.64	1.40	15	6
15	222	11	206	87	3.13	39	4.39	20	40	4.99	1.40	15	6
15	251	8	180	89	2.15	66	3 70	20	40	4.30	1.40	15	5
Aver	...	11 9	....	74	3.10	36	4.20	20	40	4.80	1.40	15	6

Unmounted machine hammer drills are now (1915) often used in tre work, because of their portability and ease of starting a hole.

They are often operated by a portable gasoline-driven compressor, at a daily cost 3 drill-runners @ $3.50, $10.50; compressor man, $3.50; 15 gal gasoline @ 20¢, $3; renewals, deprec, $6; total, $23. Each drill should average 50 ft of hole per day *in* rock, thus costing 14.5¢ per ft.

special carriages, for carrying a boiler, and a drill mounted on a bar, were ... sewers at Havana, Cuba. Wt per outfit, 5 000 lb; drills were Sullivan, ... Rock varied from very soft to flint-like hardness. Time studies of 4 ... drilling 40 holes: total drilled, 366 ft; aver depth of hole 9.15 ft; aver ... time, 26.4 min per hole = 2.9 min per ft; changing steel, 1.4 min per ft; ... drill on bar, 0.6 min per ft; moving machine from hole to hole, 1.0 min ... total time, 2 231 min; average per ft, 6.1 min (20).

Table 18. Rock Excavation in Trenches (Original)

	Width, ft	Depth, ft	Holes in row	Spacing of rows, ft	Depth of holes, ft	Drill speed, ft per min	Kind of drill	Ft hole per cu yd	Dynamite per cu yd, lb (g)	Cost per cu yd		
										Drilling	Blasting	Total
	6	1 5	3	3.0	3.0	35	Steam.....	4.5	2.6	$1.35	$0.40	.....
	8	12	3	4.0	13.0	45	"	2.74	1.1	0.63	0.17	.....
	14	10	5	4	10.0	90	"	2.4	1.0	0.24	0.16	.....
	2.5-3	(b)	....	.....	.....	....	All methods	....	....	.....	.....	$3.80
	...	9	.....	.....	.....	.....	.....	4.5	....	.....	.....	4.97
	2	6.5-12	(c)	.....	(d)	158	Air hammer	....	(e)	.....	.....	3.40
	...	11.6.	2	4(f)	....	12.5	Single-hand	....	2.25	.....	.....	4.80
	..	7	1	3	2.5	....	1 man churn	....	....	.....	.....	2.90

...es usually 3 ft apart. Hammer drilling, 1 holder, 2 strikers, on small jobs. ... for 10-ft holes 2.5 in, finisher 1.25 in. Aver rate of drilling per gang, 8 to 10 ft ... 0.5 to 3 sticks, 75% dynamite per hole. (*b*) Depth of rock, 0 to 12 ft. (*c*) Two ... of holes, 18 in apart, and a center hole staggered. (*d*) 6 to 8 in below grade, ... ft, 4 ft. (*e*) ½ to 0.5 lb per hole. (*f*) Holes staggered, 4 ft apart in soft rock, ... hard. (*g*) Dynamite was 40% to 60%, except (*a*) which was 75%.

12. Subaqueous Excavation

Methods employed are: exploding dynamite on the rock surface, unwater- ... rock by cofferdams or caissons, and drilling from platforms or scows. ... work is a branch of Civil Engineering, to books on which the reader is ... (11, 20).

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SECTION 6

TUNNELING

BY

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Note.—Numbers in parentheses in text refer to Bibliography at end of this section.

Scope of section. The following discussion is restricted to driving of tunnels of small cross-section, the entire area of which is excavated in one operation. Most mining tunnels and adits, and many intended for irrigation and water supply, are small enough to be driven in this manner. For tunnels of large cross-section, as for railroads and some aqueducts, the work consists of two operations: the driving of an advance heading, followed by enlargement to full section. The method of work in such advance headings is essentially the same as for mine tunnels and certain of its features, of interest to the mining engineer, will be noted. In this discussion no description is given of the numerous systems of enlarging, timbering, and lining railroad tunnels, for the details of which the reader is referred to the works of Drinker (32), Prelini (48), Searler (56), and the publications of the civil and mining engineering societies.

1. Representative Tunnels

Tables I to IV contain data relating to 36 representative tunnels, most of them for mines, which will illustrate the chief features of tunneling. These data comprise: location, dimensions, kind of rock penetrated, type and number of drills used in headings, organization of work, kind and consumption of explosive, and cost per foot of completed tunnels. In other parts of this section are additional tables, giving details respecting drills and their mountings for tunnel work, repair costs, number of holes per round, and cost per foot of hole.

Table I. Tunnel Data

Number	Name	Location	Purpose	Length, feet
1	Buffalo Water	Buffalo, N Y	WS	6 575
2	Burleigh	Silver Plume, Col	Dr & De	3 000
3	Carter	Ohio City, Col	Dr & Tr	6 600
	Catskill Aqueduct:			
4	Moodna Siphon	Orange Co, N Y	WS	25 200
5	Rondout Siphon	Ulster Co, N Y	WS	23 600
6	Wallkill Siphon	Ulster Co, N Y	WS	23 400
7	Yonkers Siphon	Westchester Co, N Y	WS	12 300
8	Central	Idaho Springs, Colo	Dr & Tr	9 000
9	Chipeta	Ouray, Col	De	2 000
10	Ft Williams	Ft Williams, Ont	WS	4 820
11	Gold Links	Ohio City, Col	Dr & Tr	3 900
12	Grand Central	New York, N Y	Sewer	3 000
13	Gunnison	Montrose, Col	Irr	30 645
14	Joker	Red Mountain, Col	Dr & De	5 055
15	Laramie-Poudre	Larimer Co, Col	Irr	11 300
16	Lausanne	Mauch Chunk, Pa	Dr	20 000
	Los Angeles Aqued:			
17	Little Lake Div	Inyo Co, Cal	WS & Irr	(*a*)
18	Grapevine Div	Kern Co, Cal	WS & Irr	(*b*)
19	Elizabeth Lake	Los Angeles Co, Cal	WS & Irr	20 860
20	Lucania	Idaho Springs, Col	De & Tr	6 385
21	Marshall-Russell	Empire, Col	De & Tr	6 400
22	Mission	Santa Barbara, Cal	WS	19 560
23	Newhouse	Idaho Springs, Col	Dr & Tr	22 000
24	Nisqually	Alder, Wash	Power	10 000
25	Northwest Water	Chicago, Ill	WS	21 180
26	Ophelia	Cripple Creek, Col	Dr & De	8 500
27	Rawley	Bonanza, Col	Dr & De	6 235
28	Raymond	Ohio City, Col	Dr & De	3 200
29	Roosevelt	Cripple Creek, Col	Dr	15 700
30	Siwatch	Leadville, Col	De	5 000
31	Snake Creek	Heber, Utah	Dr & De	14 000
32	Spiral (two)	Selkirk Mts, B C	Railroad	6 090
33	Stilwell	Telluride, Col	Dr & De	2 600
34	Strawberry	Wasatch Co, Utah	Irr	19 100
35	Utah Metals	Tooele, Utah	Tr	11 780
36	Yak	Leadville, Col	Tr & De	23 800

WS = Water Supply. Dr = Mine Drainage. De = Mine Development. T
Mine Transportation. Irr = Irrigation. R = Rectangular. A = Arched roof.
Square. C = Circular. Hs = Horseshoe. T = Trapezoid. (*a*) 12 tunnels, total len
32 000 ft. (*b*) 9 tunnels, total length, 30 300 ft.

L = Limestone. Gr = Granite. Gn = Gneiss. Sh = Shale. Cg = Conglomer
Ac = Anthracite coal. SGr = Soft granite. MGr = Medium granite. Sl = Sl

Table II. Tunnel Data (*Continued*)

Width, feet	Height, feet	Kind of rock	Power	Ventilation	Size of ventilation pipe	Direction of ventilation current	Drills	Number of drills in heading
5	15.5	L	Steam	(*a*)	(*a*)	(*a*)	Ing-Serg	4
6	7	Gr & Gn	Steam	Fan	10 & 12	Ex	Burleigh / Ing-Rand / Leyner	2
5 5	7 5	Gneiss	Hy & HE	P Bl	10	Ex	Leyner	2
8	18	Gr & Sh	Steam	(*b*)	12	Ex	Ing-Rand	4
18	18	L & Sh	Steam	Fan	14 & 20	Ex	Ing-Rand	4
18	18	Shale	Elect	Fan	12 & 14	Bl & Ex	Ing-Rand / Sullivan	4
18	18	Gr & Gn	Elect	(*c*)	(*c*)	(*c*)	Ing-Rand	4
& 5	8 & 7	Gneiss	Steam	P Bl	19	Ex	Leyner	2
7 5	7 5	" Hard "	Steam	(*c*)	(*c*)	(*c*)	Ing-Rand	2
5	6 5	Basalt	Elect	Fan	15	Bl	Ing-Rand	1
6	8	Gneiss	Steam	Fan	15	Ex	Ing-Rand	1
8	8	Gneiss	.....	(*c*)	(*c*)	(*c*)	Ing-Rand / Sullivan	2 & 3
10	12.5	Gr & Sh	Steam	P Bl	15	Bl & Ex	Leyner / Sullivan	4
12	11	.....	Steam	Fan	15	Ex	Leyner	2 & 3
9 5	7 5	Gr	Hy & HE	P Bl	14 & 15	Ex	Leyner	3
12	8	ShCgAc	Steam	Fan	2 @ 16	Bl	Ing-Rand	2
9	10	SGr	Elect	P Bl	12	Bl & Ex	Leyner	2
9	10	MGr	Elect	P Bl	12	Bl & Ex	Leyner	2
12 3	12.75	Gr	Elect	P Bl	18	Bl & Ex	Leyner	3
8	8	Gr	Elect	P Bl	18 & 19	Ex	Leyner	3
8	9	Gr & Gn	El & Hy	Fan	12 & 13	Ex	Leyner	2
5 & 6	7	ShSlL	Elect	P Bl	10	[illegible] & Ex	Leyner	1
8	8	Gneiss	Elect	P Bl	18	Ex	Leyner	2 & 3
9 5	11	Rhy	HE	Fan	14	Ex	Leyner / Ing-Rand	2
5	(*d*)	L	.....	(*c*)	(*c*)	(*c*)	Ing-Rand	4
9	9	Gr & Br	Steam	P Bl	15	Bl	Sullivan	2
& 8	7	An	Steam	P Bl	12 & 13	Ex	Leyner	2
9	9	Gn & Gr	Steam	P Bl	14	Bl & Ex	Leyner	3
12	6	Gr	Elect	P Bl	16 & 17	Ex	Leyner	2 & 3
6	7.5	Gr	Elect	P Bl	10	Ex	Waugh	2
9 5	6.5	Diabase	Elect	P Bl	16	Ex	Sullivan	2
22	27	L	Steam	....	.....	.....	Ing-Rand	6 & 8
7	7	Cg & An	Elect	Fan	10	Ex	Ing-Serg	2
8	9 5	Sh & Sl	Elect	P Bl	14	Ex	Sullivan	2
12	8	Qtzte	Hy	Fan	12	Ex	Ing-Rand	2
7	7	SlGr	Elect	(*c*)	(*c*)	(*c*)	Ing-Rand	2

= Rhyolite. Br = Breccia. An = Andesite. Qtzte = Quartzite. Hy = Hydraulic. = Hydro-electric. P Bl = Pressure blower. Ex = Foul gases exhausted through g pipe. Bl = Fresh air supplied through ventilating pipe. Bl & Ex = Blow ; but exhaust after shooting. (*a*) Tunnel driven under compressed air, no ... used. (*b*) Jet of compressed air in ventilating pipe. (*c*) No ventilation ... except by opening compressed air line. (*d*) Area equivalent to a 14-ft circle.

Table III. Tunnel Data (*Continued*)

Tunnel number	Mounting of drills	Number of drill shifts daily	Number of mucking shifts daily	Number of drillers per shift	Number of helpers per shift	Number of muckers per shift	Type of haulage	Capacity of cars, cu ft	S
1	VC	3	3	4	4	10	Elect	27	
2	VC	1	1	2	2	3	Horse	....	
3	VC	1	2	2	1	2	Horse	21	4
4	VC	3	3	4	4	10	Mules	41	
5	VC	2	3	4	4	10	Mules	40	
6	VC	3	3	4	4	8	Elect	40	
7	VC	2	3	4	4	8 & 12	Mules	40	
8	HB	1	1	2	2	4	Horse	30	
9	HB	2	2	2	1	4	Mules	20	
10	VC	2 & 3	2 & 3	1	1	3	.....	18	
11	VC	2	2	1	1	2 & 3	Horse	25	
12	VC	1	1	2 & 3	2 & 3	2	2 men	(*a*)	
13	HB & VC	3	3	4	2 & 4	5 to 8	Elect	35& 54	4
14	VC	1	2	2 & 3	2	4	Mule	30	
15	HB	3	3	3	2	6	Mule	16	6
16	VC	3	3	2	2	4 & 5	Elect	78	
17	HB	1 & 2	1 & 2	2	2	5	Mule / Elect	32	
18	HB	2	2	2	2	5	Elect	32	
19	HB	3	3	3	3	9	Elect	32	
20	VC	1	1	3	2	3	Horse	22	
21	VC	1	1	2	2	4	Horse	28 &25	40
22	HB	3	3	1	1	4	Elect	22	40
23	HB & VC	1 & 2	& 2	2 & 3	2	3	Elect	57 &35	40
24	HB	2	3	2	2	4	Elect	27	
25	VC	2	2	4	4	6	Mule	22	
26	VC	3	3	2	2	3	Air	....	
27	HB	2 & 3	2 & 3	2	2	3	Horse	17	40
28	HB	1	1	3	2	2 & 3	Horse	32	40
29	HB	3	3	3	2	4	Mule	16	60
30	HB	2	2	2	None	2 & 3	Elect	33	
31	HB	2	2	2	2	3	Horse	20	4
32	VC	2	2	6 & 8	6 & 8	(*b*)	Horse	108	
33	VC	1	1	2	2	3	Horse	22	
34	VC	3	3	2	2	6	Elect	47	
35	HB	2	2	2	2	4	Elect	32	40
36	HB	3	3	2	2	2	Elect	30	

VC = Vertical column. HB = Horizontal bar. (*a*) Used a ½-cu yd bucket flat car. (*b*) Used a Marion shovel operated by compressed air.

Table IV. Tunnel Data (*Continued*)

[illegible]	Pounds explosive per cu yd	Length timbered, feet	Average monthly progress, feet	Cost per foot of tunnel, dollars	Begun	Completed	References to bibliography
[illegible]	4 8	.....	235	.....	July, '07	Apr, '10	6, 39, 42, 54
[illegible]	..	None	100	20	1869	(*a*)	.....
[illegible]	8	100	200	10–15	1897	(*b*)	.....
[illegible]	....	(*c*)	165	.....	Feb, '10	June, '11	58
[illegible]	4 to 5	(*c*)	220	.....	Mar, '09	May, '11	38, 53, 58
[illegible]	4 3-4 6	(*c*)	300	.....	Oct, '09	Dec, '10	10, 41, 58
[illegible]	4 to 5	(*c*)	155	.....	July, '10	July, '11	.....
[illegible]	5 to 7	100	200	.....	.....	.....	.....
[illegible]	5 to 6	115	340	12 (*d*)	Aug, '07	Mar, '08	23
[illegible]	5 to 10	(*c*)	85	27.89	May, '07	May, '09	34
[illegible]	5 to 6	200	200	19 (*d*)	May, '06	Dec, '12	.....
[illegible]	....	.....	.....	.....	1907	1908	2
[illegible]	5.5	14 500	250	70.66	Jan, '05	July, '09	29, 30, 45
[illegible]	3 to 5	(*e*)	215	.....	.....	1907	35
[illegible]	3.9–4.9	630	509	39.54	Dec, '09	July, '11	9, 13, 22, 27
[illegible]	....	.....	340	19	July, '06	Feb, '12	24
[illegible]	4.5	14 745	(*f*)	(*f*)	1909	1913	8, 49
[illegible]	.....	1 500	(*f*)	(*f*)	1909	1913	8
[illegible]	5 to 6	16 400	350	40.50	Oct, '07	Feb, '11	3, 14, 16, 37, 43
[illegible]	8 to 9	None	125	3.06	1901	1911	.....
[illegible]	..	150	160	18.88	1901	1911	.....
[illegible]	6 to 8	960	210	19.91	.....	1912	.....
[illegible]	.....	1 000	.....	(*f*)	1893	1910	19, 20, 50
[illegible]	....	(*g*)	300	15–20 (*d*)	1910	1912	.....
[illegible]	....	.....	400	.....	1908	.....	5
[illegible]	.....	.....	350	.....	1905	1907	29
[illegible]	6	1 618	350	19.88	May, '11	Oct, '12	51
[illegible]	3 to 4	100	200	15	1903	(*b*)	.....
[illegible]	....	None	285	27.27	Feb, '08	Nov, '10	4, 17, 18, 20
[illegible]	....	600	200	.....	.....	(*b*)	.....
[illegible]	7	350	250	.....	May, '10	(*b*)	12, 46
[illegible]	..	All	105	.....	Jan, '08	June, '09	11
[illegible]	8 to 10	None	150	23.38	1901	1906	.....
[illegible]	5 to 6	2 500	300	36.78	1906	1912	31, 44, 59
[illegible]	4 to 5	500	250	15	1906	(*b*)	47
[illegible]	4 to 5	8 000	200	20	1886	1910	.....

[illegible] ...ves 200 ft while testing Burleigh drill. This was first use of machine drills in ...erican mine tunnel. (*b*) Driven intermittently; not yet completed. (*c*) Lined ...crete. (*d*) Permanent equipment not included. (*e*) Practically all timbered. ...e detailed costs, following Art 19. (*g*) Practically no timbering.

DRILLING

2. Choice of Drill

The prime requisite of a rock drill for tunneling is ability to perform e service under adverse conditions, and in spite of rough usage and abuse. from durability, the important determining factors in the choice of a dr rate of drilling, and costs of maintenance, attendance, and power. (See pressed Air Drills," Sec 15, Art 24–26.)

Durability. The work of a rock drill in tunneling is done under con that would quickly ruin almost any other type of machinery. Consta severe vibration is unavoidable; in many cases the drill is lubricated larly and imperfectly; dirt and grit can be kept out of the moving par by the exercise of the greatest care; the drillmen rarely understand t chine's construction and often ignorantly subject it to shocks and strai not intended to bear. To give satisfaction under such conditions r elimination of complicated details, rejection of parts not absolutely ess determination of the proper size and strength of parts, and selection of ma having proper wearing qualities. Since the pneumatic drill has been in p of development for more than 50 years, it is naturally better able to me difficult conditions of tunnel driving than any of the newer types; it is, fore, the drill almost universally used for such work at the present tim certain cases, notably the Arlberg and Simplon railway tunnels in the Al 33, 48, 52, 55, 56), the Brandt rotary hydraulic drill has been employed a tageously (see *The Engineer*, Jan 25, 1895, p 74). The perfecting of th contained electric drill is now (1913) receiving much attention. The r thus far obtained, although not conclusive, indicate that such a machin possibility to be reckoned with in the future. (Sec 16, Art 15.)

Rate of drilling is of course largely dependent upon the character c rock. Study of Table V (showing that Ingersoll-Rand and Sullivan drills, in shale and sandstone, rarely drill over 10 ft per hr, while Leyner dr granite and other hard rock rarely fall below that figure) seems to warra general statement that a hammer drill gives the higher speed in the r horizontal holes required for tunnel headings. This is probably due t mode of operation, admission of water to the hole, type of chuck, and the reciprocating steel. A piston machine strikes comparatively heavy, sma blows, which quickly dull the bit, especially if the rock be hard; after w until the steel is changed, the penetration must be accomplished by crus The faster blows of the hammer drill, being lighter, do not dull the bit so qu and the penetration is effected by the speedier action of chipping. The a cation of water, through the hollow steel to the bottom of the hole, cool bit, preserves its temper, and in flat holes removes the cuttings promptly the front of the bit. Since in the Leyner chuck the drill steel needs only inserted and given a half turn, the delay in changing steel is minimized. Fi as the bit does not reciprocate, holes are more readily started, when the face is oblique to the axis of the hole, than with a reciprocating drill.

The Brandt hydraulic rotary drill made an average of 8 to 9 ft of hole per hr i gneiss of the north heading of the Simplon tunnel, during three months of 1899. In the heading, where the rock was much harder, the average drilling speed during the same p was 5 ft per hour. The footage drilled by this machine is limited by the extremely h mounting required, which increases the difficulty of shifting and setting up. This is p offset by the fact that the machine utilizes a high percentage, 70% according to Prelini of the theoretical power supplied it, and that the exhaust water injected through the h bit, as in hammer drills, keeps the bottom of the hole clean, thus increasing the effici of boring.

Table V. Rates of Drilling in Tunnel Headings *

Tunnel	Make of drill	Rock	Speed per drill per hr, feet	Remarks
[illegible] Water	Ing-Rand	Limestone	6–7	
[illegible]	Leyner	Granite	10	Approximately
[illegible] Aqueduct:				
[illegible]bout	Ing-Rand	Shale	8	Fair average
[illegible]dell	Ing-Rand	Shale	10.5	Normal conditions
[illegible]	Leyner	Gneiss	8–16	
[illegible]ams Water	Ing-Rand	Trap	2	Extremely hard rock
[illegible]cks	Ing-Rand	Gneiss	8–10	Approximate
[illegible]	Leyner	Breccia	12–15	
[illegible]-Poudre	Leyner	Granite	15	Ordinary conditions
Angeles Aqueduct:				
[illegible]e Lake Division	Leyner	Granite	15.84	Aver of 15 accurately timed shifts
[illegible]erine Division	Leyner	Granite	10	
[illegible]abeth Lake Div.	Leyner	Granite	16	Average for 1909
[illegible]	Leyner	Granite	13	Average 3 drills
[illegible]-Russell	Leyner	Granite	10–20	
[illegible]	Leyner	Shale and sandstone	30	Favorable ground
[illegible]use	Leyner	Gneiss	10	
[illegible]				
[illegible]works end	Ing-Rand	Rhyolite	8–10	
[illegible]ge end	Leyner	Rhyolite	10	Much harder rock
[illegible]	Sullivan	Granite	8–10	
[illegible]	Leyner	Andesite	15–20	Average of 4 accurately timed shifts, 19.7 ft
[illegible]nd	Leyner	Granite	8–12	
[illegible]	Waugh	Granite	10–12	Stope drill
[illegible]reek	Sullivan	Diabase	8–12	See below †
[illegible]	Ing-Rand	Andesite	5–10	
[illegible]erry	Sullivan	Shale and sandstone	13	Test run
[illegible]Metals	Ing-Rand	Quartzite	6–8	Test run

* [illegible]des time used in setting up and tearing down column or bar, in shifting ma[illegible] new holes, and in changing steel; but does not include mucking set-up nor [illegible]charging, blasting, and clearing smoke.

† [illegible]ng a competition test in which both drills were mounted on the same bar in [illegible]ecure identical conditions of rock, etc, a Sullivan machine drilled 21 ft per [illegible]e a Leyner drill made but 20 ft. The conditions were unusual, however, be[illegible]er under pressure was encountered in practically every hole drilled, and doubt[illegible]ensed the results greatly.

[illegible] electric drill, during experiments on the Catskill aqueduct, N Y, is reported to [illegible]ed from 4 to 10 ft of hole per hr in gneiss. An electric drill of another make [illegible] average of 12 ft per hr in mica schist, while in Fordham gneiss the average was [illegible] ft. A third type of electric drill is said to have averaged about 10 ft of hole [illegible] mica schist (data from private communications to author).

[illegible] of repairs. Table VI shows the cost of repair parts for pneumatic [illegible] a number of tunnels. Corresponding data for hydraulic and electric

drills are not available; but it is probable that the former, because of i and complicated parts, is more expensive to maintain than the air dril is certain that the repair costs of the self-contained electric drills ar than the figures in the table, because they are still in the development

Table VI. Cost of Drill Repair Parts

Tunnel	Dates	Tunnel advance, feet	Rock	Drill
Carter	Jan, '10–June, '11	3 000	Granite	Leyner
Gunnison	Mar, '06–Aug, '06	.....	Granite and sandstone	Sullivan
Los Angeles Aqueduct, Little Lake Division:				
1 B South	July, '09–May, '11	1 030	Granite	Leyner
2		1 344	Granite	Leyner
2 A		835	Granite	Leyner
3		3 013	Granite	Leyner
4		1 173	Granite	Leyner
7		2 935	Granite	Leyner
8		1 559	Granite	Leyner
9		3 256	Granite	Leyner
10		4 402	Granite	Leyner
10 A		3 129	Granite	Leyner
Elizabeth Lake	Jan, '09–Jan, '10	4 476	Granite	Leyner
Marshall-Russell	June, '08–June, '11	6 000	Gneiss	Leyner
Strawberry	Jan, '09–June, '12	19 000	Shale and sandstone	Sullivan
Rawley	May, '11–Oct, '12	6 235	Andesite	Leyner
Yak	Sept, '07–Aug, '08	3 000	Granite and sandstone	Leyner

* Includes materials for drill repairs but does not include drill steel to repl wasted in sharpening or destroyed by crystallization.

† Includes labor of machinist making repairs, which is not included in the other

The attendance required by the Leyner drill is somewhat less than other machines. For most types of the larger size compressed-air pisto (also for electric machines), a runner, and a helper are required for eac and sometimes an extra man is provided for every three machines to the dulled steel. Different engineers report that from 5 to 7 men are r with the Brandt hydraulic drill. With the Leyner machine and one forms of piston drill, although a runner is necessary for each drill, one can usually attend to two drills or two helpers to three drills, and at th time keep a supply of sharp steel on hand.

Power required. The most serious disadvantage of the pneumatic its well known wastefulness of power. The lowest catalogue rating for consumption of any type of drill used in tunneling is 65 cu ft of free air p at 100 lb pressure, while drills using as much as 150 and even 175 cu min are numerous. On the basis of 20 hp for every 100 cu ft of free min compressed to 100 lb pressure, and without any allowance for loss of through friction in pipes or leakage in the machines as they become w pneumatic drill requires the application of from 13 to 35 brake hp at th

or while the drill is in operation. The Brandt rotary drills in the Simplon consumed 13 hp (the minimum figure for air drills); electric drills re- a maximum of 6 hp and some of the smaller machines run on less than

The conclusion from these data is that the pneumatic drill, because of its re- and low cost of maintenance, is best for tunnel work, in spite of its power consumption. As between the different air drills, the Leyner (now the Ingersoll-Leyner) is preferred by many for tunneling be- 1. it consumes somewhat less power and affords greater speed in drilling nearly horizontal holes required in tunnel headings; 2, it eliminates dust. reliability and its cost of maintenance and attendance are, for all practical identical with other pneumatic drills. If in the future, however, the obtained, electric drill can be so improved as to compare favorably with air drill in reliability, its lower power consumption will make it a formid- rival of the pneumatic machine.

3. Number of Shifts per Day

One shift. **Advantages:** 1. Drilling and mucking are done on separate the heading is clear when the drill-shift comes on, so that the machines set up at once and the next round begun. The drill-men and helpers, waste no time in mucking, preparatory to mounting the drills; an point when columns are used, because for setting them up the débris be cleared out down to the floor. 2. During the drilling the drill-runners helpers are not hampered by the muckers, with the consequent waste of and energy hardly to be prevented when two crews are working simul- in the restricted space of the heading. 3. Starting promptly, the holes can usually be completed within the allotted time. But, even cannot be done, sufficient extra time is available without delaying the shift. 4. The drilling and mucking shifts can be so arranged as to of time in waiting for blasting smoke to clear away: a serious con- where ventilation is poor.

One shift. **Disadvantages:** 1. Since the daily progress in driving is limited advance gained from a single round of holes, the total rate of advance Most tunnels are useless until completed. Hence, if the work is tened to the utmost, the capital invested in tunnel equipment is tied up than necessary, whence the charge for interest and depreciation is pro- increased. 2. The realization of benefits to be derived from the is delayed. For example, if a drainage adit is being driven, the ex- of ore below water level is delayed; or, if the adit is intended to lower of underground transportation, the loss on the additional tonnage in the old way should be charged against the slower method of driving Similarly, with an irrigation tunnel, an entire season's crops may because of the increased time required by the one-shift system. 3. The administration and other fixed charges is operative during the full period truction; hence, these charges per foot of tunnel will be smallest when maximum number of hours per day are employed in driving. In conclusion, the one-shift system is cheaper in wages, this economy is offset by due to delay in completing the work.

Two shifts per day permit faster progress than one shift, and if the work be duly organized there will be but little added cost per foot for excavation. muckers should begin work in advance of the drillers, and first remove the débris from the face to permit the drillers to set up their machines

promptly. If the drilling and mucking shifts begin simultaneously a drillers take part in the work of clearing back for the set-up, not only higher priced men lose time, but the two crews seriously interfere wit other. In a few tunnels, three gangs of shovelers have been required to the rock broken by two rounds. This system is obviously expensive, l the cost of the extra shovelers must be charged against an advance w but slightly, if at all, increased by their efforts; furthermore, this plan the disadvantage of simultaneous working of two crews. When two seem necessary, it is probable that a change should be made either in the and number of the holes or in some other details.

Three shifts per day. The chief advantage of this system is greate progress, because the full 24 hours are employed in driving. A certain a of crowding in the heading is unavoidable, but it is more than offset by efficiency due to the necessity for perfecting the organization. With th shift system, where drilling and mucking are done simultaneously, th know that they have some leeway before the next shift comes on, an speed is not so urgent; but with three shifts the reverse is the case muckers have constantly before them the task of removing the waste the drillers have finished their work. Conversely, the drillers endeavor t plete their round of holes by the time the muck is cleared away, so t delay may be attributed to them. Both crews are in turn inspired to work by the knowledge that they are expected to finish their labor with eight hours and that they will be followed immediately by a competing Besides, after the holes are drilled, the muckers assist the drillers in down the machines and removing them to a place of safety during bl The time spent by drillers in clearing muck and setting up machines reduced by the use of horizontal-bar mountings and by properly directi blasting the holes. Delays due to adverse conditions are not always pr able, but the men are stimulated by rivalry with other shifts to reduce t the minimum. While the ideal results of the 3-shift system are obtaine by perfected organization, still, by its adoption, the maximum efficiency realized.

4. Choice of Drill Mounting

American practice of mounting drills for tunnel work is almost evenly d between the horizontal bar and the vertical column (for details, see S Art 27), but, since drill carriages have been employed in several Eur tunnels, some reference to them will be made.

Horizontal cross-bar. The mode of employing this type of mountin best be shown by a description of its use at the Laramie-Poudre tunnel 22, 27).

As soon after blasting as ventilation permitted (ordinarily 10 to 15 min), the wo returned to the face, with a tunnel car containing the cross-bar, machines, tools, etc. The three drillers, sometimes with the assistance of the foreman, after ren any loose rock from the roof, cleared a space in the top of the pile of broken rock f or three feet back from the face and four or five feet from the roof, to get room for ha the drills. Owing to the mode of blasting employed, little work was ordinarily re to clear out the required space. Meantime, the helpers unloaded the bar and ma from the car, placed them on the rock pile, and connected the hoses to the air and mains. As soon as the necessary space was cleared, the bar was held in position b drillers and helpers transversely across the tunnel, at a measured distance from th and roof, where it was blocked, wedged, and screwed tightly in place. The ma were then mounted, the hose connected, and drilling started. As many as possi the round of holes having been drilled from this position of the bar, and the waste

sometime been removed by the shovelers (an operation which was usually finished the holes were down), the machines were dismounted, the bar lowered and reset 18 above the floor. The drills were then replaced and one or two holes drilled by achine from this position of the bar. The drilling outfit was then put on a car moved from the heading during blasting. This system, sometimes slightly modi- has been used at several other tunnels and adits with equally good results.

The **vertical column** permits a procedure similar to the above in some re- but different in certain important particulars. Owing to the vibration ed by the drills, neither method of mounting is satisfactory unless the rmly wedged against solid rock. With a vertical column the vibration ensified, because the drills are usually mounted on arms projecting at ngles from the columns, thus affording a leverage for any movement of rill. Hence a solid foundation is all the more necessary, and to secure it the débris must be removed from immediately in front of the tunnel This may cause considerable delay, even under normal conditions, ally with the 3-shift system. In the majority of cases where vertical are employed, not more than two rounds per day are attempted, and extra work of clearing away is performed by a crew of muckers before the begin work.

Carriage mounting for drills has been used with good results at the Loetsch- tunnel (21, 52, 55, 56) in the Bernese Alps, and elsewhere.

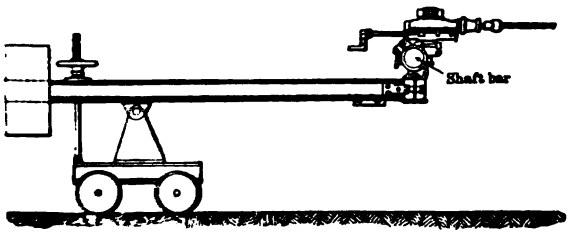


Fig 1. Drill Carriage, Loetschberg Tunnel

The first Loetschberg type (Fig 1), a horizontal bar carrying the drills was mounted end of a steel beam, which was pivoted on a truck and counterbalanced by a heavy But, even with a long beam, before the carriage could be brought sufficiently face for the cross-bar to be in position, it was necessary a passage partway through of the rock pile. While part of the material was away, the remainder being both sides of the tunnel, to moved during drilling. When age was finished, the drill was quickly moved to the cross-bar swung around and position, and drilling This carriage was super- one without the counter- beam, the drill bar being

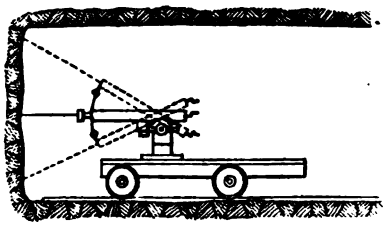


Fig 2. Modified Drill Carriage, Loetschberg Tunnel

rectly upon a short post mounted on the truck (Fig 2). With this device prac- of the broken rock had to be removed before the carriage could be brought to

The time required to begin drilling after blasting, including both tha in setting up the machines and that consumed in mucking, is an importa sideration in the choice of method of mounting. With a *horizontal bar* Laramie-Poudre tunnel (22), the normal cycle of operations was as f mucking back, 15 to 20 min; wedging the bar in place, 5 to 10 min; mo the drills and coupling the air and water for Leyner drills, 10 to 15 min. total time under ordinary conditions was 30 to 45 min, but it was not u for the drills to be in operation within 20 or 25 min after the drillers r the heading. At other tunnels using horizontal bars the time requir similar work was 30 to 60 min. When a *vertical column* is employed, to the larger amount of material to be mucked, the time consumed in s up the drills, with the 3-shift system, ordinarily ranged from 2½ to even in favorable circumstances rarely less than 2 hr. In the Loetsc tunnel the mucking time was about 1½ hr with the first type of ca and 1½ to 3 hr with the later model; but to accomplish this, nearly tw many men were employed as are usually found in American tunnel hea After the heading was cleared, however, the machines could usually be s in 5 to 10 min.

The amount of broken rock to be removed has another bearing on the lem, aside from the time consumed in clearing. To reduce delay in st drilling, it is not customary to clear the heading completely before settir much of the rock being merely shoveled to one side and removed later. preliminary work is often done by the drillers, especially with the 3-shi tem; and where (as with the vertical column) much of it is necessary, men are tired by the time the machines are ready, and consequently ca work so rapidly nor efficiently in drilling the round. On the other ha the temporary clearing away is done by the muckers, they lack stimu knowing that it is dead work and that every shovelful must be moved later. This disadvantage exists also with 2-shift work, when the mu start ahead of the drillers to clear away for a column set-up. The hori bar and, to a less degree, the drill-carriage methods have the advantage quiring a smaller proportion of duplicated work.

Adaptability of mounting. With a vertical column and piston drills, o hammer drills on a horizontal bar, the holes can be placed effectively. with either type of drill, carriage mounting is somewhat handicapped the Loetschberg tunnel it was found that, with the carriage there used holes could not be pointed to secure maximum efficiency from the expl Though the average depth of the round (about 4 ft) was much less than i center-cut system, heavy charges were necessary to make the holes bre the bottom. With the shallower holes more time was spent in the dead of setting up and tearing down the drills. Moreover, from a single set- the carriage mounting, the bottom row of holes (lifters) could not be d near enough to the horizontal to leave an even floor, and trimming was s times required to maintain the grade. On the other hand, dynamite is ch than labor, and, since shallow holes are of larger diameter at the bottom deep ones, the explosive is more concentrated. The rock is therefore br smaller and projected farther from the face, thus facilitating mucking, th at an increased cost of explosive (Art 7).

Wear and tear is less with the carriage mounting, because the drills are ke the bar continuously and are not thrown around on the floor or on the muck Thus, dirt and grit do not get into the drills, increasing friction and abrasion the cost of repairs, as well as leakage of air, with consequent loss of efficie

Facility in changing to a new hole is an important advantage possessed ... horizontal bars and drill carriages. When these mountings are ...ed, to start a new hole it is necessary only to slide the drill along the bar ...mp it in place. With a vertical column, not only the machine but the ...ing arm as well requires adjusting, and, since the adjustment is vertical ... horizontal, the entire weight of drill and collar arm must be lifted or ...ed at nearly every change.

Summing up the various factors, the horizontal bar proves to be the best ...g for tunnel work. It enables drills to be put into operation with a ... loss of time and by the fewest hands. It minimizes rehandling of ... The holes can be pointed in such a manner that the maximum strength ... explosive is utilized; deep holes are drilled, so that the time spent in ... drills is diminished, and the holes are so placed as to insure breaking ... and floor smoothly and at the desired grade. It is especially adapted ... rapid-drilling hammer machines. In common with the vertical mount... is subject to the disadvantage of allowing grit to become lodged in the ... but this can, in part, be prevented by care in handling.

5. Number of Holes per Round

Physical character of the rock is the most important factor in determining ... proper number of holes per round in a tunnel of given cross-section, and, as ... seldom twice alike, the exact number can be ascertained only by trial. ...eral, igneous rocks require more holes than sedimentary, but there are ...fferences in both classes. The holes must be spaced more closely in a ... close-grained rock, than in a brittle rock, even though the latter may be ... and more difficult to drill. Bedding planes and joints are of great assist... rendering a rock more easily broken and with fewer holes. If the number ... is correct, most of the rock will be broken into fragments small enough ... shoveled readily. The blast should be heavy enough to project a part of ... rock some distance from the face; this saves time in setting up the ma... and gives the shovelers more room, enabling them to work to better ...ntage on the scattered material. Table VII shows the number of holes ... used in American tunnels, for different kinds of rock.

Table VII. Number of Holes per Round in American Tunnel Headings

Tunnel	No of holes	Rock	Area of heading, sq ft	Square feet of heading per hole	
				Sedimentary rocks	Igneous rocks
...	16	Granite and gneiss	42	.....	2.6
... Water	22	Limestone	120	5.5	.....
...	10–11	Gneiss, granite, and porphyry	41	.....	3.7–4.1
... Aqueduct:					
... Foust	22	Limestone, sandstone, and shale	120	5.5	.....
...kill	24	Shale	120	5.0	.....
...ma	24	Sandstone and shale	120	5.0	.....
...kers	21	Gneiss	120	.....	5.7
...al	18–24	Gneiss	35	.....	1.5–1.9

Table VII (*Continued*). — Number of Holes per Round in Americ Tunnel Headings

Tunnel	No of holes	Rock	Area of heading, sq ft	Square heading
				Sedimentary rocks
Chipeta	15–19	.....	57	.....
Ft Williams Water	14–20	Basalt	35	.....
Gold Links	12	Gneiss and granite	48	.....
Grand Central Sewer	18	Gneiss	40	.....
Gunnison	24	Altered granite	60	.....
Joker	19–21	.....	130	6.2–6.9
Laramie-Poudre	21–26	Close-grained granite	70	.....
Lausanne	15–21	Shale, conglomerate, and coal	85	4.0–5.6
Los Angeles Aqueduct:				
Elizabeth Lake	25	Granite	145	.....
Little Lake Division	14–16	Medium granite	90	.....
Grapevine Division	20–21	Hard granite	90	.....
Lucania	25	Hard granite	65	.....
Marshall-Russell	18–20	Granite and gneiss	72	.....
Mission	12–14	Shale and slate	37	2.6–3.1
Newhouse	19	Gneiss	65	.....
Nisqually	18	Rhyolite	95	.....
Northwest Water	22	Sedimentary rock	110	5.0
Ophelia	20–24	Granite	80	.....
Rawley	25–27	Andesite	55	.....
Raymond	14	Gneiss and granite	80	.....
Roosevelt	24–26	Hard granite	60	.....
Siwatch	12	Granite	45	.....
Snake Creek	18	Diabase	65	.....
Spiral	21	Limestone	175	8.4
Stilwell	16	Conglomerate and andesite	50	3.1
Strawberry	16–18	Limestone, sandstone, and shale	50	2.8–3.1
Utah Metals	12–16	Quartzite	80	5.0–6.6
Yak	18	Limestone, sandstone, shale, and granite	50	2.8

6. Arrangement of Holes

The arrangement of drill holes, greatly influenced by local conditio arely the same in any two tunnels; but nearly all the systems adopt merican mining tunnels may be classified under three heads, according t ype of cut employed.

The "cut" or "cut holes" are those which are pointed in such manner that, lasted first, they remove a core or wedge of rock from the solid face, thus decreasir ork to be done by the remaining holes.

The wedge or "V" or center cut, the one most commonly employed, ists of several pairs of holes directed toward each other from opposite sid

...sing. They break out a wedge-shaped prism, usually extending from ... to roof.

... 3 shows a typical wedge cut, like that of the Buffalo Water tunnel (6, 39, ... Holes 1 to 8 comprise the cut and were blasted simultaneously by elec-

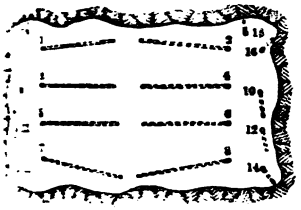


Fig 3. Wedge-cut Round of Holes

Fig 4. Wedge-cut Round for Arched Headings

... Side holes 9 to 14 were next fired together, the back holes 15 to 18 ... fired last.

For arched or semicircular headings the arrangement of holes is somewhat ...ied. Fig 4 illustrates the round used in the headings of the large siphon ... of the Catskill Aqueduct (10, 38, 41). The cut holes 1 to 6 were blasted ...er, followed by the "relievers" 7 to 12 and finally by the "trimming" ... to 22.

With a horizontal-bar mounting further modification is necessary, though ...eneral arrangement remains the same. Fig 5 shows a round employed in ...aramie-Poudre tunnel (22). Holes 1 and 2 were called "short cut" holes; ... "long cuts"; 7, 8, 9, 10, 19, and 20, "relievers"; 11 to 14, "sides"; 15 to ...acks"; and 21 to 23, "lifters"; the numbers indicating the order of

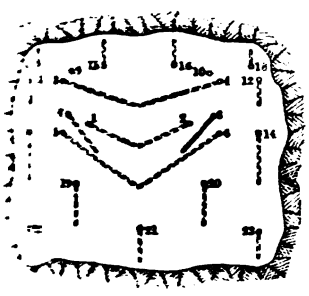


Fig 5. Wedge-cut Round Drilled from Horizontal Bar

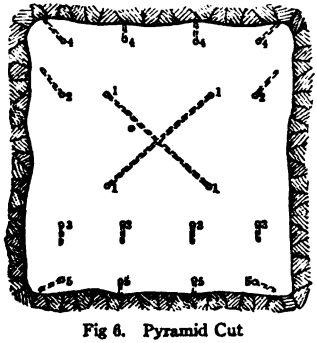


Fig 6. Pyramid Cut

...ge. The lifters and two relievers, 19 and 20, used only in hard ground, ...e only holes drilled from the lower position of the bar. Three machines ...employed per round, the left-hand machine drilling holes 17, 11, 5, 13, 7, 1, ...d 22; the center machine drilling holes 16, 10, 15, 9, 3, and 21; the right-...machine drilling holes 18, 21, 6, 14, 8, 2, 20, and 23, in the order stated.

The pyramid cut consists usually of four holes, drilled so that they or nearly meet at a common point (generally near the axis of the which, when properly blasted, remove a more or less pyramidal core. shows a round of this type as employed at the Yak tunnel (Tables I No 36), in which the cut holes (numbered 1) were blasted simultaneou lowed by the remaining holes in the order indicated. In most cases the cut is made with vertical columns, but it can be drilled just as effectiv the horizontal bar, by making 2 or possibly 3 holes with each machine f lower set-up (Fig 7).

The bottom or draw cut (Fig 8) has been employed at a number of of which the Carter (Tables I to IV, No 3) is typical. The holes were in the order indicated, 1 to 3 comprising the cut. This method often fa

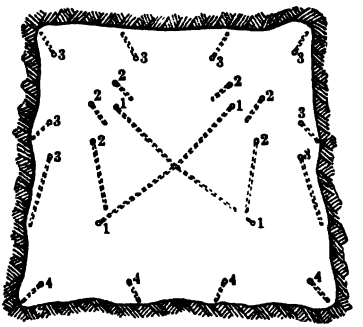


Fig 7. Pyramid-cut Round Drilled from Horizontal Bar

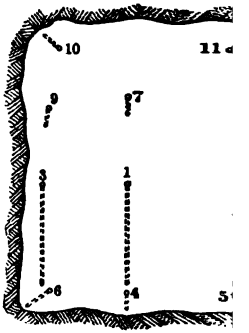


Fig 8. Bottom-cut Ro

the only solution for placing holes in a small tunnel, in which it is impract to drill an effective wedge or pyramid cut. The cut holes should be from as near the top of the heading as possible, and so directed that th be assisted by "lifter" holes (Fig 8, Nos 4, 5, 6), both rows being blast gether.

Importance of the cut holes. It can be proved theoretically that, w hole is drilled in homogeneous rock, the maximum efficiency is obtained the charge when the "line of least resistance" (the shortest distance fro charge to a free surface of the rock) is at right angles to the axis of the and that the minimum efficiency occurs when the two coincide. Althou practice a homogeneous rock is rare, and the results of drilling and blasti quantitatively influenced by differences of rock texture and the preser absence of bedding planes, joints, and cracks, yet these results are found in general agreement with the theoretical deductions. It is obvious that tunnel heading, where there is but one free face, it is impossible to dril blast a single hole in such manner as to produce maximum efficiency. with a number of holes, arranged according to any of the preceding sys and blasting the cut first in order to increase the area of free surface, bett sults are secured from the remaining holes. Hence, the position and dire of the cut holes are of prime importance, the others being merely spaced ficiently close together to break the rock small enough for easy handling.

The angle between opposite holes of the cut should be as wide as can be ...ed with a given depth of round, so that the line of least resistance may ...rmate a perpendicular to the axis of the hole. Therefore these holes ...d start as close as possible to the opposite walls of the heading, keeping ...w the necessity for leaving sufficient space for handling the bits and feed... drill. For these requirements in narrow headings a greater proportion ...e total width must be sacrificed than in wide headings; in the latter an ...ntageously wide angle is easily secured.

...onnecting cut holes. Owing to mechanical reasons, the gauge of drill bits ...e decreased with each successive change of steel; the hole is therefore ...st at the bottom, just where it should be as large as possible, to con...e the charge. This defect can be partly overcome by drilling opposite ...s so that they connect, thus concentrating the explosive at the point of the ... It is specially desirable to do this when firing with fuse, because of the ...al impossibility of gauging the lengths of two fuses to make them det... their charges simultaneously. Furthermore, the combined efficiency of ...o connecting charges is greater than when exploded separately. Assum... that the holes are in homogeneous rock, and that they make equal angles ... the free face, if the charges are detonated simultaneously their maximum ...ng effect will be exerted along the resultant of their forces, which theo...ly coincides with the line of least resistance. The result may be some... modified in practice, but it is an established fact that where the ground ...gh and difficult to break, better results are obtained by drilling the cut ... to intersect.

7. Depth of Hole

Advantages of shallow holes. 1. With properly directed holes, an in...ed efficiency is obtained from a given charge; for, since the width of the ...g is practically constant, the angle between the line of least resistance ...e axis of the drill hole is a function of the depth of round. The width of ...ade, therefore, and hence the efficiency, increases with shallow holes. ...e holes may be of larger diameter at the bottom, thus increasing their ...ity for explosive and concentrating it where most needed. 3. Since drill ...e not charged to their full depth, the mass of rock between the outer ... a charge and the free face (which may be considered a measure of the re...e to be overcome) is not so great. 4. The rock tends to break into ...r fragments. 5. The rock is projected farther, instead of being piled ...mediately in front of the new face. This facilitates mucking and saves ... setting up the drills. 6. The same care is not required in starting ...w holes, nor is it necessary to direct them with so great accuracy; never... the desirability of connecting the cut holes must not be overlooked. ... entire cycle may be completed in a single shift. Men do not work so ...without rivalry or special incentive, and it is important to have an accu...measure of the work done by each shift. In most cases there is a sufficient ... of time for ordinary delays.

The chief disadvantage of shallow holes is that a greater number of rounds ...e made to secure the same daily advance. This results in waste of ... time; for, under ordinary conditions, one hole of a given depth can be ... quicker than two holes of the same total footage, because of the time lost ...ging to a new position. Though the difference in drilling time itself ...ly small, each extra round required to make a given advance causes ... time in charging and blasting, waiting for smoke and gases to be re-

moved, clearing the débris from the face, and in setting up the drills, all o is unproductive work.

The above conclusions are substantiated by the results obtained from shallow in the Loetschberg tunnel, previously cited.

Length of tunnel: 47 676 ft, driven from the two end headings only. (Headi holed through, March 31, 1911; total time, 53 months; by machine drills, 46 51 driven in 1304 days.)

Bottom heading: 6 ft 6 in. high by 10 ft wide (nearly same size as some doub mine tunnels).

Rock: hard granite and gneiss.

Holes per round: usually 12 to 15; depth, 4 ft; diameter at bottom, 2 in.

Arrangement of holes: 4 vertical rows, of 2 to 4 holes each; all nearly normal

Drills used: from 4 to 5 Ingersoll-Rand, F-94, 3½ in diam.

Approximate time for a complete round:

	Mi
Setting up drill carriage and cross-bar	
Drilling	
Removing carriage from heading	
Charging and firing	
Clearing out smoke	
Mucking through middle of rock pile to make room for setting up carriage	
Total time (4 hours)	

Work carried on in three 8-hr shifts, each shift drilling and firing two rou which basis the normal rate of advance would be 7 ft per shift, or 21 ft per 24 hr rate was reduced by ordinary delays to a general average of 18 ft per day, in each h The maximum advance in any month was 944.6 ft (28 days), or 33.7 ft per day (39, 55).

The advantages resulting from a change to a shallower round were str brought out in the Laramie-Poudre tunnel (9, 22).

During the 7 months from April 1 to Oct 31, 1910, at the east end of the tunnel, were driven (average, 453 ft per month) using a round 10 ft deep in a heading wide; but during the next 8.8 months, from Nov 1, 1910, to July 24, 1911 (wh tunnel was holed through), 4 798 ft were driven, or an average of 545 ft per with a 7-ft round. This 22% increase of speed was made notwithstanding that th was farther from the portal; and, since there was no change in equipment or ch of rock, it may be attributed solely to the shallower holes. For an advance of 9 ft with 10-ft holes, it was unusual to drill and blast more than two rounds in (frequently less), as the daily average of 14.5 ft testifies. But with 7-ft holes, n could 3 rounds be made, advancing an average of 6.5 ft each, but a good margin was left for delays, which often meant increased footage. Thus, in March, 191 American hard-rock monthly record of 653 ft, or over 21 ft per day, was establis

At the Rawley tunnel (51), in driving through andesite, 8 ft holes were drill heading 7.5 ft wide. The cut holes were blasted separately, and it was usually sary to reload and shoot them a second and occasionally a third time. The cyc therefore lengthened to about 10 hr, and sometimes 14 hr were needed. Lat depth of hole was reduced to 5 ft, making it unnecessary to shoot the cuts sepa thus, instead of only two 7.5-ft rounds in 20 to 22 hr, by working 3 shifts it was p to drill and blast four, sometimes five, 5-ft rounds per day. This increased th progress from 15 to nearly 23 ft, at small added cost for labor and with a decre fully 25% in the cost of explosive.

The most economical depth of hole must be determined by trial in ea dividual case, because of the numerous variables influencing the result. Us a depth of round of from 60 to 75 % of the width of the heading will be satisfactory. Table VIII gives an analysis of American practice:

Table VIII. Depth of Round in American Tunnels

	Type of cut	Height of heading, ft	Width of heading, ft	Average depth cut holes, ft	Average depth other holes, ft	Average depth round drilled, ft	Percentage average depth of round to width of heading	Rock
[illegible] Water . . .	Wedge	8	15	8	7	7	46.5	Limestone
[illegible]	Bottom	7½	5½	9	8	8	106 *	Gneiss, granite and porphyry
[illegible] Aqueduct:								
[illegible]bout	Wedge	8	14	10	8	8	57	Limestone, sandstone, and shale
[illegible]cil	Wedge	8	14	12	10	10	71.5	Shale
[illegible]ota	Wedge	8	14	10	8	8	57	Sandstone, shale
[illegible]ams	Wedge	8	14	8	6	6	48	Gneiss
[illegible]	Wedge	7	5	8	7	7	140	Gneiss
[illegible]ams	Bottom	6½	5	6	5	5	77 *	Basalt
[illegible]ks	Bottom	8	6	6	5	5	62.5 *	Granite and gneiss
[illegible]on	Wedge	6	10	7	6	6	60	Altered granite
[illegible]	Wedge	11	12	10½	9	9	75	. . . . .
[illegible]-Poudre	Wedge	6½	9½	8	7	7	74	Close-grained granite
[illegible]ez	Wedge	8	12	8	7	7	58	Conglomerate, shale, and coal
[illegible]a	Wedge	8	8	9	8	8	100	Hard granite
[illegible]-Russell .	Pyramid	9	8	10	9	9	112	Granite and gneiss
[illegible]	Bottom	7	5	8	7	7	100 *	Shale and slate
[illegible]t hard [illegible]rand)	Pyramid	7	5	8	7	7	140	Sandstone
[illegible]ras	Pyramid	8	8	6½	5½	5½	69	Gneiss
[illegible]y	Bottom	11	9½	8	6½	6½	59 *	Rhyolite
[illegible]est Water	Wedge	10	13	10	9	9	69	Sedimentary
[illegible]	Wedge	9	9	7	6	6	67	Granite
[illegible]er	Wedge	7	7½	9	8	8	106	Andesite
[illegible]ad	Wedge	9	9	12	10	10	111	Gneiss and granite
[illegible]est	Wedge	6	10	7	6	6	60	Hard granite
[illegible]	Bottom	7½	6	5	5	5	67 *	Granite
[illegible]reek . . .	Wedge	6½	9½	6½	5½	6	63	Diabase
[illegible]	Wedge	10	16	12	10	10	63	Limestone
[illegible]	Wedge	7	7	6½	6	6	86	Conglomerate and andesite
[illegible]ery	Wedge	6	8	7	6	6	75	Limestone, sandstone, and shale
[illegible]als	Bottom	8	10	6½	6	6	75 *	Quartzite
. . . . .	Pyramid	7	7	5	4	4	57	Limestone, sandstone, shale, and granite

* Height of the heading, instead of its width, is considered in this ratio when the [illegible] is employed.

BLASTING

8. Choice of Explosive

(See Section 4 for full treatment of Explosives)

A suitable explosive for tunneling, or other underground service, shou
duce the minimum of poisonous gases; should not be easily affected by m
nor by changes of temperature. Gelatin dynamite fulfills these requi
and is almost invariably used in American tunnels, although ammonia dy
was employed in some tunnels of the Los Angeles aqueduct (completed,

The proper strength of explosive for a particular case must be dete
by trial. Such widely divergent results are obtained in different localiti
the same grade of explosive and in apparently identical rock, that it is
to be dogmatic, even with accurate knowledge of local conditions. Gen
tough, close-grained, igneous rock requires a stronger dynamite, while
mentary rock (or an igneous rock that has been altered, weathered, c
tered) can be blasted as effectively with a lower grade of dynamite.

A notable example of the use of very high-grade explosives is that of the R
tunnel (4, 18), where for the tough, close-grained Pike's Peak granite, "100 %"
dynamite (blasting gelatin) was required to obtain satisfactory results. It is
to have been the first dynamite of this grade used in tunnel work. Table II
grades of dynamite used in a number of American tunnels.

Different grades in the same hole. The practice of loading the bot
a hole with high-grade dynamite, and using 40 or 60 % for the remainde
charge, is not uncommon, especially in the Western states. It prod
greater disruptive force at the bottom of the hole, where most needed.
practice entails the inconvenience of handling two different grades of dy
both in the heading, and in the thawing-house. In some cases equally g
sults might be secured by the use of lower grade dynamite in shallower
or a change in the type of cut. But for very hard, tough rock, it is a
practice and reduces somewhat the cost of explosive.

9. Quantity of Explosive

Data on the quantity of explosive used in American tunnels are g
Table IV. It will be seen that the general average consumption is 5.75
cu yd. The consumption of explosive in the Simplon and Loetschberg
ings was somewhat higher than this; at the Simplon the average charge
heading was 6.5 lb per cu yd, and at the Loetschberg the average from Ja
1908, to July, 1910, was 6.0 lb of 85 % dynamite per cu yd in the north
ing and 6.86 lb in the south heading (55).

10. Charging the Holes

Position of the primer. Opinions differ as to the proper position
primer (*i.e.* the cartridge containing the detonator). Some maintain
should be the outermost cartridge; others, that its proper place is at
the bottom of the hole. In our opinion the best practice is to place the
at top of charge, preferably in next to last cartridge, thereby protecting
nator during operation of tamping.

Advantages of inserting primer last. 1. The danger of igniting the dynam
side spitting of the fuse is eliminated; the dynamite thus being detonated and not

efficiency of the explosive is obtained and the resulting gases are not so injurious. ...ans that when the fire in the fuse travels past the full length of the charge, there is ...e that the flame may burst through the fuse covering and ignite the dynamite. 2. The ...e of dynamite is more likely to be well compacted, leaving no air spaces to decrease ...ness of the explosion. When the primer is the first or second cartridge in the ...e remaining cartridges (especially the one next to the primer) can not safely be ...roughly compacted.

...disadvantages of inserting primer last. 1. It is maintained that primer is liable ...ut off by the explosion of a neighboring hole. Possibly this might happen, but ...would be a strong indication that the hole was misplaced or too heavily loaded. ...fuse may be torn out by flying débris. The remedy is simple: before igniting, ...se carefully close up to the mouth of the hole. 3. To insure that the explosive ...reach the bottom of the hole, a stronger detonator, or a higher strength explo... ...ld be used. The latter remedy would have the effect of reducing the length ...e in the hole, since a smaller bulk of explosive would suffice.

...e. — It is sometimes desirable to prime bottom cartridge for rotation blasting; in ... cartridge must not be slit, and fuse must be of best quality (Sec 4, Art 9).

...se of tamping. Tamping is absolutely necessary, to develop the full ... explosive, to assure complete detonation, to minimize noxious fumes, ...reduce the flame. Unless thoroughly tamped, many explosives do not ...e completely; all give worse fumes, and in tunneling the cut holes should ...mped with special care, to prevent them from blowing out. Even water ... in down holes) is inferior to sand, and it is good practice to tamp ...ed holes with rock screenings.

...e tunnel men hold that any gain in efficiency of explosive is more than ...by the disadvantages of using tamping; as causing delay in charging and ...cience in cases of misfire. Notwithstanding this, the omission of tamp... ...ur practice, largely the result of prejudice. Much of the common over... ...of holes in tunneling could be obviated by proper tamping. In case ...re, never pick out tamping; drill and fire another hole at least 2 ft from ...e.

...rozen dynamite. Chilled or frozen dynamite should always be thawed ... use. This is necessary: 1, for the safety of the men; 2, because frozen ...ite can not be properly compacted in the hole; 3, because the full force ...en dynamite is not developed upon detonation, whence a loss in effective... (Sec 4, Art 7.)

...paration of primer. (See Sec 4, Arts 8 and 10.)

11. Firing

...e fuse. For a small round of holes the fuses are lighted by one man; for ...r number there should be two men, the best practice being for them to ...ultaneously the fuses of corresponding holes, each calling out the hole ...nts it.

... of the fuse projecting from the mouth of the hole should be closely coiled and ... split for ½ to ¾ in to expose the powder train. The lighting may be done ... ways: by a candle or acetylene lamp (spit of fuse may blow out flame and ... in the dark); better, by a "spitter," consisting of a short piece of fuse which ...shed and partly severed, at intervals of about ½ in, to expose the powder ... fire travels along the spitter, and on reaching one of the cuts flares violently ... side, and, if directed towards the blast fuse, is almost certain to ignite it prop... 4, Art 8). The firer should light each fuse separately, instead of bunching ... attempting to ignite all at once.

Fuse igniters. In very wet tunnels it is sometimes necessary to us igniter. One form consists of a short celluloid cylinder, closed at one containing gunpowder or other explosive. It is slipped like a cap on of the fuse, which is cut square. The celluloid itself is ignited by a c other means, and, being unaffected by dampness, burns strongly until it charge is reached. A flash then takes place, which lights the fuse. not very reliable, as the fuse wrapping may burn some time before pow is reached.

Electric firing. When ordinary electric detonators are employe necessary only to connect the fuse wires and pass a current through the current may be taken from an electric light or power circuit, or gene a hand-operated magneto or dynamo. All the holes are exploded neously. (For details, see Sec 4, Art 8.)

Delay-action detonators (Sec 4, Art 10) make it possible to explode consecutively, singly or in groups, with a single closure of the circuit resemble ordinary electric fuses, except that the platinum bridge does plode the mercury fulminate directly, but ignites a short train of gu inside the detonator, which requires an appreciable, though short, burning before it reaches the fulminate. By making the powder trai different lengths, two delays are obtained; or, if these be used in co with an ordinary detonator, the blasting may be performed in three sta

When using these devices in tunnel work, special detonators, called "No (Sec 4, Art 10), are generally placed in the cut holes, a "first delay" in the relie the "second delay" in the remaining holes. Delay-action detonators might made with other lengths of powder train. A fourth stage is needed for the li cause a part of their function is to project the material broken by the other h the immediate front of the new face; and, with the horizontal bar mounting, a is also desirable for delaying one lifter in order to throw the material away side of the tunnel where the capstan end of the drill bar is to be placed, thus space for manipulating the jack bar.

Advantages of electric firing. 1. Certainty of detonating all the cut h ultaneously. With fuse firing, when two cut holes are connected at the bott will explode as one; but it is impossible to make several pairs of holes, as in cut, explode together because of the variation in burning rate of the fuse, no ma exactly the lengths are proportioned. 2. Absence of smoke and noxious gase by burning of the fuse. Tests by Bureau of Mines show that a large part of t from burning fuse is carbon monoxide. 3. Charge may be ignited from a place o thus lessening dangers from premature explosions. 4. No time is lost in wa misfires.

Disadvantages of electric firing. Electric detonators cost more and trouble in keeping the leading and connecting wires in good condition, for they sarily damaged by every blast. When delay-action detonators are not used, must return to the heading after shooting the cut holes and connect the wires le the holes to be fired in each of the succeeding groups. As some time is required smoke to clear, and considerable shoveling may be necessary to uncover wire have been buried by the previous blast, it takes longer to fire a succession of h this manner.

12. Precautions

(See Sec 4, end of Art 7, as to precautions in handling, storing, th charging and firing.)

MUCKING

13. Men Employed

Size of the mucking crew. Time saved in mucking is especially important in the vertical-column drill mounting, because the set-up for a new round cannot be made until the heading is cleared. With the horizontal bar the mucking should be finished by the time the upper holes are down; if the shovel-crew is too small to accomplish this, setting up for the lower holes is delayed. Prompt removal of the muck is always essential, for many little things are to be done after the heading is clear, and reserve time is valuable. On the other hand, if the mucking crew be too large, the men will interfere with one another. Analysis of work in a number of tunnel headings shows that a shoveler requires about 5 ft width of floor space. That is, in a tunnel 10 ft wide there should be no more than 4 shovelers; in a 6-ft tunnel, only two. Besides these, a man or two should be employed for picking down the muck in front of the shovelers, handling large rocks, assisting in the handling of the cars, or doing other things to make for speed. Hence, in tunnels from 6 to 10 ft wide, the mucking crew should range from 3 to 6 men.

Alternate crews. In the Loetschberg and other European railroad tunnel headings, two sets of muckers were employed, one resting while the other was working. This tended to greater speed, because men can work harder for the few minutes required to load a car, if allowed an equal interval of rest. But, although this plan expedites mucking, the gain would hardly warrant such procedure where wages are high. At the Loetschberg tunnel the shovelers received $0.80 per day, as compared with $3.00 or $3.50 for like work in the western United States; hence the Loetschberg double crew of five shovelers would entail in this country an extra cost of $15.00 to $17.50 per shift, against $4.00 in Europe. Since the advance per shift in America rarely exceeds 7.5 ft, the extra cost of double-crew work would be at least $2.00 per foot of tunnel.

14. Positions of Working

The advantage of a rest from the severe work of shoveling can be obtained without employing extra labor, by shifting the men's positions systematically, as a number of American tunnels. In the Laramie-Poudre tunnel (13, 22), the muckers worked as follows:

When car 1 was filled, two shovelers, designated as A and B, took it to the rear, the two others (C and D) jumped to empty car 2, which had previously been thrown to side off the track, set it on the track and pushed it into position for loading. Meanwhile the remaining men (E and F) stopped picking down the rock pile, took the shovels of A and B and assisted C and D in filling car 2. Car 3 was then brought up by A and B close to where car 2 was being filled, and was thrown off the track on its side in the position formerly occupied by car 2. A and B then picked down the rock pile for the other men, while car 2 was being loaded. When filled, this car was removed by C and D while E and F set up the third car and loaded it with the help of A and B. A fourth empty car was meanwhile brought up by C and D, who then took their turn at picking down. The cycle was completed when E and F took the third loaded car to the front, brought back another empty and resumed their original position on the muck pile. Each man spent two-thirds of the time in tramming or picking down the rock pile, which is easier work than loading, and suffices to relieve the monotony of shovel-

In methodical procedure there is no confusion, no lost motion. The foreman rarely has to give an order, and does not shout at the men constantly, a bad habit unfortunately common in many tunnels; nor do the men waste time by running to the foreman for advice in minor matters.

Rapid work is attainable by the system described above. Cars of 1 capacity at the Laramie-Poudre tunnel were filled in an average of 3 or At the Rawley tunnel (51), where a similar system was used with four sh 25 cars of 17 cu ft each were loaded in 2 hr; in another case, 20 cars in including all ordinary delays incident to making up trains. Taking in sideration the number of shovelers, these figures of 5 to 6.75 min per c the Laramie-Poudre, and 7.5 to 8 min at the Rawley, compare favorab work in the Loetschberg tunnel, where 5 min were required to fill a 1-c (35.5 cu ft) by a crew of 10 men, with an extra minute to replace it, wh with an empty one.

15. Handling Cars in the Heading

A common system is to tram the cars by hand from the face to a where they are made up into trains and hauled to the portal by the m transport provided. The disadvantages of this plan are: 1, the swit siding must be moved frequently, to keep pace with the tunnel advan they will soon be too far from the face; 2, an unavoidable loss of time the loaded cars are being removed and the empties brought up, beca switch can not well be nearer than 100 ft from the face, and it is often 500 ft; 3, the loaded car must be moved by hand to the siding before the can pass it.

Loading on alternate tracks has some disadvantages. The extra l laying two tracks in the heading, instead of one, which can not be don mucking is finished, may delay starting a new round. Also, the econom for keeping the switch near the face is not so apparent with this method the other, so that shifting the tracks is apt to be neglected, and time wa the trammers. Loading on alternate tracks only partially obviates the of derailments in crossing switches.

Steel plates. The necessity for two tracks can be avoided by cover floor of the heading with steel plates for 30 or 40 ft back from the face cars can easily be jumped from the end of the track on to the first of these and pushed up to the muck pile. When one car is full, an empty one can be shunted into position for loading while the full car is being moved t the siding. This simple method is especially advantageous for larg The steel plates also facilitate mucking, since the broken material is sh from them with less labor than from the rough and irregular rock floor tunnel.

The best method, in most cases, is to use small cars; then the empties tipped off the track to allow full cars to pass, and can be righted and pu by two men, thus avoiding the complications and extra work arising fr use of switch and siding. The details of this system were well worked the Rawley tunnel.

On reaching the heading, the empties were pulled as near as possible to the f which ordinarily stood on the track some 75 or 100 ft from the face. The mule w detached from the empty train and pulled the full cars back to the one being usually the last one of the previous empty train; or, if all had been loaded, the f were pulled as near the face as possible. The empties were then hauled up to the cars and tipped off the track. All this was done by the mule and driver alone, when the shovelers had completed loading and could therefore assist. After the train had started for the portal, the muckers took the two empty cars nearest the set them on the track and pushed them to the face, where one car was again tip on its side while the other was being loaded. Getting the loaded train out of the h and an empty car in position again rarely occupied more than 3 to 5 min. The rem of the cycle was similar to that in the Laramie-Poudre tunnel, already described (A

The advantages of this system are: 1. It does away with the trouble arising from having a switch near the heading. 2. It requires the use of small cars which leave more room for the shovelers. As the sides of the smaller cars are lower, they are easier to load, and to handle in case of derailment; and, as fewer men are required for moving them, more of the muckers' time is spent in actual loading. 3. The cars have to be hand-trammed only a little more than the length of one train, this distance being constant and not varying as the tunnel advance. 4. The minimum time is consumed in replacing a loaded car by an empty. 5. Time is saved in making up trains.

TIMBERING

16. Materials

Artificial supports of timber, brick, stone, metal, or concrete may be used in tunnels. Timber is in general use, because of its cheapness in many mining districts, and the readiness with which it is cut, framed, and placed in position. Even when the permanent lining is to be of masonry or concrete, temporary timber supports are usually put in first, as the work advances.

Seasoned timber is desirable, because it better resists decay. The bark should always be removed from round logs; the space between it and the wood forms a breeding place for wood-destroying insects, while the bark itself collects and retains moisture, thus encouraging the growth of fungi and hastening decay. *Round timbers, peeled and seasoned*, are more durable than sawed timber because the sharp corners of the latter are especially liable to decay. In round and small timber, commonly used for mining, the outer half of the log is almost entirely sap-wood, containing starch, sugars, proteids, and soluble nitrogen compounds, which are the foods upon which decay-producing fungi live; they are nearly absent in the heart-wood (Sec 43, Art 20).

The corners of a square timber consist largely of this easily infected sap-wood. In timbers the corners of sawed timbers 3 or 4 years old are often so decayed that they can readily be pried off to the heart-wood. Though the outer portion of a round timber consists of sap-wood, the cells have not been ruptured by the saw and the fungi do not readily find lodgment. Round timbers, however, are not so easily handled as square; they can not be so readily alined, nor reinforced by false sets or pieces. Where timber is transported long distances, the greater weight of round sticks (especially when they are truncated cones or "churn shaped," instead of being nearly cylindrical) cause greater freight costs, and square timbers should be used. Under such conditions the saving in transportation charges will pay for a preservative treatment, applied to the timbers before being placed underground.

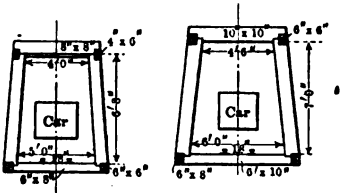
Preservatives. The best method of checking the growth of fungi is to poison the source of their food supply. Most of the preservative processes depend on the injection of zinc chloride or creosote. The former is not suitable for wet places, as it is soon leached out of the timber. Creosote, properly applied, will last much out, and is therefore the preservative generally employed for tunnel timbers. It is somewhat more costly than zinc chloride and, being a liquid, the transportation charges also are higher (Sec 43, Art 20).

For **permanent tunnel linings** brick and stone were formerly the chief materials used, and most of the older railroad tunnels are lined in this expensive manner. Steel beams are sometimes employed as roof supports in the main haulageways and gangways of coal mines, and their use is likely to extend as timber becomes more expensive. Concrete has thus far been restricted chiefly to tunnels for irrigation, or water-supply tunnels, but it will probably be applied in future to important mining tunnels, where a permanent lining is necessary (Secs 44, 46, 57, 59).

17. Types of Timbering

The simplest form of roof support is a horizontal timber, set at each a "hitch" or recess in the wall. When the sides of a tunnel are not enough to afford reliable hitches for this timber, "posts" are placed un ends, and it is then called a "cap." If the floor is weak the posts res sill, thus forming a *four-piece set*. When the roof pressure is too grea carried by a horizontal cap, the upper member is divided into two o segments, forming the *rafter or arch set*.

Four-piece sets. Fig 9 shows a four-piece set of timbers, comprisi posts, and sill, for as small a tunnel as is usually advisable. The main t are 8 by 8 in, and the posts are held apart and side pressure resisted by 8-in plank spiked to the cap. Horizontal collar and sill-braces are also



Figs 9 and 10. Four-piece Sets

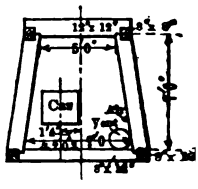
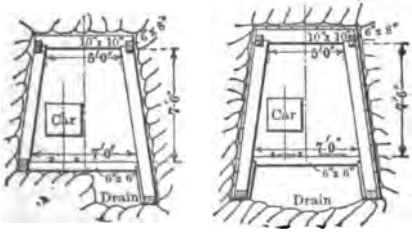


Fig 11. Set for Haulage and M

Fig 10 gives the dimensions for a similar set in a larger tunnel. The t are 10 by 10 in and the joint between cap and posts is beveled at the c for resisting both horizontal or vertical pressure. For heavier ground th and cap may be made of 12 by 12-in timber, with 8 by 8-in collar-brace single-track tunnels for a considerable traffic, it is often advisable to ma net cross-section large enough for a manway and ventilating pipe on on and car tracks on the other (Fig 11). For carrying a considerable volu



Figs 12 and 13. Sets for Drainage Tunnels

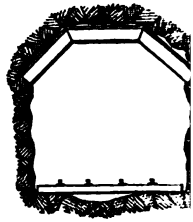
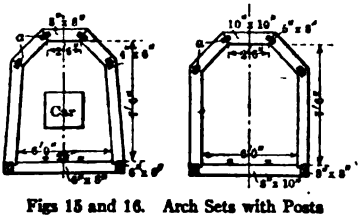


Fig 14. Arch Set without

water, the design shown in Fig 12 is satisfactory, notwithstanding the fac the timbers are unsymmetrical. The 6 by 6-in sill, which also forms th tie, is not notched into the post at the right side, but is supported by a 10-in plank spiked to the post, the upper end of the plank being recessed to receive the sill. For still larger volumes of water, the design in Fig recommended. The quantity of water and grade of tunnel determine the p depth of the drain.

Arch sets. Sometimes the tunnel walls are firm and only the roof needs port (Fig 14). A carefully constructed footing for the arch timbers is requ

with hand pneumatic drills the cost of cutting hitches is repaid many times ...ing in timbers and the smaller amount of rock to be excavated. Fig 15 ... a complete arch set for a small ... The main timbers are 8-in ... collar-braces (*a*) 4 by 6 in. ... of placing the braces near ... ter edges of the timbers (as ... quare sets, where they can be ... in from the outside), they are ... ed into the face of the timbers, ... wn. The beveled arch pieces, ... while being erected and ... are more difficult to hold in ... man square sets, are prevented by the braces from slipping. A similar ... set with vertical posts, is shown by Fig 16. This increases the width



Figs 15 and 16. Arch Sets with Posts

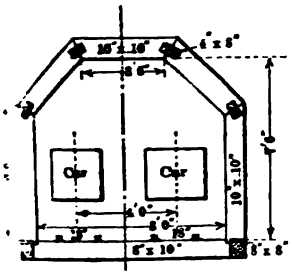


Fig 17. Double-track Arch Set

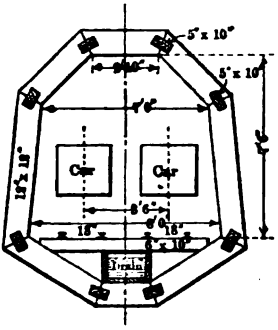


Fig 18. Inverted Arch Set for Swelling Ground

...e tunnel at the shoulders, and calls for 30° bevel joints throughout, thus ...g the side and top pieces of the arch interchangeable. Fig 17 illustrates similar timbering for a double-track tunnel. Some constituents of certain rocks oxidize on exposure to the air, softening and increasing in volume to such an extent that the excavation tends to close in from sides, roof, and even the floor. For driving through such swelling or creeping ground, the sets may be designed as in Fig 18, where drain and track are protected by an inverted arch. If 12 by 12-in timbers of this form will not resist the squeeze at the usual distance of 4 ft between centers, the sets must be spaced closer. Where zones of rock are encountered that can not be held even by this expedient, the sets may be placed 3 or 4 in apart, the decomposed rock in line with the spaces between the posts being picked out to a depth of 4 or 5 in back ...e timbers. Swelling ground is usually soft, so that this can readily be done. ...g the tunnel subsequently with concrete, access of air to the rock is pre... thus permanently overcoming the difficulty. The octagonal set (Fig 19)

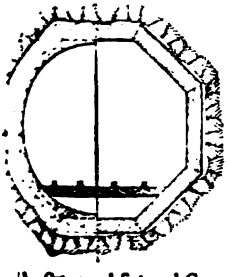


Fig 19. Octagonal Set and Concrete Lining

offers another means of supporting very heavy ground. For importan it is advisable to follow up the timbering with concrete, placed in front between the sets, as shown in the cut. In this case, to insure the sa the tunnel after the timbers have decayed, the sets should be spaced r than 12 in apart (15 to 18 in is still better), because the wider opening room for a stronger rib of concrete between the sets. The octagonal

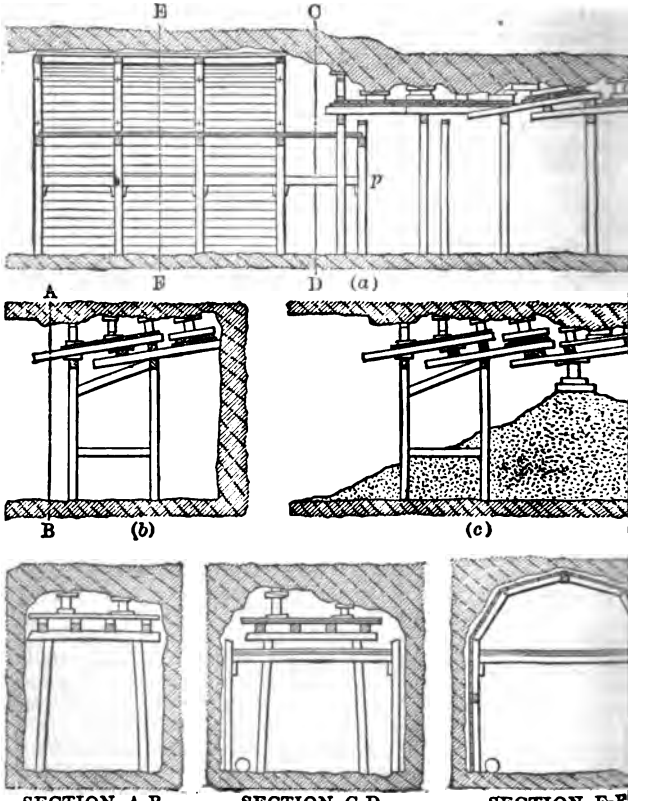


Fig 20. Elizabeth Lake Tunnel Timbering (*Mines and Minerals*)

easy to handle and where the flow of water is not too great for the dra area below the tracks, nothing better can be adopted. All the pieces in set are alike, which is a convenience in framing, storing, and erecting the

Speed of work. The system of timbering employed in the north he of the Elizabeth Lake tunnel (Fig 20) affords a good example of speed efficiency in placing timber sets. In this end of the tunnel, of which 13 were timbered, the average monthly advance was 340 ft, as compared

for the south heading, where only 3 424 ft of timbering were required in ... 13 500 ft (37).

... main tunnel (12 by 12 ft) was preceded by a short pilot heading, about 8 by 8 ft, ... of which was supported by "false" timbering. At the end of a shift, with the ... removed from the heading preparatory to blasting, the position of the timbers ... be somewhat like that shown in Fig 20, which represents a short timbered section ... full-size tunnel, also the horizontal timbers used temporarily for the pilot heading. ... pieces of the permanent sets were 8 by 8 in by 8 ft 6 in long. In the cut the sets are ... at 8-ft intervals; actually, they were often spaced irregularly, depending upon the ... of the roof. The width between posts was 10 ft 2 in in the clear, the rock section ... as nearly as possible 12 ft wide. The collar braces were usually 2 by 6-in planks, ... ends being supported by wedges or timber-ends spiked to the set. The false posts ... pilot heading were later used as permanent posts for the full-size tunnel; the bev... ends of these, intended to carry the arch pieces, were stood on the heading floor, ... their square ends supported the temporary caps, which likewise were already ... for use as permanent posts. Thus, there was no handling of timbers not intended ... regular sets. Just before blasting the false timbers were braced and wedged ..., to the roof. (Fig 20, *b*.)

... mediately after blasting, the new roof was shored up temporarily from the top of ... broken rock, by two nearly horizontal timbers, placed close to the side walls, and ... transverse timbers and blocking resting upon them (Fig 20, *c*). This was done by ... timbermen, and interfered but little with the work of mucking back from the face ... atory to mounting the drills. The débris was next removed down to solid bottom ... permit the setting of false posts which, when capped, carried the weight of the roof. ... bering the tunnel during enlargement to full size was readily done. New perman... posts (Fig 20, *a*) were erected as fast as the section was widened down the side ... Transverse spreader timbers (Fig 20, Sec *CD*) were then wedged between the ... with their undersides 14 in below the joint and resting upon timber-ends spiked ... posts. Across the spreaders were placed two layers of 2-in plank, forming a floor ... deck. This floor was 2 to 3 in below the caps resting on the false posts, so that it ... easily laid while the false sets continued to hold the roof. Working from the end of ... the wedges and blocking transmitting the roof weight to the false sets were next ... out and the loose roof rock picked down on the floor, from which it was later ... into cars. By placing the permanent posts a foot or so in advance of the false ... as in Fig 20, *a*, the arch timbers could be erected as soon as the shattered roof was ... trimmed. Lagging and wedging quickly followed, so that the roof was sup... by the permanent sets shortly after the removal of the false blocking.

18. Concrete Linings

... lining a mine tunnel the concrete is usually placed in the spaces between ... timbers and for a few inches in front of them. Where the sets are not ... too closely this is generally sufficient, even though decay of the wood ... causes weak spots in the lining. This defect can be avoided by the use of ... and caps of reinforced concrete instead of wood, a practice which finds ... wherever its greater expense is warranted. The concrete posts and caps ... made outside of the tunnel, in a mold which gives them the exact form of ... pieces, and by proper reinforcement they can be made equal, if not ..., in strength to timber. In water-supply tunnels a continuous con... ... lining performs the added function of preventing eddies and friction caused ... otherwise irregular walls, for which reason such tunnels are generally ... throughout, irrespective of the need of the roof for support. In the ... Angeles aqueduct the lining is at least 8 in thick where the tunnel is not ..., although an occasional rock projecting into the concrete space was ... removed unless it came within 4 in of the inside finished surface of the ... In timbered ground the concrete was placed between the timbers and ... minimum distance of 4 in in front of them. The inverted siphons of the ... aqueduct are lined with concrete, ordinarily 2 ft thick, though solid

rock was permitted to project to within 10 in of the interior surface. C to the great hydrostatic head, reaching 700 ft, to which these linings are subjected, all timbering was removed before the concrete was placed; necessary, especially designed steel supports carried the roof while the crete was setting. In the Snake Creek tunnel a zone of swelling groun encountered which resisted all efforts to hold it in the ordinary way, the st est timbers obtainable being crushed in less than a month's time. A co lining reinforced with steel rails (Fig 21) was therefore introduced (46).

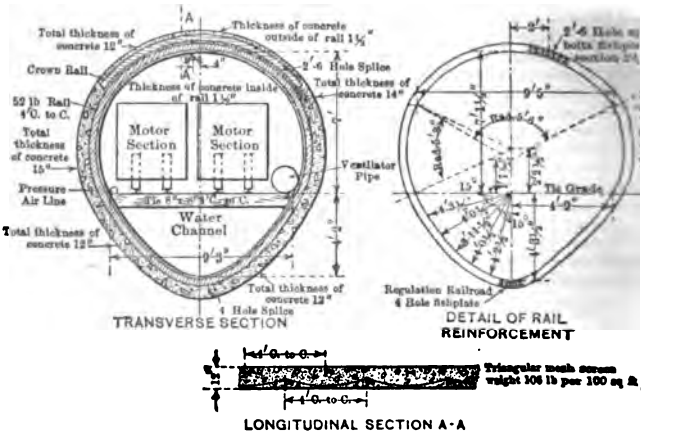

Total thickness of concrete 12"
Thickness of concrete outside of rail 1½"
Crown Rail
2'-6 Hole Splice
Thickness of concrete inside of rail 1½"
Total thickness of concrete 14"
52 lb Rail 4'0. to C.
Motor Section
Motor Section
Total thickness of concrete 15"
Ventilator Pipe
Pressure Air Line
Water Channel
Total thickness of concrete 12"
Total thickness of concrete 12"
4 Hole Splice
TRANSVERSE SECTION
Regulation Railroad 4 Hole fishplate
DETAIL OF RAIL REINFORCEMENT
4'0 to C.
Triangular mesh cement weight 108 lb per 100 sq ft
4'0 to C.
LONGITUDINAL SECTION A-A

Fig 21. Reinforced Concrete Lining, Snake Creek Tunnel (*Eng Rec*)

19. Driving Through Loose or Running Ground

Materials requiring special methods of support may be classified as follow

A. Rock which will probably stand, but under which the miners can work in safety, while ordinary timber sets and lagging are being erected.

B. Rock or other material which is certain to crumble and cause light unless promptly supported.

C. Soft material which requires immediate support and brings heavy p sure on lagging and timbers.

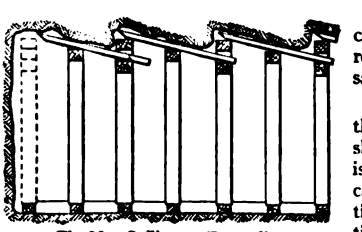


Fig 22. Spiling or Forepoling

D. "Running ground": from consistency of broken, muddy, slipp rock, down to semi-fluid mud, sil sand.

Class A. Headings driven un these conditions can be timbered shown in Fig 22. Each alternate p is slightly shorter than the others carries a double cap. This system timbers and driven lagging is sot times used also for material of Class but since each length of lagging cov two spaces and three caps, it is not recommended when driving under hea roof pressure. It is used primarily to protect the workmen, affording a ne

controlled and effective support, at the minimum expenditure of material and

—The terms "lagging," "forepoling" and "spiling," are used interchangeably about the United States, but "forepoling" is the usual designation abroad.

Class B. In this case the tail-block system shown in Fig 23 is satisfactory. If in common use, in nearly every gang of men will be some who have experience with it. A disadvantage is that a slight pressure on the forward end of the lagging pieces creates much friction between the tail block and the

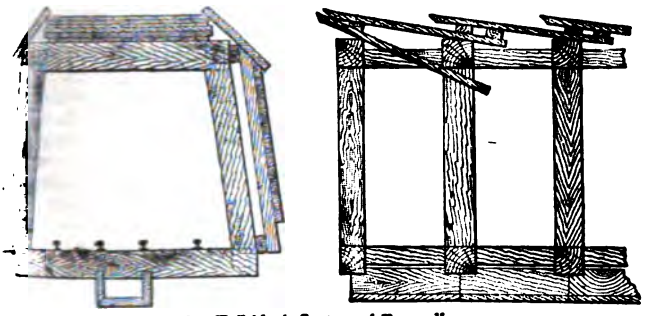


Fig 23. Tail-block System of Forepoling

so that driving the lagging may become difficult; but, when the ground is heavy and only light falls occur, this system is unequalled for cheapness and simplicity. In going through coarse, loose rock, where the jarring effect driving the spiling forward with a sledge increases the tendency to fall and the lagging may be forced forward by a screw jack, which, while slower than the hammer, is less apt to bring down loose material.

Removable tail-block. In many cases the removable tail-block system (Fig 24) possesses advantages over the preceding. When the spiling pieces are driven home, the wedges above the tail blocks are knocked out and the spiling allowed to rest on the auxiliary cap; the tail block is then removed and kept for use on the next set.

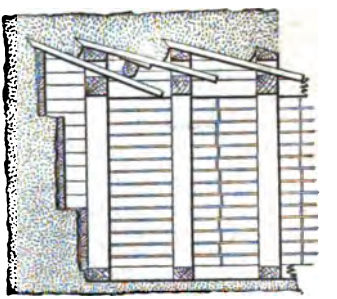


Fig 24. Forepoling with Removable Tail-block

Class C. For this kind of material a proper system of support becomes necessary. The commonest devices are the stationary and the swinging false sets.

The **stationary false set** consists of a set of timbers (Fig 25, *A*) placed midway between the permanent sets, to cover the distance through which the unsupported spiling has to carry the load. Where spiling is required for only a few sets, this system works well; but it possesses many disadvantages, and when a considerable distance must be driven through class C material, the swinging false set is preferable.

The swinging false set is shown in Fig 26, in connection with a thre arch cap for very heavy ground; it can be applied with even greater fac

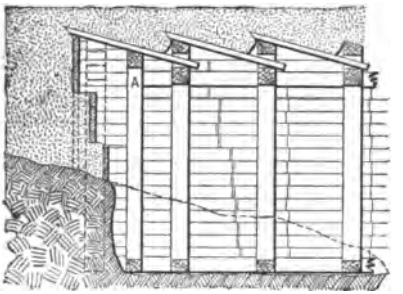


Fig 25. Forepoling with Stationary False Set

the ordinary horizonta There are no tail bloc does the spiling have driven across three sets bers, as in Fig 22. The coming on the front e the spiling is carried dire the swinging false set, a spiling can be driven wi less hammering than is re with the tail blocks. A cated in the longitudin tion, the posts of the sw false set rest and rotate sill of the permanent se when first erected occup position shown by the dotted lines. They carry a cap of heavy steel p which supports the front end of the spiles *a*. While driving, the only pr to be sustained is that of the rock immediately above and in front of *A;* wh with the tail-block system, a few pounds weight on the front end of a brings 4 or 5 times as much weight on its supports. As the spiling is d

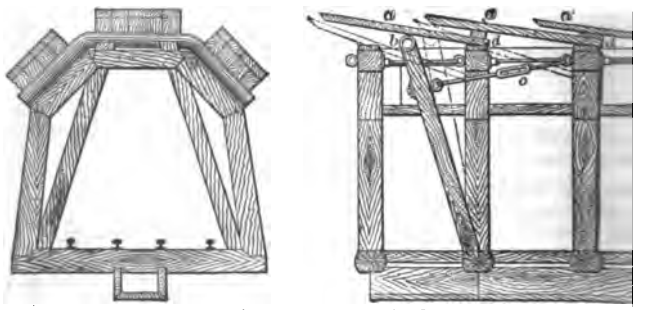


Fig 26. Swinging False Set

the turnbuckle *c* is slowly unscrewed, allowing the false set to fall forw until the spiles are nearly horizontal. When all the spiles have been d home, and the supporting block *d* placed under them, the turnbuckle is sla still farther, until the swinging set loosens. Then the hanging rods are unho from the eye-bolts and the false set is advanced to its next forward positi

In addition to the swinging sets and hangers, this system requires that the tin for at least 5 or 6 sets from the face shall be connected by tie rods, in much the manner as hanging bolts are used in shaft timbering. This, however, is an advan because, by screwing the timbers up tightly against the braces, they can be block position more easily and quickly. Also, the timbers are held in place so rigidly if hard ground is encountered in any part of the face, heavier charges of explosive safely be used than if the timbers were held in place only by blocks and wedges.

Class D. For driving any considerable distance through the difficult cla material, the shield and air-lock give satisfactory results as to speed, sa

economy of operation, in connection with railroad tunnels; but the high cost is rarely, if ever, justified for the narrow zones and small areas of mixed rock, or other bad ground, generally encountered in mining tunnels.

When the ground will stand for a short time in a nearly vertical face, the manner of driving lagging does not differ materially from that suited to classes B and C, except that the ground must be carefully and closely lagged, and the small corners between the vertical and horizontal lagging entirely covered by breast boards or "lacing," as shown in the cross-section, Fig 23.

Running ground. Plastic or semi-fluid material is difficult to hold back except with breast boards across the face (Fig 24). The head boards must be carefully blocked and advanced from the top downwards, never in the reverse direction. To prevent runs of thin mud from between the lagging or lacing boards, tough, wiry hay may be used as calking, a bale of it being kept at hand near the face. Working with breast boards in the face is a slow and difficult operation. As there is usually pressure from above as well as from the face, great care is necessary to prevent the boards from being forced inward, admitting an inrush of soft material. It is often necessary to tie the breast boards to one another and to the posts with pieces of inch boards and wire nails. For this kind of work, experience, skill, and constant care are demanded.

Boulders or rock in class D material. The difficulties of driving through ground designated as class D are greatly increased when boulders or masses of solid rock extend part way across the heading (Fig 25). The soft ground must be thoroughly lagged and breast-boarded from 6 in to 1 ft beyond the space necessary to erect the timber set, and the rock projecting into the tunnel is carefully broken with light charges so as not to disturb the breast boards or lagging.

When much water is encountered in a firm rock heading, it merely causes discomfort and inconvenience to the men. But, if heavy inflows of water occur in soft rock or running ground, quantities of sand or silt may be brought in, and if the lagging and breast-boarding are driven so close that the water is entirely shut off completely, the resultant pressure may become dangerous in the confined condition of the work. Sometimes the volume of water is so great that even in fairly hard rock, anything like a reasonable rate of progress is impossible. In such cases, it is usually cheaper, safer, and quicker to run a small lateral to tap the water and drain the face, thus facilitating the work of miners. When large volumes of water continue to enter the heading as it is advanced, it may be best to carry a lateral parallel to the main tunnel and keep its face somewhat in advance of the main heading, thus draining the water through a smaller opening, the alinement of which needs not be maintained with exactness. By this means the main heading is generally relieved of both the volume of water and the pressure accompanying it, and the cost of driving is reduced.

COSTS OF TUNNELING

The following data, showing the cost of driving a number of tunnels under different conditions, are as complete and accurate as could be secured. These data were in all cases obtained from persons in charge of the work, or who were in a position to know the actual costs, and may be taken as authentic. Accompanying the cost figures are schedules of the wages paid the different classes of labor and data as to the speed of advance.

Laramie-Poudre Tunnel (9, 13, 22, 27) 11 306 ft

	Cost per ft		
Superintendence and foreman	$1.50	Picks, shovels, and steel	
Drilling	4.47	Explosives	
Mucking and loading	4.92	Lamps and candles	
Tramming and dumping	4.63	Oil and waste	
Track and pipe	0.47	Blacksmith supplies	
Power house	0.35	Liability insurance	
Blacksmithing	0.84	Office supplies, telephone, and bookkeeping	
Repairs	0.47		
Bonus to workmen	1.75		
Maintenance, of buildings, and fuel	0.62	Permanent equipment (less approximately 10% average)	
Machinery repairs	0.12	Total	
Air drills and parts	1.33		

Permanent equipment included power plant, camp buildings and furnishings rails, cars, etc.

Wages: Drillers, $4.50; helpers, $4.00; muckers, $3.50; blacksmiths, $5.00; $4.50; dumpmen, $3.50.

Maximum progress in any calendar month: 653 ft, March, 1911. Average progress: 509 ft (for the 16 months when finished plant operated).

Special features: Inaccessibility; the tunnel was located about 60 miles fr nearest railroad siding, and the roads were mountainous and very steep in places

Elizabeth Lake Tunnel (3, 14, 16). In the following table the first states the cost per foot for driving the north heading 13 370 ft through granite, requiring 13 031 ft of timbering; the second column relates to the heading, 13 500 ft through medium to hard granite, requiring only 3 42 timbering:

	North heading	So hea
Drilling and blasting	$11.25	$14
Mucking and tramming	11.70	11
Engineering and superintendence	1.27	0
Drainage	0.45	0
Ventilation	0.22	0
Light and power	5.55	4
Timbering	8.48	2
Cost of shaft at sta 30 + 16	0.93	.....
Permanent equipment (full charge, no salvage — estimated)	3.70	3
	$43.55	$38

Wages: Drillers and helpers, $3.00; muckers, $2.50; blacksmiths, $4.00; h $2.50; motormen, $2.75; dumpmen, $2.50.

Maximum progress in any calendar month: 604 ft, April, 1910. Average m progress per heading: 350 ft.

Minion Tunnel, South Portal, May, 1909 to Sept, 1911, 5 515 ft

	Cost per ft		Cost per ft
[illegible]stration*	$1.14	Track and pipe	$1.22
[illegible]	9.20	Miscellaneous supplies†	2.46
[illegible]	2.12	Drill parts (including steel)	1.02
[illegible]	1.97	Bonus	0.48
[illegible]ering (563 ft)	0.30		$19.91

*Includes superintendence, office supplies, and general charges.
†Includes candles, electric-light bulbs, shovels, picks, blacksmith's supplies and fuel, [illegible] mechanics' supplies.
‡Rates: Drillers, $3.50; helpers, $3.00; muckers, $2.75; blacksmiths, $4.00; helpers, [illegible] motormen, $2.75; dumpmen, $2.50; power engineers, $2.75.
§Maximum progress in any calendar month: 414 ft, Feb, 1911. Average monthly [illegible] 110 ft.

Newhouse Tunnel (19, 20, 50)

	Jan to Aug, 1909, 2 233 ft	Sept to Dec, 1909, 1 098 ft	April to Aug, 1910, 693 ft
[illegible]	$6.72	$6.98	$11.73
[illegible]ives	4.15	3.52	4.57
[illegible] and caps	0.39	0.36	0.44
[illegible]moval of broken rock	1.49	1.47	2.22
[illegible]	1.99	2.16	2.82
[illegible]timbing	1.57	2.61	2.00
[illegible] drills, repairs, and steel	1.50	2.74	2.86
[illegible]ment, ties, rails, pipe, etc.	1.74	1.78	2.19
[illegible]	0.79	0.80	1.85
	$20.34	$22.42	$30.68

*Rates: Drillers, $4.00 to $4.50; helpers, $3.25 to $4.00; muckers, $3.50; motormen, [illegible] dumpmen, $3.00; blacksmiths, $3.50 to $4.50; helpers, $3.00.

Stilwell Tunnel

Year—	Footage	Cost per ft
1900–01	12	$23.88
1901–02	490	22.98
1902–03	377	27.94
1903–04	702	21.69
1904–05	1 077	21.19
1905–06	292	30.37
Average for	2 950 ft	$23.38

*These costs include all labor, supplies, repairs, powder, fuse, caps, candles, tools, [illegible] and general expenses; also total value of electric-drill plant with which tunnel [illegible]ed and total value of air-drill plant which succeeded it, together with buildings, [illegible] and ventilator, with no credit for salvage on any permanent equipment. The [illegible] dated from Sept 30. The tunnel was driven in 1901–03 with electric drills; [illegible] cost for 1905–06 was due to station cutting, where the tunnel was double size.
†Rates: Drillers, $4.50; helpers, $4.00; muckers and trammers, $3.50; blacksmiths, $4.50.
‡Maximum progress in any calendar month: 170 ft, Aug, 1904. Average monthly [illegible] 130 ft (last 10 months).

Rawley Tunnel, 6 235 ft (59)

	Cost per ft		Co
Drilling and firing	$5.25	General expenses	$1
Mucking	2.16	Permanent plant	3
Tramming	1.13	Timbering (1 618 ft)	1
Track and pipe	0.44	Boarding house, debit balance	0
Miscellaneous underground expenses	1.44	Total	$20
Power plant	2.50	Credit salvage on permanent plant	1
Blacksmithing	0.73	Net cost	$19
Miscellaneous surface work	0.83		

Drilling and firing include labor, powder, fuse, caps, supplies, and repairs. M tramming, and track and pipe include labor and supplies. Miscellaneous under expenses include wages of foremen, underground telephone, etc. Power plant i labor, supplies, and fuel. Blacksmithing and miscellaneous surface work includ and supplies. General expenses include salaries, office supplies, telephone, etc. nent plant includes machinery and buildings, with labor of installation, steel rail manent supplies, and repairs. Timbering includes labor and supplies.

The salvage on the permanent equipment was approximately 50% on salable a such as machinery, rails, cars, etc.

Wages: Drillers, $4.50; helpers, $3.75; muckers, $3.50; blacksmiths, $4.50; d $4.50; power engineers, $4.00.

Maximum progress in any calendar month: 585 ft, July, 1912. Average m progress: approximately 350 ft.

Roosevelt Tunnel (14, 18, 20)

Total cost of portal heading		$111 980.06
Contractor's percentage		11 404.88
Cost of shaft headings		262 126.55
Total cost		$385 511.49
Distance driven	14 167 ft	
Average cost per ft		$27.27

Portal Heading

Month	Foot-age	Cost per ft	Month	Foot-age	C p
Feb. 1 to Mar. 1908	284	$22.880	Dec.	334	$18
Apr.	302	20.470	Jan. 1909	435	16
May	398	19.780	Feb.	290	22
June	485	20.212	Mar.	340	21
July	585	18.882	Apr.	316	21
Aug.	480	19.250	May	402	18
Sept.	360	21.880	June 8 days	62	40
Oct.	367	23.880	Average for	4 911 ft	$21
Nov.	380	21.30			

SHAFT HEADINGS

Month	Footage	Cost per ft	Month	Footage	Cost per ft
[illegible] 1908 (two headings)	49	$105.520	Dec, (one heading)	251	$35.153
"	141	44.380	Jan, 1910 "	282	28.820
"	177	40.110	Feb "	259	30.636
[illegible] 1909 "	261	24.060	Mar "	344	27.620
"	601	23.700	Apr "	376	25.313
"	639	26.256	May "	393	24.856
"	670	25.020	June "	373	26.616
"	552	28.340	July "	350	25.247
"	498	27.375	Aug "	372	25.029
" (one heading)	319	32.871	Sept "	342	28.450
"	410	27.747	Oct "	372	27.361
"	355	32.400	Nov "	192	27.786
"	380	28.178	Average for.....	9256 ft	$29.10*
"	298	34.200			

*Omitting Oct, 1908.

TYPICAL DISTRIBUTION OF EXPENSES PER FOOT

	Portal heading, July, 1908, 203 ft	Shaft heading, Feb, 1910, 259 ft		Portal heading, July, 1908, 203 ft	Shaft heading, Feb, 1910, 259 ft
[illegible]enance of buildings, [illegible] etc	.....	$0.096	General expense.....	$0.16	$0.342
[illegible]ery and repairs...	$0.61	1.158	Lumber, ties, and wedges	0.01	0.119
[illegible]ls and parts.....	0.99	1.930	Horses and feed.....	0.01	0.324
[illegible] picks and steel..	1.90	1.930	Drillers and helpers.....	4.21	4.050
[illegible] fittings.....	.....	0.193	Muckers	4.11	3.065
[illegible]en	1.09	1.480	Blacksmiths and helpers.	3.43	1.362
[illegible]ives	6.90	5.032	Engineers.....	1.79	1.300
[illegible]e and candles.....	0.36	0.217	Pipe and track men.....	.....	0.675
[illegible] waste.....	0.09	0.252	Drivers and dump men .	(a)	2.355
[illegible] power.....	2.06	2.440	Foremen.....	1.50	1.753
[illegible]th supplies.....	0.09	0.150	Mine telephone.....	.....	0.008
[illegible] insurance.....	0.17	0.213	Bookkeeper.....	0.12	0.193
			Totals.....	$29.60	$30.637

[illegible]ided with muckers.

[illegible]. Drillers, $5.00; helpers, $4.00; muckers, $3.50; power engineer, $4.00; [illegible]s, $5.00; helpers, $3.50; dumpmen, $3.50; drivers, inside, $5.00; outside, $4.00. [illegible]mum progress in any calendar month: 435 ft, portal heading, Jan, 1909. Aver[illegible]nthly progress: Portal heading, 300 ft; shaft headings, 270 ft; all headings, 285 ft.

Strawberry Tunnel (44, 59)

	Footage	Cost per ft
[illegible]eading, previous to 1909.....	1 613	$60.05
" during 1909.....	3 892	33.58
" " 1910.....	5 021	30.56
" " 1911.....	3 491	41.52
" Jan to July, 1912.....	2 382	36.79
" Oct, 1911, to July, 1912.....	2 682	33.04
[illegible]verage for.....	19 081 ft	$36.78

[illegible] Drillers, $3.50; helpers, $3.25; muckers, $2.75; motormen, $3.25; brake[illegible]; blacksmiths, $4.00; helpers, $2.75. Maximum progress in any calendar month: [illegible] Nov, 1910. Average monthly progress: 320 ft per heading.

Strawberry Tunnel (44, 59) (*Continued*)
DETAILED COSTS PER FOOT

	West heading, 1909, 3 892 ft	West heading, 1910, 5 021 ft	West heading, 1911, 3 419 ft	West heading, Jan–July, 1912, 2 382 ft
Labor:				
Engineering	$ 0.49	$0.61	$0.45	$0.36
Superintendence	0.73	0.60	0.82	0.56
Shift bosses	1.22	1.25	1.65	1.08
Time keepers	0.36	0.22	0.38	0.26
Drillmen and helpers	3.15	2.85	4.07	3.08
Miners (for handwork, trimming, etc)	0.25	0.28	0.37	0.43
Muckers	2.96	2.93	5.13	4.95
Track and dumpmen	0.74	0.71	2.00	1.55
Mule drivers	0.39	.....	.....	.....
Motormen and brakemen	0.44	1.49	1.87	1.33
Electricians and blowermen	0.07	0.13	0.08	0.18
Disabled employees	0.19	0.16	0.48	0.48
Timber men	0.22	0.28	1.72	2.59
Miscellaneous	0.40	0.07	0.05	.....
	$11.61	$11.58	$19.07	$16.85
Materials:				
Powder, fuse, caps, etc	3.08	3.52	2.61	2.72
Lumber	0.29	0.22	0.80	2.13
Oils, candles, etc	0.22	0.20	0.43	0.32
Ventilating pipe	0.64	0.65	0.77	0.70
Track, including ties	0.68	0.74	1.52	1.51
Compressed-air pipe	0.40	0.28	0.36	0.30
Drill repair parts (including hose)	0.18	0.24	0.34	0.32
Miscellaneous	0.19	0.07	0.25	0.39
	$5.68	$5.92	$7.08	$8.39
Repairs:				
Machine shop expense (including labor and supplies)	0.93	0.90	2.16	1.39
Blacksmith shop expense (including labor and supplies)	1.22	1.23	1.54	1.02
	$2.15	$2.13	$3.70	$2.41
Power (all purposes)	7.65	5.70	5.20	3.75
Pumping (labor and material)	.....	.....	.....	.....
Depreciation:				
Haulage equipment	0.09	0.20	1.85	2.20
General equipment	1.00	1.00	0.50	0.50
	$1.09	$1.20	$2.35	$2.70
General expense	3.96	3.32	3.00	1.90
Camp expense	1.21	0.63	1.10	0.79
Corral expense	0.25	0.08	0.02	.....
	$5.42	$4.03	$4.12	$2.69
Total	$33.60	$30.56	$41.52	$36.79

General expense includes a proportionate charge for the expenses of the Provo
such as salaries, stationery, telephone, and supplies; also a proportionate charge f
expenses of the Washington, Chicago, and supervising engineer's offices. The
office covers approximately 68% of this charge, the Washington office 23%, the Cl
office 2%, and the supervising engineer's office 7%.

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SECTION 7

SHAFT SINKING IN ROCK

BY

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MINING ENGINEER

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Note. — Numbers in parentheses in text refer to Bibliography at end of this section.

1. Cross-section of Shafts

Cross-section may be rectangular, round, or elliptical. Choice depends on ... material, character of ground, and local practice. RECTANGULAR section ... where timber is cheap, and volume of water moderate. It requires ... excavation for given hoisting area, is best adapted for aver conditions in ... rock and is most widely used. CIRCULAR section is adaptable to ... or metal lining, and is strongest against heavy ground or water press. ... less surface and hence less resistance to passage of air, while the ... areas about hoisting compartments furnish required ventilating, pipe, ... space without providing special compartment. ELLIPTICAL section ... of strength of cylindrical form and space economy of the rectangular. ... to timber and to keep plumb when sinking.

2. Division into Compartments

Rectangular shafts with compartments side by side are most conve timbering (Fig 1). With the oblong section, width of collar between th cut holes may be made large, resulting omy of explosive (Art 7). In heavy if long dimension of the compartments the shaft, the dividers are brought c gether and are better able to withst pressure. In steeply dipping strata, t walls are best supported, and safety c increased, by laying out the shaft with its short axis parallel to strik strata. If such location is impossible, special precautions must be taker the long wedges of rock from which support has been removed.

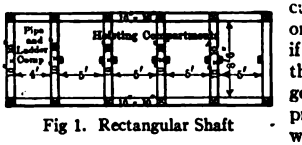


Fig 1. Rectangular Shaft

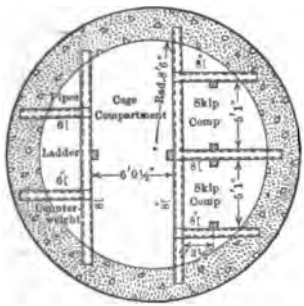


Fig 2. Plan of Cylindrical Shaft

In firm ground, where structural st lining is not the determining factor, sl be made to suit operating conditions. shafts, or those having a width of 2 ments, do not permit convenient arra of stations; landings are separated a plicated track arrangements are require nect them; and miners are in dang

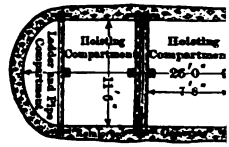


Fig 3. Plan Modified Ellipti

crossing between compartments. The Forman shaft (Comstock lode) was sunl form of an *L*, 3 compartments being in line with a large pump and service comp at one end; it proved unsatisfactory; unequal pressures caused timbers to crac and repairs were costly.

Cylindrical and elliptical shafts, the shape of which assures str strength, may have compartments arranged as operating conditions d Fig 2 and 3 show typical plans.

3. Size of Shafts

Cross-sectional dimensions are influenced by: purpose; tonnage hoisted; quantity of timber and supplies to be lowered; number of under employes; amount of water to be raised; method of hoisting (cage or skip); acter of ground; and questions of ventilation, cost, and operating charges minimum cost per foot of depth, a shaft has a cross-section of approx 4 or ft; smaller sections cost more, due to cramped positions of miners and in to drill holes to get best effect from explosive. When a shaft larger th minimum is required, size may be determined from above factors.

Prospect shafts are frequently of minimum size, and may have but 1 compar which serves for both hoisting and ladderway.

Air shafts may have 1 or 2 compartments, the second being a small ladderwa fireproof curtain wall. Single-compartment shafts in solid rock are frequently cyli and left unlined. Most economical size for air shaft, or air compartment of working may be determined by balancing annual charges of different cross-sectional areas cost of forcing the required volume of air through these areas.

Sinking shafts are usually divided into 2 or more compartments, one of which contains ladders, pipes, and wires. Shafts used exclusively for hoisting sometimes have no ladderway; all chance of injury to human life being avoided, hoisting is governed accordingly and is frequently automatic.

Size of compartments. Minimum size is determined by cage needed to accommodate the mine car in use (Sec 11, Art 3; Sec 12, Art 22–24); or, for skip hoists, the volume of ore to be hoisted per trip is the governing factor (Sec 12, Art 2). In American metal mining, hoisting ¾ to ⅞-ton cars, size of hoisting compartments, inside of timbers, varies from 4 by 5 to 5 by 7 ft; or for a 3-compartment shaft, a net clear section of approx 4.5 to 6 by 12 to 18 ft. In coal mining the cars are larger and compartments 6 by 10 or 7.5 by 12 ft are common.

Two-compartment shaft (one hoisting compartment), equipped with an underground hoist, was common in the early days of American metal mining. This is uneconomical of power, and is to be recommended only for shallow mines of small production.

Three-compartment shaft, with balanced hoisting in 2 compartments, is favored for economy, and is therefore usual for mines of moderate production.

Lowering men and materials in a 3-compartment shaft, hoisting 24 hr per day, leaves from 50 to 70% of total time available for ore carrying; this percentage may be increased by using a small cage compartment in manway (Fig 4), in which can be handled supplies and mine officials, and when the shaft is used for sinking. Such a compartment is unsuited for handling miners when changing shift and is awkward for handling timbers. Its cage is generally operated by a small auxiliary, unbalanced hoist.

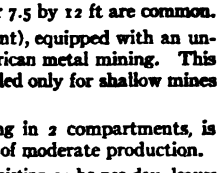


Fig 4. Shaft with Service or Sinking Compartment

Four-compartment shaft usually has 2 ore-hoisting compartments, a service compartment, and a pipe and ladderway. The ore-hoisting compartments are relieved of other work; men and supplies are handled promptly in the service compartment, which in emergency may assist in hoisting ore.

A small portion of manway is sometimes set aside for a counterweight, enabling the hoist to operate in balance. In a 4-compartment shaft, the ore-hoisting compartments are often equipped with skips, and the service compartment with a cage. Its firm (Fig 21), so that timber trucks can be run onto the cage without standing the timbers. In very wet shafts, to facilitate handling the pumps, it may be desirable to make a part of the pipe and ladderway as a small cage compartment.

Five-compartment shaft, with 4 ore-hoisting compartments and a manway, has advantage of balanced hoisting at all times. Two compartments are generally from 40 to 60% of the time for service work.

Conditions of great depth, large production, or a combination of both, involve practical questions of hoisting and ventilation which make necessary large shafts with numerous compartments.

The northeast shaft of the Government Gold Mining Areas, Witwatersrand, has 7 compartments, rock section, 45 by 10 ft. The 7-compartment Wolhuter shaft, Rand, has a section 46 by 9 ft. In the Lake Superior iron district, U S, 13 by 21-ft shafts are in use. U S colliery shafts range in rock section up to 14 by 50 ft.

Rock section should be cut of sufficient size to allow 3 or 4 in outside of timbers, for lagging and wedging. Rock removed in excess of this involves needless expense in breaking, mucking, and timbering.

4. Sinking Plant

Temporary plant used for sinking commonly consists of following equipment. Hoisting apparatus: windlass, whim, or engine, together with hoisting rope, and

buckets or skips. Support for sheave: tripod, derrick, or headframe. ment for removing water: bailing tanks, steam, or electric sinkin Boiler plant. Air compressor, if machine drills are used.

Windlass and whim (Sec 12) are often used in starting a shaft and in sinkir surface soil before the power hoist is installed. Amount of water influences which their use is practicable; for depths over 20 to 30 ft their application to new work in remote regions, or where mechanical power is not available.

Sinking engines should be strongly built, of duplex type; many pend on their reliability. For depths to say 500 ft, and hoisting with bucket (Sec 12, Art 21), a friction-gear engine of 12 to 30 rated hp is a pending on size of shaft. After depth of approx 500 ft has been reached engine, preferably of reversing type, is commonly employed.

Tripod made of timbers bolted together at top, from which point sheave is s forms a simple sinking headframe. A second sheave is sometimes fastened t of tripod leg nearest the hoist, to lead rope off horizontally.

Stiff-leg derrick is often used for sinking through surface soil; it does not exer on ground immediately surrounding shaft, nor interfere with placing timbering o of permanent shaft collar. Donaldson (2) recommends for colliery shafts a der 40-ft boom and 30-ft mast of 12 by 12-in timber. Where derrick is used for s depths of 100 ft or more, provision should be made to prevent it from swinging wh is in the shaft.

Sinking headframe design is the same in principle as that of operat (Sec 12). It is smaller; usually has one sheave; and the distance sheave and crosshead in the dumping position is small. Sinking he should embody features for dumping buckets or skips; for protecting w on the surface and in the shaft from falling pieces of rock while dumpi for minimizing work of topmen, in dumping buckets and removing brok A contractor's sinking frame should be so designed as to be portabl erected and dismantled.

If conditions permit erection of the permanent hoisting plant and headframe for expense due to inefficient engine and duplication of equipment may be avoided. temporary plant is necessary, it should be so placed that the permanent plant erected in its proper location without interfering with sinking. In some cases, th hoist has been placed opposite the service compartment and later used for its o

Water up to approx 1 000 gal per hr may be hoisted, much of it filling broken rock in the buckets; for greater quantity sinking pumps are n (Sec 40, Table 11). For given character of rock, cost and speed of sha ing vary directly with increase of water.

Spring Mines vertical shaft, on the Rand (41), was sunk through heavily water strata. Size, rock section, 42 by 9 ft. Labor per shift, 70 natives, 1 white foren white assistant; 3 8-hr shifts per day. Cost per ft of shaft, $238.35. Sinki divided into 3 periods. 1. From time of striking water to installation of pump 2. From installation of station to installation of permanent pumping plant. 3 installation of permanent pumps.

	Period 1	Period 2	Peri
Depth beginning, ft	459	650	1 1
Depth ending, ft	650	1 142	2 8
Depth sunk, ft	191	492	1 6
Aver speed per month, ft	27.3	70.3	1
Water, gal per min	1 389	1 199	.....

Questions of steam economy in design of boiler plant must be subservi other considerations, particularly in a shaft where a sudden flow of water m encountered and pumps drowned if sufficient steam is not available.

...st of contractor's new sinking plant for a 500-ft shaft, making 30 to 40 gal of water ... is given by Donaldson (2) as follows:

...sting engine	$1 000	2 buckets	$150
...0-hp boilers and setting	1 800	Rope	150
...pe and accessories	500	Buildings	500
...0-hp feed-water heater	300	Dump cars and rails	300
...ir compressor	1 750	Electric lighting plant, 10 kw.	750
...ls and steel	1 000	2 sinking pumps	500
...ar and clamps	100	Small tools and sundries	500
...	400	Total	$10 200
...frame	500		

...st of erecting and dismantling plant, $1 000 to $2 000.

5. Organization

...re are two general systems of shaft-sinking organization. In one, the ...e men drill and blast the round, and broken rock is loaded by lower-... men. Efficient management is needed to prevent miners and muckers ...ering; when properly handled, this system is generally the cheaper per ft of ... The other system uses the men as drillers or muckers as occasion demands. ... first system both crews may work together in the shaft, or independently ...arate shifts. In the latter case, the bottom is carried in benches, and im...tely on going down all hands clear enough space for drills to begin work.

...sinking the No 5 Tamarack shaft, Michigan copper district, the cut and one side ...drilled and blasted in advance of the other side. Drillers' shift was timed to begin ...ddle of the muckers' shift. Muckers first cleared the side which had not been ...d and made it ready for the machine men, who then drilled and blasted before the ...d of the next shift of muckers. Drillers and shovelers each worked 2 shifts per 24 hr ...minimum amount of interference.

...k Oak shaft, Soulsbyville, Cal, was sunk with hammer drills. One crew drilled ...ire round of holes, and blasted and mucked the inner row of cut holes, in one ... During the two succeeding shifts, remainder of the ground was mucked and shaft ...d down ready for the drillers.

...e of the best machine-sinking records on the Rand was made at the Village Deep, ...g 3 8-hr shifts per day. The holes drilled during one shift broke ground for the ...g 2 shifts of muckers. It was often necessary to load and blast a round a second ...n a third time, the bottom being cleaned between blasts. This made necessary the ...ce of machine men or their equals in intelligence between regular drilling shifts for ...g, charging and firing the holes. Sometimes holes are blasted in relays, the broken ...ing removed after each blast.

...l men in the shaft drill or muck as needed, each shift can pick up the work ...less of the stage at which the last one left it. This system, operating 3 ... per day, makes for speed, with slightly increased cost per ft of shaft; but ...ives the use of experienced drill runners to do muckers' work.

...ection No 3, 55° inclined shaft of the Canadian Copper Co, the record for part of ... given in Table 1, was sunk by using 12 3½-in piston drills mounted on 6 columns, ... on a machine. Same crew did the drilling, shoveling, or timbering, as required, ...ing 3 8-hr shifts per day. They received a bonus equal to 1% of their wages for every ... over 100 ft per month. Hoisting was done in skips by a 2-drum geared electric hoist. ...er was 40% Forcite, fired electrically by delay-action fuses. Little timber was re... except for skip rails. Size of shaft is 35 by 9 ft, with 5 compartments each 6 ft 6 in ... 6 in.

...e features of the hammer drill which make it possible to take advantage ...e peculiarities of the face tend at the same time to minimize the regularity ...r work, and it is usual to find the same men drilling or mucking, as required.

...sbury shaft, Newport mine, Ironwood, Mich (3) during the 9 months ending Dec 1, ... was sunk at the aver rate of 185 ft per month, using 12 Jackhamer drills with same 12 ...lling and mucking.

Table 1. Partial Shaft Sinking Reco

Month	Advance for month	Percentage of time							Tons hoisted	Rounds	Drilling shifts	Feet drilled	Labor			
		Drilling	Blasting	Blowing smoke	Shoveling	Timbering	Setting up	Miscel delays					Drill runners	Drill helpers	Shovelers	Surface trammers
1915	*a*															
Aug	126	28.3	13.1	2.7	47.2	3.5	4.5	0.7	3590	18	308.3	6678	513.6	519.0	987.8	359.4
Sept	146	28.2	7.8	5.8	50.0	2.3	5.0	0.9	4161	21	294.0	7291	476.4	478.8	1038.9	349.4
Oct	164	34.6	6.2	3.7	45.4	2.9	6.4	0.9	6200	21	302.7	7671	471.5	489.2	1080.8	467.0
Nov	144	28.4	5.9	2.5	50.0	3.6	4.6	5.1	4178	18	288.8	6696	483.7	494.8	983.5	355.0
Dec	133	26.8	5.5	2.2	57.5	2.5	4.1	1.4	4419	16	267.8	6676	417.1	416.3	1127.5	338.0
1916																
Jan	112	26.8	5.2	2.4	52.9	3.9	4.2	4.7	3565	16	277.4	6630	468.2	461.8	1043.3	358.0
Feb *b*	99	29.6	6.0	2.3	52.6	2.2	4.9	2.5	3391	13	256.2	5700	388.3	397.6	871.0	299.0

a. Shaft 206 ft deep, Aug 1, 1915. *b*. S

Where shafts are sunk by hand no attempt is usually made to distingui tween drillers and muckers. Drilling commences when the last bucket of h rock has been hoisted, and men return to bottom as soon as the smoke has cl

Record sinking on the Rand has been done by hand (Kafir labor). Each shift o down loaded rock broken by preceding blast; the next round was then drilled and l at end of shift. Three rounds were drilled, blasted and mucked per 24 hr. As mar are employed in the shaft as can be crowded into the space. At the State shaft, 45 ft, 82 Kafirs were used per shift. In very deep shafts, air is blown down so that wh last hole is fired the next shift starts down, meeting the ascending fumes; the ti quired to pass through the foul-air zone is so short that no bad effects result.

Delays may be classed as follows: removing drills and other equipment shaft bottom preparatory to blasting; clearing bottom of smoke; clearing walls and timbers of loose pieces of rock and securing bad ground after bla lowering timbers, which may be put in place during drilling.

Where pumps are in use the suction is removed from bottom and pump stopped blasting; in very wet shafts, this may result in several feet of water accumulating time of blast, which acts as a cushion to protect timber and pumps, and absorbs m the powder smoke. This must be pumped or bailed miners can go down.

Table 2. Bonus Paid White Employes, Village Deep Shaft (36)

Advance per month, ft	Bonus per ft	Division of bonus	
		Occupation	%
60 to 70	$ 9.73	Machinemen	60
70 to 80	14.59	Timbermen.....	10
80 to 90	19.45	Boss shovelers	8
90 to 100	34.04	Topmen.....	4
over 100	48.63	Miscellaneous.....	12

Rock section of shaft, 28 by 8 ft.

By operating 2 shifts pe much of the above work be done between shifts; is also time to make up fo usual delays. With 3 shif such opportunity exists; is increased, but at a sli greater cost per ft than w shifts.

Bonus system of pa workmen lends itself readily to shaft sinking, and in many cases has increased s and reduced cost per ft. Bonus is paid either as a percentage of wages for foot in excess of a given standard, or the company may allow a lump sum ft to be apportioned among workers, as at Village Deep, Rand (Table 2).

Mine, Creighton Mine, Canadian Copper Co

Timbermen	Ft drilled per drilling shift	Ft adv per drilling shift	Ft adv per round	Tons per skip	Skips per hr	% time loading	% time hoisting	Tons per ft advanced	Tons per shoveler shift	Powder 40% Forcite: Total lb	Powder 40% Forcite: Lb per ft advance	Shifts of labor per ft advance: Drillers	Helpers	Shovelers	Surface trammers	Bosses	Nippers	Hoistmen	Timbermen
109.6	21.7	0.41	7.0	1.2	8.4	71.3	28.7	28.5	3.6	7700	57.5	4.1	4.1	7.8	2.9	0.8	0.7	0.6	0.9
76.9	24.8	0.50	7.0	1.4	8.4	72.2	27.8	28.5	4.0	8400	66.7	3.3	3.3	7.1	2.4	0.7	0.6	0.6	0.5
109.6	25.3	0.54	7.8	1.4	9.5	67.8	32.2	37.8	5.7	6150	37.4	2.9	3.0	6.6	2.8	0.6	0.6	0.6	0.7
112.0	23.2	0.50	8.0	1.1	8.9	55.9	44.1	29.0	4.2	9050	62.8	3.4	3.4	6.8	2.5	0.7	0.6	0.6	0.8
85.1	24.9	0.60	8.3	1.4	7.9	48.1	51.9	33.2	3.9	11350	85.3	3.1	3.1	8.5	2.5	0.7	0.6	0.6	0.6
111.7	23.9	0.40	7.0	1.3	7.1	52.9	47.1	31.8	3.4	12800	114.3	4.2	4.1	9.3	3.2	1.0	0.7	0.8	1.0
98.1	22.2	0.39	7.6	1.5	7.4	51.0	49.0	34.3	3.9	10350	104.5	3.9	4.0	8.8	3.0	0.9	0.6	0.7	0.6

Feb 24, 1916, to cut station at 14th level.

Safety precautions (4). No other operations should be carried on, nor tools material raised or lowered to or from other points in the shaft, while men are in bottom, unless they are protected from falling material by a well constructed TIMBER PENTICE extending over nearly the entire area of shaft, with openings for passage of buckets. In deepening a working shaft, an ROCK PENTICE should be left, or timber bulkhead built before sinking begins.

Trap doors, normally in closed position, should be provided at collar to cover shaft with additional set of trap doors when dump point is above collar, to prevent of falling rock breaking through the collar doors. At Woodbury shaft (3) in the shaft was provided with a felt hat stiffened with resin and shellac. These hard and capable of resisting severe blows from fragments of falling rock.

or skips should stop at least 15 ft above bottom, until rung down by one of Ladderways should be provided to within such distance from bottom as will injury to them from blasting; from end of these, chain, wire rope or wooden ex- ladders, should be provided to reach bottom of shaft. This assures safety of men failure of hoisting engine, fire or sudden inrush of water.

electric hoists are used, electric lights in shaft bottom advise sinkers of interrup- current. In some districts the law provides that all blasting in shaft sinking shall by electricity.

6. Drilling

Drills used in shaft sinking are: hand-churn, single or double hammer, and or hammer machine drills.

Hand-churn drill in the hands of energetic workmen may be used with ad- in soft rock. Hole is usually started with hammer and drill. More must be taken in shaping the bit than for hammer drilling, and a low temper able.

Hand hammer drilling is best applicable with low-priced unskilled labor. number of comparatively shallow holes, approx 3 to 4 ft, are drilled per Advantages are:

Greater speed, especially where many drillers may be packed into the shaft bottom, Rand, where the world's records in speed have been attained. Such speeds can approached in hand drilling with high-priced labor, as the men will not tolerate such conditions. At the State shaft, Witwatersrand, 45 by 10 ft, 82 Kafir drillers

were employed per shift, giving a max advance of 233 ft per month. (*b*) In placing full advantage may be taken of all peculiarities of rock in shaft bottom, resulting in economy. (*c*) Lighter charges are used in shallow holes, compared with machin ing, with less, shatte shaft walls, and timbering. Shaft sec confined more exactly sired size, and timbe facilitated. (*d*) By a ing bottom in a se benches, the burden hole becomes less, wit omy of powder. In Rand shafts, the top is sometimes 40 ft abo sump. This method feres somewhat with ing, especially if onl hoist is in operation. bench system may also be used with machine drilling, but is not so advantageous hand work. (*e*) None of the delays incident to machine drilling, as for setting up and hoisting them out before blasting.

Table 3. Comparison of Hand and Machine Drilling, Goldfield, Nev (40)

	Hand	Machine
Men per shift	5	6
Shifts per day	3 8-hr	3 8-hr
Aver advance per day, ft	3.81	4.2
Powder per ft advance, lb	8.0	11.68
Powder per ton rock broken, lb	0.93	1.25
% of total time:		
Drilling and blasting	37.6	29.2
Mucking	40.4	45.5
Timbering	22.0	25.3

Comparison of hand and machine drilling at Goldfield, Nev. A 7 by 16-ft v shaft was sunk from 470 to 700 ft by hand, 60% of the drilling being done by churn and 40% by single-hand work; from 700 to 1 200 ft, 4 2¾-in Ingersoll-Rand ma were used. Rock medium hard.

Table 4. Comparison of Powder Consumption, Hand and Machine-dri Holes, Rand

Hand		Machine	
Shaft	Powder per ton rock broken, lb	Shaft	Powde ton r broke
Central Roodepoort	0.7	Cinderella Deep	0.6
Simmer East	0.5	Durban Roodepoort	1.3
Brakpan	1.1	Simmer and Jack No 2	1.1
		Village Deep	1.0

Table 5. Comparison of Hand and Machine Sinking, Rand (29)

Name	Size, ft	Aver advance per month, ft	Rock	Cost per breaki and shov
Machine drilling				
Rand Collieries	34 by 9	120	Quartzites	$2.61
		103	Hard diabase	2.92
		112	Soft shales	2.05
Kleinfontein	34 by 9	107	Quartzites	2.82
Hand drilling				
Brakpan	43 by 9	124	Quartzites	$2.17
City Deep	46 by 9	126		2.23
Hercules	47 by 9	119		2.04
Wolhuter	46 by 9	95		2.55

Hammer drills (Sec 15), which work best on down-holes, are well adapte shaft sinking. They partake of the advantages of hand work, without nece

skilled labor willing to be packed in the shaft bottom. They are best ... for vertical or steeply inclined shafts, where fractures or cleavages are ... in strata, since in this case the placing of each hole with regard to local ... of the rock is of most importance.

... drills have also been used successfully in ground where each successive round ... in practically the same manner. The advantage of the hammer over the re... drill is not in better drilling speed, but in the shorter time for setting up, tear... shifting from one set-up to another.

... Black Oak shaft, Soulsbyville, Cal (5), sinking was carried to the 1 600-ft level ... reciprocating drills, one miner and a helper on each machine. Below the 1 600-... hammer drills were substituted for the reciprocating machines, with the same ... of men in the shaft. The result was a 25% increase in rate of advance and a ... air.

... shaft (45), near Ducktown Tenn, rock section 7 by 14 ft, penetrated very hard ... schist. By substituting 6 Jackhamers for 4 reciprocating drills, an advance ... per month was made for 315 ft; about double the best previous record. (For ... details, see Art 21.)

Reciprocating drills (Sec 15) have general application for heavy shaft work, ... from question of speed compete successfully with other drills, and ... Rand even with hand work, though the conditions there for manual labor ... In rectangular shafts, reciprocating drills are usually mounted on ... bars (Sec 15), placed across the shorter dimension of the shaft. From 1 to ... drills are mounted on each bar.

... large American colliery shafts, drills are mounted on tripods; but the tripod is ... and more cumbersome than the shaft bar, and altogether the practice has little ... it in a rectangular shaft. With circular or elliptical shafts, the use of shaft ... difficult and tripods are the rule. In Europe several types of sinking frames ... devised for circular shafts, on which the drills may be so mounted as to com... entire cross-section. The frame with drills attached is raised to the surface ... blasting (6, 7).

7. Depth of Drill Holes

Depth of hole (Sec 6, Art 7) is a function of shape and size of shaft and char... rock. The deeper the cut holes the greater the width of collar possible ... the two opposite rows, and the less the number of breaking-out holes ... Usual depth of HAND-DRILLED HOLES is from 2 to 5 ft. With RECIP... DRILLS, the maximum depth consistent with powder economy is ad... as this reduces the percentage of time lost in setting up and taking down ... holes, and in raising and lowering the drills before and after ... The proper depth of hole should be determined by a series of experi... any particular rock and other conditions. In absence of other data, ... may be assumed equal to one-half, or in soft ground three-fourths, the ... the shaft. In some districts, the practice has been to drill long holes ... 2 or even 3 times; this involves waste of explosive unless, before charg... hole is partly filled with sand or other easily removable material.

... to 10 or 12-ft holes, or deeper, is the excessive weight of steel necessary to ... a hole at a size large enough for a sufficient charge of powder. Long steel is ... to handle in sinking buckets. The drill steel for HAMMER DRILLS being ... that for reciprocating machines, the limit of length of hole which it is possible ... still have a cross-section large enough to hold enough powder to break well, ... the way the steel holds its gage.

... Woodbury shaft, Ironwood, Mich, 10-ft holes were possible, in soft slates, while ... and quartzites, starting with a 2¼-in bit, 8 ft was the max depth of hole.

... hard rock at the Gordon shaft, near Ducktown, Tenn, reciprocating machines ... 7-ft holes and made approx 5 ft advance per round. Hammer drills put down ... holes, and made approx 2 ft advance per round.

8. Location of Drill Holes

Drill-hole location in sinking is governed by fact that only one free exposed, and that to obtain max efficiency from charge 2 free faces are nec (Sec 5, Art 5). One or more key or cut holes are drilled at an angle to t face, and blasted in advance of the other holes, which are placed to take tage of new face formed when the cut is blasted. The cut is usually drilled middle of the shaft section, though in large shafts some engineers prefer it towards one end, retreating towards the opposite end in blasting, and each row of holes in series. This arrangement tends to throw the broke to one end of the shaft so that the opposite end may be speedily cleared machines to begin work while mucking is in progress. Local conditions, ding planes, reentrant angles, position of hoisting compartment, etc, som influence position of cut. The max blasting force is developed if oppos holes meet at the bottom and are fired simultaneously. The obtuse an tween opposite cut holes should be as great as conditions permit (Sec 6,

Hand-drilled holes are usually shorter and more numerous than machine-drilled; they are placed with care to take advantage of bedding planes, cracks and shape of face, not with especial regard to symmetry of arrangement. The cut is sometimes carried well in advance of the other holes, which recede towards the shaft ends in a series of benches.

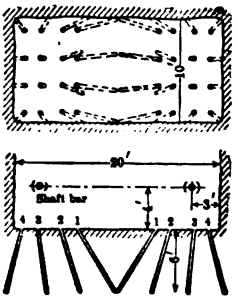


Fig 5. "V," Center or Wedge Cut for Shaft Sinking

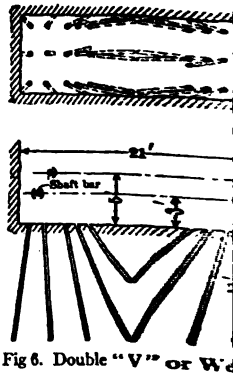


Fig 6. Double "V" or W

Machine-drilled holes are placed and pointed in accordance with a system designed to give desired results increased powder consumption. A symmetrical a ment facilitates the operation of the drills, and the max number of holes to be drilled from each of the shaft bar (Sec 6, Art 6).

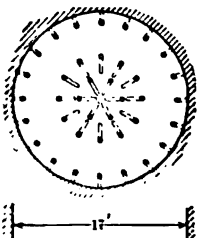
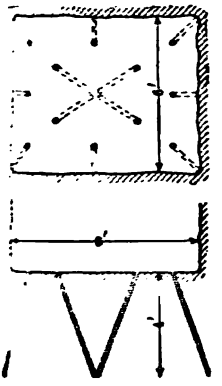


Fig 7. Pyramid Cut for Circular Shaft

"V," center or wedge cut is most commonly u comprises a number of pairs of holes inclined towa other from opposite sides of the short center line shaft, their bottoms being close together or preferabl ing. Fig 5 shows a simple "V" cut, with positions bar for drilling. Holes in lines 1 comprise the cut blasted first, holes in lines 2, 3, and 4 being fired in su Fig 6 shows 4 rows of holes for a deep cut. In soft it is sometimes sufficient to drill only one line of c on each side, or, if the second line is used, to drill th half the depth of corresponding hole on opposite side

Pyramid cut is rarely used except in circular sh consists of center holes surrounded by others more concentric and so inclined as to form a sump at c shaft. Fig 7 shows arrangement of holes in a Catskill Aqueduct shaft. Pyramid been applied to rectangular shafts in the Joplin, Mo, district (Fig 8). Cut h

...st, followed by side and corner holes in the order named. In small shafts, ...ay be used to blast the sump.



Pyramid Cut for Rectangular Shaft

Center and wedge cuts are sometimes combined by pointing the holes nearest the center of

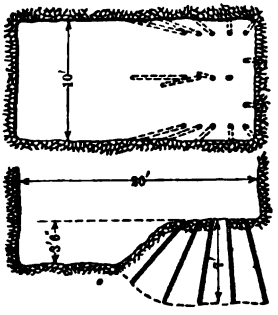


Fig 9. Bench or Stope Cut

... to take out a small sump; the remaining holes are then arranged in ordinary ... fashion.

Bench or stope cut is sometimes used in tight ground (Fig 9). The cut is alternated ... side of the shaft to the other. There are always 2 more ... faces, thus increasing the efficiency of blasting. This ... to project the broken rock towards the opposite end of ... instead of directly upwards as do the wedge and center ... there is less delay in setting up drills, and risk of ... to timbering and pumps is reduced; also one end of shaft ... lower than the other, facilitating mucking and drainage.

Fig 10. Bottom Cut for Flat Inclined Shaft

Bottom cut, similar to that used in tunnel work (Sec 6, Art 6), ...times useful for flat, inclined shafts (Fig 10).

8. Explosives, Blasting, and Mucking

Explosives (see Sec 4 for full discussion). 40% gelatine dynamite is generally ... sinking in American practice, where holes are drilled by hand or recipro-

Table 6. Consumption of Explosive in Shaft Sinking

Shaft	Size of shaft, ft	Explosive used (dynam)		Rock
		Strength, %	Lb per cu yd	
...s coal, Pa	13 by 26	40	1.26	Shale, slate
...	14 by 18	40	1.12	Sandstone
...educt	10 by 22	60	2.10	Quartz conglom
...	17 diam	60	0.94	Shale
..., Cal	18 by 11	40	4.6	Schist, slate (*a*)
...com	12 by 24	50	1.3	Shale
... Nev	4 by 8	40	3.0	Andesite
Eagle, Cal	7 by 12	40	1.3	" (*b*)
...nd Jack No 2, Rand	7.5 by 21	93 (*c*)	2.3	Quartzite
... East, Rand	.....	93 (*c*)	1.05	" (*b*)

(*a*) Inclined 60°. (*b*) Hand drilled. (*c*) Blasting gelatine.

cating drills. In some cases, 60% dynamite is used in the cut holes and bottom of others. The relatively small, deep hole drilled by hammer dr led to the use of stronger dynamites; 60 or 80% gelatine is common, w hard ground 2 or more sticks of "100% gelatine" are sometimes placed bottoms of holes. In Rand shafts, 93% blasting gelatine is common. The amount of charge should be determined by experiment.

Mucking is facilitated if the major portion of the rock is broken in pieces weighi 20 to 200 lb, which can be loaded by hand. Shoveling in a shaft bottom is diffic takes more time than handling an equal quantity of rock in large pieces. Where of the shaft is drilled and mucked in advance of the other, shoveling is facilit covering the bottom, opposite the side to be blasted, with steel plates; the blast throw the broken rock onto the plates.

Blasting may be done with ordinary cap and fuse, timed to give succe shots, or by electricity. By the latter method the entire round may be once, or the different series of holes timed to detonate as desired by use c electric blasting caps or electric delay-fuse igniters.

In some districts the law requires electric blasting in sinking. Timbering carri to the bottom is liable to injury from concussion and rock bombardment caused simultaneous electrical detonation of an entire round of holes; this danger is less use of delay devices, or by firing the series of holes in succession.

In using ordinary fuse and caps, many engineers advise doubling the fuse, 2 de with equal lengths of fuse being inserted in each hole. If the fuse is wound arou erly placed nails or hooks and cut at only one of them, with caps crimped on bo the mark made by the other nail or hook will show the place where the fuse is to be spit at blasting time.

Where the shot-firers begin spitting at each end of the shaft and work tow middle, they need not walk over lighted fuse, possibly stepping on one and puttin The men are also nearer the bucket at the finish. This method involves cutting for each series of holes at different lengths, so that the cut holes which are general center will explode first. When spitting begins at the center, all fuse may be cut length and proper order of firing naturally follows; and if the entire round is through any cause, the cut holes and those nearest to them are sure to explode. keeper fuse, cut so that it will burn out about 2 min before the first charge is explode, serves as a warning that it is time to get to safety. In wet shafts on th fuse is spit by torch, or "Cheesa stick," made by splitting blasting gelatin, wra around a pine stick about 18 in long and covering with clay. The fumes give readily absorbed by the water without any bad effect. In part of the Woodbu (3), the ends of the fuse were lighted by being placed in a cardboard box containi powder which was ignited by an electric fuse. Great care was exercised and end kept dry. Considerable smoke resulted from burning black powder and long lengths

Mucking, or loading broken rock into hoisting conveyance, occupie 50% of shaft-sinking time. Number of men employed depends on me organization, size of shaft, and number and type of containers for hoisti For VERTICAL SHAFTS, a bucket or skip is sometimes suspended from bo sinking cage, which is provided with long guide shoes to permit lowerin the last set of timber. In other cases, sinking crossheads (Sec 12, Art used to prevent bucket from swinging. On the Rand, skips have been us guide shoes long enough to engage the lower end of guides when the sk is on shaft bottom; they are considered safer than buckets. For IN SHAFTS, buckets sliding on skids, or suspended from a carriage runnin cableway, and skips running on regular track, are in general use. Ten track, capable of being raised on blasting, is provided to reach from end bering to shaft bottom.

Empty bucket should always be available at bottom to render muckers inde of irregularities and delays. In large shafts, where more shovelers are employ can crowd around one bucket or skip, two compartments are sometimes used for

10. Ventilation (See also Sec 14)

...e sinking, enough air must be delivered at shaft bottom to remove powder ... and rock dust, and to enable sinkers to work in reasonably pure atmosphere.

Natural ventilation, set up automatically, may serve to great depths; in ... ases, **mechanical** ventilation must be adopted at the outset. Some of the ... governing natural ventilation are, character and temp of surface atmos. ... f strata penetrated, and amount of water falling in shaft.

Natural ventilation (Sec 14, Art 10) may be aided in several ways. If a small portion of shaft area is partitioned off by a brattice (Sec 14, Art 6), and this compartment ... up into the headframe by a chimney, difference of air head will cause circulation. ... steam sinking pumps are used, and steam pipes carried in a separate compartment, ... is usually sufficient to establish a rising current; a steam jet directed upward ... shaft bottom will accomplish same result. It often happens that, even with no ... the space around the steam pipes is upcast while the opposite side is downcast.

Fan or blower on surface, connected to a wooden or sheet-iron pipe 12 to 18 in diam, ... close to shaft bottom, may be used to force down fresh or exhaust foul air. It is ... satisfactory to use a fan to exhaust blasting fumes promptly; the fan is then ... to supply fresh air. If the pipe or chimney be of wood, the boards should be ... and painted to reduce leakage.

... compressed-air drills are used, the exhaust will generally remove dust and pre... air from becoming vitiated. Powder smoke is ... out by opening the air valve at the bottom ... Spraying water down the shaft facili... powder smoke.

11. Sinking in a Working Shaft

... shafts are frequently deepened while ... mining operations are being carried on ... Sinkers should be protected from falling ... by a rock pentice (Fig 11); or by a ... timber bulkhead, which is sometimes loaded ... or 15 ft of waste rock.

... are two general methods: (*a*) rock is ... direct to surface from shaft bottom ... special sinking compartment; (*b*) rock ... to the lowermost working level, by ... electric or compressed-air hoist, whence ... to the surface by regular hoisting ...

The rock pentice may be left across ... shaft area, in which case a short incline ... sunk from the level above before the ... widened to full section. If the pentice ... across bottom of hoisting compartments ... small opening may be cut in line with the ladderway, through which ... carried on (Fig 11).

Fig 11. Sinking Under Rock Pentice

12. Shaft Raising

... are sometimes raised instead of sunk, where an additional opening is ... a working mine. A raise is made on the center line of the proposed ... a cross-section just large enough for good delivery of the broken rock ... provide room for drilling. Where speed is an object, several points of ... may be opened by driving drifts from the mine levels to intersect the line ... When the different sections have been connected, the raise is ... out to the desired cross-section, thus eliminating any slight errors of ... If work is to be done in one lift, the raise is sometimes carried at ... section.

Shaft raising is faster and cheaper than sinking, no shoveling nor pumping bein essary. Expense of tramming rock to main hoisting shaft must be taken into ac but hoisting itself is cheaper, due to greater efficiency of the regular plant as con with a small sinking hoist. The broken rock may sometimes be used undergro stope filling. In bad ground, temporary timbering is placed, part of which may quently be salvaged. (For further notes on shaft raising, see Gordon shaft, Art

13. Design of Lining

Character of shaft lining depends chiefly on character of ground and of shaft. No exact rules are applicable for computation of loading and str of shaft support, and except where hydrostatic pressure is present, the stren the lining is based on previous experience under similar conditions. The must be strong enough to support the hoisting guides, and to withstand on shaft walls which usually increases with size of section. Wet ground to be heavier than dry.

In some firm rocks, it may be only necessary to keep loose pieces from dropping i shaft. In case of heavy water-bearing strata, the lining must not only resist thr walls but also dam out the water. Under these conditions, circular shafts line tubbing (Art 18) are usually best.

Materials used for shaft lining are timber, steel, C I, concrete, brick and

Shaft collar. The height above surrounding ground is determined by ties for disposing of waste rock, or other local conditions, as surface drainag a level place, collar is sometimes raised 15 to 20 ft and waste filled in arou If the shaft begins in solid rock, a collar set is framed by extending the end plates, or both, 4 to 6 ft on either side of shaft mouth; from this s hung the succeeding sets, until the first bearing set (Art 14) is placed.

Where collar set is at the surface, the extended ends of plates rest on timbers or c piers. A collar set above the ground level may be placed as described and timberi up to the required height; or the collar set itself may be raised and supported b and diagonal bracing until waste is dumped around it.

If soft overburden must be penetrated before solid rock is reached, the upp tion of shaft may be cribbed (Art 14), or preferably lined with concrete or ma

Often an open pit is made to bedrock, the sides being allowed to take their natur so as not to interfere with placing of the concrete curb. After collar is finished, th around it is filled. Guide bolts may be imbed concrete, or regular shaft sets placed inside which should extend far enough into solid rock out surface water. Both timber and concrete are frequently designed to provide foundations headframe.

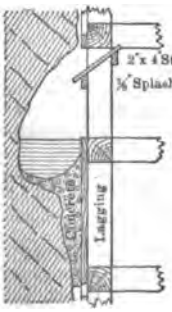


Fig 12. Water Ring for Timbered Shaft (2)

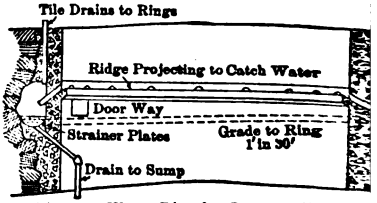


Fig 12a. Water Ring for Concrete-lined S (*Mines & Min*)

Water rings are placed at intervals in vertical shafts to intercept falling which is then led to a sump and pumped to surface. A groove is cut arou shaft to a depth of 1.5 or 2 ft, on the edge of which a dam is made of timb

...ed in clay or concrete, or of clay or concrete alone, to form a channel behind ... water is collected and led thence to a sump.

...bered shafts, water is guided into the ring by short planks placed in an inclined ... to intercept its fall (Fig 12). In concrete-lined shafts (Fig 12a), the rings are ... behind the lining, water being led to them by lines of tile pipe, placed vertically in ... the concrete; a small projection on inside of lining serves to catch the water fall... shaft. The ring should have sufficient grade to the outlet pipe.

14. Timbering

Rectangular shafts are commonly supported by timber; in recent years, the ... steel and concrete is increasing. Timber is subject to decay and in dry ... is a source of danger from fire; but, in shifting or swelling ground, it is some... preferable, due to greater elasticity and ease of replacement. Vertical shafts ...ported by **sets**, usually composed of 4 pieces of timber, framed together at ... and wedged against the walls. Types of timbering are cribbing, and ... shaft sets.

Cribbing may be of any size timber, round, hewn on two sides or squared. ...shafts are often cribbed throughout. For large shafts cribbing is chiefly ... passing through the soil ...ing bedrock. But, in ...ground on the Comstock ...ere cribbing was used ...ire depth of large shafts. ...west form is made of poles ...ired lengths and laid on ...each other; vertical strips ...side the corners keep the ...position. Planks may be ...laced on edge, with ends ...into each other, or cut ... and held in position by CLOSE CRIBBING consists ...ers placed skin to skin. ...14, 15, 16, show joints for ...bing. In Fig 16, $B = 0.5A$,

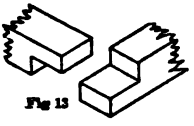


Fig 13

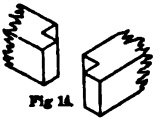


Fig 14

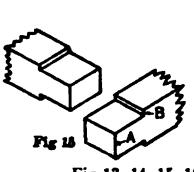


Fig 15

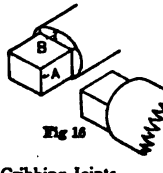


Fig 16

Fig 13, 14, 15, 16. Cribbing Joints

...ers being laid up log-house fashion. OPEN CRIBBING allows some space between ...e timbers; Fig 15 is a common type of framing, in which B is less than $0.5A$; ... $B = 0.5A$ the joint is same as in Fig 16.

...racter of ground permits, sufficient depth is excavated to place several feet of ...at a time; a set is placed and wedged at bottom, and cribbing built up to meet ...om above. In unstable ground, the sets are placed as the excavation advances.

Shaft sets. Each set (Fig 17) consists of 2 side pieces, or WALL PLATES, ...l to the long axis of shaft; 2 end pieces or END PLATES, parallel to short ...d at least 4 posts or STUDDLES, to keep the plates in proper horiz position. ...is divided into compartments, the dividing timbers, in same plane as the ... are called DIVIDERS, centers or buntons; opposite the ends of each divider ...sionary to set extra posts. Wall and end plates and corner posts are ...of same size; dividers are generally same depth as plates, but are usu...ower, interior posts may be of same size as dividers, or smaller.

... Sec 12, Art 19) are attached to end plates and dividers. Guide posts (8) are ...ed to stiffen and support guides; they run vertically behind guides and are ...into end plates and dividers. Guide girts (8) may be provided for same purpose ...posts; they are placed horizontally, in same plane as end plates and dividers, and ...ed between midpoints of studdles. The girts take some of side thrust of shaft ...ides supporting the guides.

For proper alinement of guides, room may be allowed in design of timbering for ir of a 2 or 3-in distance piece or filler between guide and end plate or divider. In cr bering shifts under pressure, the thickness of these fillers is increased or diminis alinement, without moving main timbers.

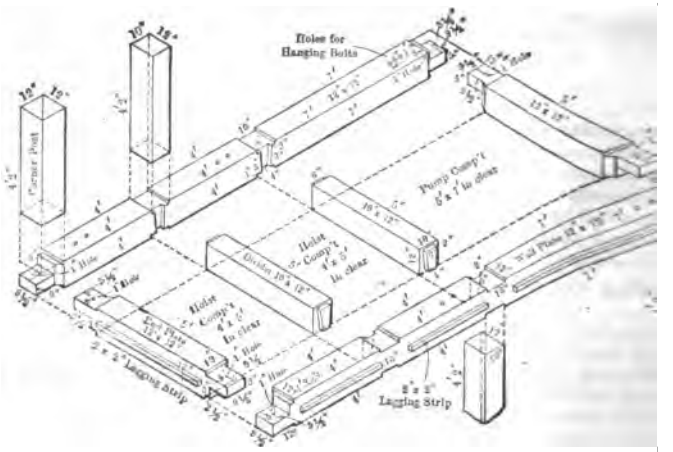


Fig 17. Shaft Set

Sawed timber is preferable to round for shaft sets, as better framed jo possible. Size of timber varies from 6 by 6 in for small shafts in good to 14 by 14 in for large shafts in heavy ground. Under ordinary condit by 10 in timbers are common. Distance between sets is variable; in so maximum is 6 to 7 ft, in bad ground they are placed closer. Usual dista to 6 ft. All the timbers of any one set are framed together accurately at wedged firmly against shaft walls by means of blocking and wedges and by at least two plumb lines hung in opposite ends of shaft.

Lagging. If ground is firm, with no tendency to shell off, no lagging i sary behind the sets. Ordinarily, however, safety demands complete c of rock walls. Lagging material: round poles placed skin to skin, or slabs and planks are common; galvanized, corrugated steel and buckl have also been used. Where several sets can be placed before being lag lagging is cut into lengths spanning 2 or more sets and shoved into Where conditions demand that lagging keep pace with shaft sets, 2 by GING STRIPS (Fig 17) are nailed to outside of wall and end plates and 2 to ging planks are set between these strips. Space between lagging and sh is packed with any available filling to prevent the walls from "starting."

To facilitate removal of broken or rotted lagging, cleats may be spiked to top an sides of the plates, to form a groove in which the ends of short lagging plank Thus placed, between instead of behind the sets, lagging is in shorter lengths and access. This method is not applicable to heavy ground.

Hanging bolts (Fig 18) are used chiefly for convenience in placing ti position. Each bolt is composed of 2 parts, exact duplicates, with hoo end to engage hook of its mate, and threaded at the other end, which through the wall plate and is secured by washer and nut.

...ance of adjustment, length of each bolt, from inside of hook to top of threaded ...d be 3 to 4 in greater than half the distance between the top of wall plate of one ...e bottom of the plate of next set. Diam of bolts ranges from 0.75 in. for small ... 1.25 or 1¾ in for large shafts with heavy timbers. C-I washers of large diam ...e used to prevent nuts from cutting into timber. 2 or 3 bolts per wall plate are ...ding on size of shaft section.

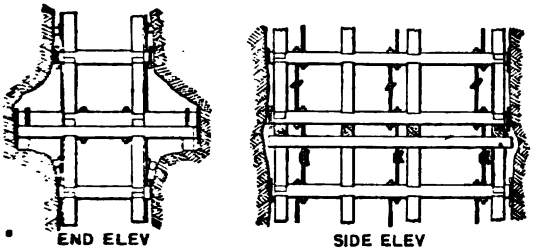


Fig 18. Bearing Timbers (8)

...aring sets, or bearers (Fig 18), are placed in the shaft at intervals of 50 ... to prevent general displacement of the timbering, and to take its weight ...cks and wedges, by which ordinary sets are secured, fail of their purpose ...or in part. Alternate wet and dry periods may cause this condition. ...ly, wedging should be such that ordinary sets are as solid as bearing sets.

...s are of two forms: (*a*) independent timbers; (*b*) extra long end plates, the ends ... are wedged in hitches in the shaft walls. Independent bearers are preferable; ... heavy timbers, of same breadth as end plates, but deeper as desired, and are ...arallel to end plates of regular set. No change is made in framing of regular plates, ...t on the bearers (Fig 18). Gains or daps in wall plate, which ordinarily receive ...uddles, now fit into a similar gain on bearer; studdles fit into gains on under side ... End plates and dividers are placed in usual manner and are sometimes bolted ...n as shown. Bearers are ordinarily placed only under end plates. Where weight ...t is great, as in large shafts, or where rock tends to slip, rendering it difficult to cut ...e hitch, they may also be placed under one or more of the dividers, as in Fig 18. ...reased strength, which theoretically should be sufficient to support the weight of ...e to next bearer above, 2 or even 3 bearers may be placed one on top of the other. ...ows sills placed parallel to wall plates in continuous hitches; on these sills the ...are securely seated and wedged.

...raming (9, 10). Usual method for vertical shaft sets is shown in Fig 17. ...ity of workmanship is important, so that each piece will go into place ...t cutting or trimming underground.

...ork is best secured by timber-framing machines, by use of carefully constructed ... or, in absence of these, by drawing a base line lengthwise on center of the most ... the 4 faces of the timber to be framed, and making horizontal and vertical ...ments with reference to this line. If the timber faces are not true, one may be ...e before laying out the framing. The face on which the base line is drawn should ...de of the shaft when set is in position.

...enons (horns) are 0.5 the thickness of the plates; those of wall plates are ... bottom half of plate; of end plates, the top half. Hence, when assem... end plates rest on wall plates.

...le is sometimes bored in center of tenons (Fig 17), and a wooden pin driven to ...ates together while erecting. A 45° bevel at each corner of a set is usually made, ...ecent plates are in contact to their full depth, thus minimizing effect of area of ...t by cutting tenons. At Butte, Mont, 0.5 to 0.25 in is generally cut off the end ...ion, to make the beveled edges draw close together while being blocked and ...levels are sometimes omitted at corners.

ment; hence it is better to set the instrument in one of the hoisting compartments a any error fall in ladderway. In steep shafts, transit must have auxiliary telesc bracket to screw into timbers, or a stretcher bar, are often better than a tripod for shaf

While it is customary to check up the setting of the timbers every few days by t tools usually employed by timbermen are spirit level, carpenter's square, plumb bo straight edge long enough to span 3 sets. The straigh[illegible]e is often made as one si framed triangle, the top of which is horiz when the b[illegible]h of the straight edge is inclination of shaft.

Horizontal alinement may be secured by making use of tacks placed as described transit alinement. In setting a sill, a string is stretched from a tack in the new tim one similarly located several sets above. The plate being placed is then wedged until the string is exactly over the tack in last completed set. Sill is leveled with level. The hanging-wall plate or cap is set by stretching string as for sill, and at th completed set drop a plumb lin[illegible] from the string and wedge the cap over until plum hits the footwall string.

Cylindrical shafts may be lined during sinking by lagging driven behind wood steel rings placed at vertical intervals of 4 to 6 ft (14), which in turn are kept in pla wood or steel distance pieces. Steel plates are also used. These forms of lining ser temporary support previous to placing permanent metal or masonry lining.

Swelling ground is difficult to hold unless provision is made for cutting away protr rock and easing the shaft timbering. This is best done, without interruption of hoi by making the shaft opening large enough to allow a 2.5 to 3-ft space outside of re shaft sets. Auxiliary or jacket sets (8) are then placed and wedged around regular Floors may be laid on jacket sets and lagging placed outside of them in the regular in some cases regular are also boxed in with p In space thus prov work of easing timbers proceed as required.

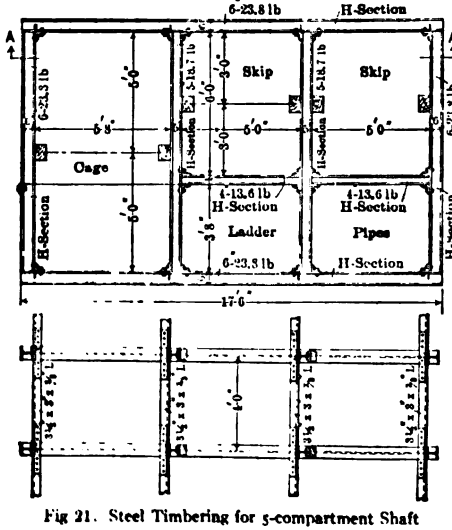


Fig 21. Steel Timbering for 5-compartment Shaft

15. Steel Linin

Steel shaft "timl ing" is often used ins of wood for both ver and inclined shafts All members used to f timber sets find t counterpart in steel bering.

Stresses in **wall** and plates are: **bending**, d side pressure of rock and compression, due t reactions of adjacent pl Stresses in **buntons** or d ers are: direct compre in case of vertical sh while in inclines or type of shaft in mo ground, the buntons also resist **bending.** or **studdles** are in d compression in vertical shafts but are subject to bending in inclined shafts. **Hence** plates and dividers, rolled shapes must be selected which have a comparatively large radius of gyration, and are equally strong about both axes of symmetry (Sec 43).

The H section is used extensively; it is adapted to withstand combined stresses, affords ample bearing against the sides of shaft. This shape has also been used for guides, cast-steel racks, sometimes bolted on the steel guides to engage the safety seem of doubtful value due to the heavy stresses and shocks. **Angles** are largely us

...and do away with necessity for permanent hanging bolts. I-beams are commonly ...or bearers. T-rails alone or reinforced with angles, and steel cross ties, have been ...wall and end plates; I-beams, double channels, and Z-bars for buntons; and rails ...where the weight of shaft is carried by lower bearers. Where there is enough ...below the lowest set to permit turning into position, each set may be riveted to... surface and lowered intact, chain blocks being used to swing it into place. If ...ary to carry timbering so close to bottom that there is insufficient room to turn the ...et, or where it is difficult to lower the entire set without interfering with dividers, ...be shop-riveted in parts and the underground assembling done with bolts.

16. Concrete Lining

Reinforced-concrete shaft sets, with members corresponding to those of ...sets, have been successfully used in both vertical and inclined shafts (11). ...vidual members are cast in steel or wooden forms.

...Iron Mining Co, Mich, used a 1 : 2 : 3 mixture, poured wet, in molding sets for ...shafts; Ahmeek Mining Co, Mich, used 1 : 3 : 5 mixture for plates and dividers, ...: 4 mixture for studdles of an 80° inclined shaft. After removal of forms, mem... allowed to harden for several weeks before placing. Reinforced concrete slabs, ...2 : 3 or 1 : 2 : 4 mixture, are molded for use as shaft lining between successive sets ...partitions between compartments. Where used as lagging between sets, the slabs

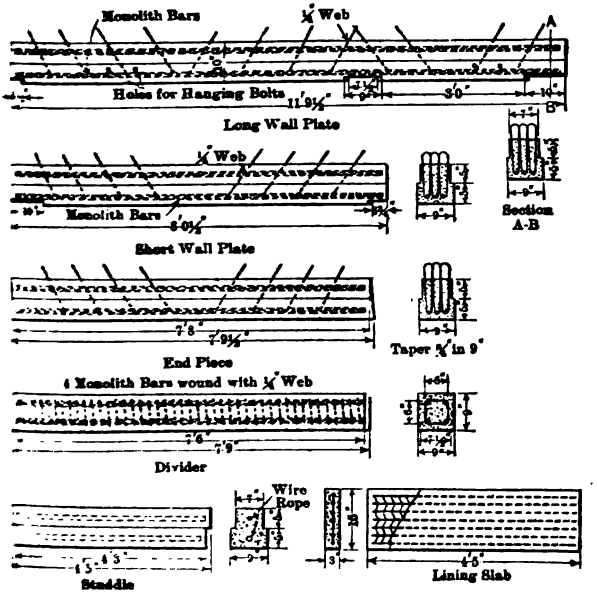


Fig 22. Reinforced-concrete Shaft Sets (43)

...ay rest on offsets provided on plates for this purpose, space between slabs and rock ...ang filled with broken rock or other material. Slabs used as compartment par... may be bolted to dividers.

...a set has been blocked into alinement it is sometimes tied to the rock by placing a ...ottom and filling around set with concrete. In some cases, members are molded ...forcing extending from the ends, which serves to bind the sets to a mono...

concrete lining poured after they are in place. Wall plates of long shafts, on acco their great weight, are sometimes made in 2 or more sections. Holes are cored to assembling and hanging bolts.

In a 3-compartment shaft of the Ahmeek mine (43), concrete sets (Fig 22) repla by 12-in timbers. Costs per set delivered at shaft mouth: timber, $37.60; co $22.50. Unit costs: timber, $28 per M; crushed stone, 35¢ per cu yd; sand, 60¢ per cement, $1.15 per bbl; reinforcement, $12 per shaft set. Labor of placing a concr is more than for timber because of its weight. 7 men placed one set per 9-hr shift.

Monolithic concrete lining (2, 11), both with and without reinforcin been successfully applied to inclined and vertical shafts of all shapes, thou flat inclines some reinforcing is needed.

Concrete lining may be designed to resist hydrostatic pressure and dam c water. Except in passing through a zone of surface water, however, it is ally best to collect water in rings (Art 12) if in moderate volume, and pu surface; excessive volumes are handled by grouting (Sec 8, Art 7).

Concrete lining is sometimes poured around steel or molded-concrete shaft s other cases, dividers of wood, steel or reinforced concrete are placed while pouring. neither sets nor dividers are employed, one or more reinforced curtain walls betwee partments may serve to stiffen the lining; these walls having openings at inter prevent injury from suction when hoisting. Bolts for guides are imbedded in th walls, or holes are cored in dividers and end plates to receive them. Concreting started at the bottom of a shaft; if so, temporary timbering is usually required for

Details of monolithic lining. After sinking as far as is safe, hitches ar bearing timbers placed, and a platform placed upon them. With the pla as a foundation, forms are placed, a temporary floor laid on top of forms, an concrete poured.

Mixtures from 1 : 2 : 4 to 1 : 3 : 6 have been successfully used, depending on m available and strength required. Concrete is mixed on surface and lowered in bu or delivered through a pipe, say 4 in diam. Buckets with distributing chutes at b place concrete directly in forms without shoveling; where delivered by pipe, it r received in similar buckets and placed as desired.

If the rock section has been broken to nearly correct size, inside forms alone are re If large cavities exist in rock walls, outside forms with back filling are necessary; o ties may be filled with dry rubble, which answers both as outside form and fillin such places it is customary to increase the thickness of w use steel reinforcing, or both. It is important that all inf water be cut off by the grouting, or piped through the forms concrete is placed, to avoid separation of cement from the Minimum thickness of concrete is from 8 to 18 in, dependi character of ground.

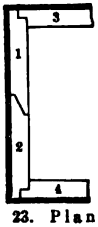


Fig 23. Plan of Wooden Form for Concrete Lining

Forms are of wood or steel, built in sections from 2.5 to high; aver about 5 ft. They should be designed to strip after concrete is set. If greased before pouring concrete, come away easier, and leave better surface. WOODEN FORM usually of 2-in plank dressed to uniform size, and tied togeth wood or steel bracing. Fig 23 shows a simple type (the nu indicate order of stripping). STEEL FORMS, of ⅛ to ¼-in p with angle bracing, are extensively used for circular and elli shafts (12.)

Reinforcing for concrete linings is commonly of old mine rails or hoisting rope; s ard steel shapes, or steel bars and expanded metal, are also used.

Concrete blocks have been used in Europe for shaft lining. An interlo reinforced block molded in Z-form has been successful in Belgium. After pl the blocks, spaces behind them are filled with concrete.

Steel and wood shaft timbering and lagging, covered with wire netting, have been c with a layer of concrete grout 1.5 to 2 in thick, applied by a cement gun under 25 to pressure. The grout serves to fireproof and lessen decay of wood and prevents cor of steel.

17. Masonry Lining

Materials. In Europe cylindrical shafts are common and masonry WALLING is usually used as lining, where the volume of water is not excessive. Good hard-burned brick, or durable stone or cement blocks laid with hydraulic cement, are the materials used. For small-diam shafts, brick should be shaped to curvature of shaft; for diam over 12 to 14 ft, ordinary straight brick are used. Cut stone is too expensive where other material is available. Where water under low head is to be dammed back, a special form of brick lining (COFFERING) is used (see below); for heavy pressures, C-I tubbing (Art 18).

Details. Shaft is sunk until strata is reached firm enough to permit placing foundation for lining. These foundations or WALLING CRIBS are usually spaced 30 to 50 ft apart. In very soft ground, a temporary lining (Art 14) is used. Diam of shaft is enlarged 2 to 3 ft to form seat for crib, this work being done with as little blasting and shattering of rock as possible. In firm ground, walling may be started directly on the rock seat. Usually the cribs are of wood or C-I, made in sections, and wedged or grouted into position. In very soft ground, the crib is laid on 1 or 2-in iron bars, driven into holes 2 to 3 ft deep around the circumference of shaft. Thickness of walling is usually 9 to 18 in. Space between lining and rock is filled with gravel, cinder, or concrete. Various types of curbs have been devised for placing brickwork, so that walling and sinking may proceed simultaneously (6).

Fig 24. Walling Crib and Water Ring Combined

Water rings. In wet strata weeping courses are left in lining, and water collected in wood or C-I water rings or troughs, which are often combined in one casting with walling crib (Fig 24). Above each water ring, the walling for a height of 4 or 5 ft is tapered back, so that 3 to 6 in of clear space is allowed for ring (Fig 25). Under the walling crib, a supporting shoulder of rock, 3 to 5 ft deep, is left before enlarging to full rock section. Great care should be taken in removing this shoulder when completing the next lower walling section.

Fig 25 Walling and Water Ring in Place

Coffering (1) is watertight lining formed by several concentric brick or concrete-block walls, with a space between filled with concrete, steel plates set in concrete, or clay. It must start from an impervious stratum and rest upon a watertight foundation or wedging crib, as used for tubbing (Art 18). There are usually 4 to 8 concentric rows of brick, often laid in stretcher courses. The bottom course of alternate rows frequently consists of split bricks, so as to break joints vertically. Hydraulic cement is used. Formulas have been derived for thickness of coffering (14).

Cost. English engineers place cost of WALLING CRIB at $25 to $50, and cost per ft of ordinary brick WALLING at $0.75 to $1.00 per ft of shaft diam; bricks at approx $5 per M. COFFERING costs more, depending upon its thickness and cost of enlarging shaft to admit extra thickness of brickwork. Costs in England for 20-ft shaft were $33 per ft; for a 16-ft shaft, $17 per ft.

18. Tubbing

Conditions for use are: a heavily water-bearing rock formation, underlaid by impervious stratum, which in turn lies above the mineral deposit. The quantity of water must not be too great to be pumped during sinking.

Construction. Tubbing is a watertight lining, usually of C-I rings, made in segments. The lowermost ring rests on a WEDGING CRIB, which makes a tight joint with the impervious stratum. Most of the ground-water is permanently shut out, thus greatly reducing the pumping cost during the life of the mine.

The wedging crib (Fig 26), of wood blocks, or better of C I, is made in segm and wedged tightly against the shaft walls (see Art 17).

The wedging crib seat is prepared by carefully dressing the rock around the shaft p eter. A gasket of thin pine boards, sometimes covered with tarred flannel, is laid b the crib is placed, or the crib may be set on a bed of fre poured concrete. After crib is in position, pine wedges driven filling space between it and the rock walls. Oak and sometimes follow the pine wedges, the steel being withdr and hardwood wedges driven into the cavities. The crib t to spring under the tension of wedging, and is propped d from the walls. Dry moss is sometimes placed in space bet crib and rock, and wedges driven into the moss. Two or cribs may be laid, one on top of the other. Instead of wedg the crib is sometimes secured and the joint made watertigh tamping concrete behind it and the 2 or 3 rings of tub next above.

Fig 26. Wedging Crib and Outside-flange Tubbing

There are 2 forms of C-I tubbing: (*a*) outside flange, (*b*) side flange. Tubbing with outside flanges and ribs is cas segments 1.5 to 3 ft deep, 8 to 12 segments to the ring. A in center of each segment facilitates handling and serve relieve water pressure during erection; the hole is subseque plugged. Gaskets of dry pine or sheet lead are placed between the segments, the jo of which are tightly wedged, after being placed in position. Tubbing with inside fla and ribs is cast in segments 2.5 to 5 ft deep. Flanges are carefully machined and bo together with lead gaskets. No wedging is necessary. The space between the tubl and shaft walls is filled with concrete.

Advantages of inside flanges are: (*a*) tensile strength is greater, as inside ribs ar compression; hence less weight is required, and deeper rings are possible, at less cost ft; (*b*) the wedging required with outside flanges throws strain on the inner edges of plate, whereas it is evenly distributed with bolted, inside flanges. Disadvantage of side flanges and bolting is that the rings are rigid, and shifting ground causes abnor strains in the lining.

System of suspended tubbing, used in Germany, requires no temporary timbering cuts off part of inflowing water during sinking (15).

Strength of watertight lining depends on the lining material, diam of shaft, and h of ground-water, which is usually taken as the entire height of tubbing above the *point* question. Stresses due to rock pressure and wedging are indeterminate; corrosion by a water is also an important factor. It is customary to use several graded thicknesses of t bing, from maximum at the bottom to a minimum at the top, based on hydrostatic press

A number of formulas have been proposed for thickness of tubbing. Lupton (17) gi for outside-flange tubbing for a depth of 200 ft: $x = \frac{D}{20} + \frac{1}{8}$ in, in which x = thickn in; D = diam, ft. This formula is used in connection with the following, good for any dep $y = x\sqrt{\frac{P}{200}}$, where y = thickness, in, for any depth other than 200 ft; x = thickness 200 ft, as found by first formula; P = depth, ft. Thickness should never be less than 0.5

For width of flanges Lupton gives the formula: $z = A\sqrt[3]{\frac{D}{200}}$, where z = width of flan in, for any depth other than 200 ft; A = width of flange, in, for 200 ft depth as given bel

Diam of shaft, ft	Width of flange, in	Diam of shaft, ft	Width of flange, i
Under 10	4.0	16 to 18	5.5
10 to 12	4.5	19 to 22	6.0
13 to 15	5.0	.....	.....

Thickness of flanges is same as that of the plates.

A German shaft, 16.6 ft diam, tubbed from 394 to 581 ft with flanges inside, was divid into 2 sections. Thickness of metal: bottom section, wedging crib 2.36 in, tubbing 2.16 i top section, wedging crib 2.16 in, tubbing 1.97 in.

Examples of English Shafts Lined with Outside-flange Tubbing (14)

Depth of section tubbed, ft	Thickness of tubbing, in	Diam of shaft, ft	Depth of section tubbed, ft	Thickness of tubbing, in
101 to 303	7/8 to 1 7/8	16	340 to 450	2
110 to 255	1 1/2	11	60 to 260	3/4 to 1 3/4
275 to 340	1 3/4	10	300 to 420	1 1/2 to 1 7/8

983. In England (6, 16), cost of tubbing installed ranges from $40 to $100 ... shaft; castings cost $25 to $35 per ton, and placing $6 to $10 per ton. ... it for a 14-ft shaft, were: 0.75-in tubbing, $44; placing, $5; gaskets, ... etc, $2. Preparing seat and placing wedging crib, $200.

18. Kind-Chaudron Process (15, 44)

Conditions for use: heavily water-bearing rock formation, with an impervious stratum overlying the horizon to be reached, similar to the conditions ... lining by tubbing (Art 18), but where the inflow of water is too great ... ped while sinking. The stratigraphy essential to the success of the ... is rare in the U S, but common in parts of Europe, where about 90 Kind- ... shafts have been sunk.

Method. The shaft is excavated by a boring process, under water, by means ... drop tools, called TREPANS, the method being an extension of the rod ... toring (Sec 9, Art 5, Canadian rig). The water stands at its natural ... l the work is finished. A small shaft may be bored first and then enlarged to full diam, or the full section bored in one operation. On completing ... seat is prepared to receive the bottom section of lining (tubbing), called ... SHOE, which automatically makes a tight joint with the underlying ... stratum. The tubbing is put together at the shaft mouth, ring after ... added as the entire lining is lowered through the water. When the ... in place, the space between it and the shaft ... with concrete, and the shaft is pumped ... The success of the process depends upon the ... tightness of joint between the tubbing and ... its stratum.

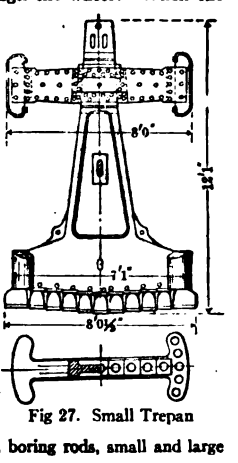


Fig 27. Small Trepan

... (15) says that sinking by ordinary methods is ... the Kind-Chaudron process for mean depths ... 100 ft, with less than 1 700 gal of water per ... mean depths less than 300 ft with less than 900 ... For depths greater than 500 ft, where water-... are at least 160 ft thick, and make more ... gal per min, the Kind-Chaudron process is ... the cheaper.

... here it is anticipated that this process will ... the shaft is commonly started by ordinary ... are being taken to keep the walls of com... clear of all projections which might ... by interfere with the passage of boring ... shaft were suddenly flooded.

... ed consists of a suitable headframe (with ... handling the heavy boring tools), power plant, boring rods, small and large ... bits and special tools.

... A shallow hole, the diam of small trepan (Fig 27), is made in the center of ... assist in starting the trepan, which cuts an advance center bore 4 to 10 ft diam. ... usually kept at a minimum distance of about 30 ft in advance of the enlarge-

ment, but it is sometimes drilled to the entire depth before enlarging begins. C from small trepan are removed by sludger (Sec 9), while those from large trepan are and periodically raised, in a large bucket suspended in the advance bore. Drillir are attached to rods suspended at surface from the walking beam by a temper scr 9, Art 5), which feeds down as shaft is deepened.

Trepans are massive steel frames, to the lower edge of which are at chilled-steel bits weighing about 100 lb each. Bits are placed unsymmet to cover entire area of shaft bot trepan rotates; they are also set t the shaft bottom towards cente facilitating removal of cuttings. T are inspected frequently and du removed.

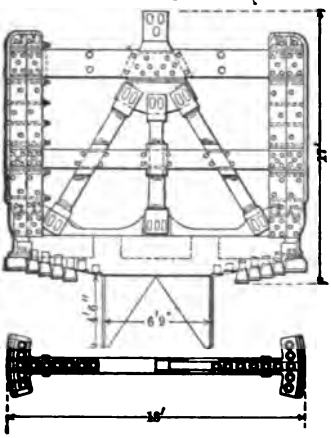


Fig 28. Large Trepan

Weight of Trepan: for small diar 2 to 15 tons; for large, 15 to 30 to enlarging trepan (Fig 28) has a project tion extending into advance bore, t keeping proper alinement. This pu also served by attaching to trepan head fitting in the shaft. These cro may carry bits to dress down the sha Strokes per minute vary from 8 to 25 of stroke, 6 in to 2 ft.

Temporary lining. If shaft wal badly, caving areas may be lined tem with sheet-steel casing, supported shoulder of rock left for the purpose pended from surface. Diam of sha be reduced below such temporary lini

Permanent lining. When sh been sunk well into the impervious stratum, the bottom is cleaned by a tool, and moss-box and tubbing lowered into place. The lining has inside (Art 18), and all flange faces are machined and packed with sheet-lead gaskets before being bolted together at surface.

The moss-box (Fig 29) is formed by constructing the 2 bottom tubbing rings so that they can telescope; the annular space between bottom ring and shaft wall is filled with moss secured to bottom ring by wire netting while lowering. When bottom ring reaches its seat, the weight of tubbing above forces second ring to telescope over the first, compressing the moss against shaft walls and forming a watertight joint.

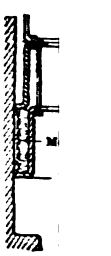


Fig 29. Se Moss-b

In deep shafts, the tubbing becomes excessively heavy; it may be rendered buoyant by inserting a false bottom or diaphragm, above the moss-box. In this diaphragm is placed a vertical EQUILIBRIUM PIPE, controlled by valves which admit ballast water as necessary. Section after section of tubbing is added, and the whole lowered until the buoyancy is greater than the wt of tubbing. Enough water is then admitted to sink the lining and compress the moss. Cement grouting i lowered in trip-bottom boxes, to fill the annular space between shaft walls and t When the concrete has set, the shaft is pumped out and the false bottom r Wedging cribs are placed below the moss-box and sinking is resumed by ordinary m

Modifications. The moss-box has sometimes been omitted, the concreting h pended on to make a tight joint. In this case, special bits are placed on the enlarg pan to cut a level seat for the bottom flange of the lining. A modification introd Lippman is to bore out the entire shaft in one operation, using a trepan of specia

Speed and cost. Aver speed of sinking in northern France is given (6 to 33 ft per month, to depths of 600 ft; diam, 8 to 12 ft; cost, $66 to $200

... sinking 14.5-ft diam shafts in the Ruhr district, Germany, to mean depths be... and 1 148 ft are given as follows (18):

... and equipment, 50% of first cost	$25 000 to	$35 000
...aneous	12 500 to	25 000
...g per ft	90 to	195
...ete per ft	12	
... and supplies per ft	60 to	105
...r per ft	135 to	210

... gives cost of sinking only, exclusive of tubbing, at $83 to $250 per ft; ordinary ... 9 to 12 in per day. In general, complete cost of 12-ft shafts ranges from ... per ft; 14 to 16 ft shafts, $250 to $500 per ft. The 1 335-ft Georg shaft, ...ianagia, is the deepest thus far sunk by Kind-Chaudron process.

SPEED AND COST DATA

20. Small Shafts

..., Mont. 4 by 6-ft prospect shafts, 15 ft deep, with no water, cost about $5 per ... depth of 7 ft; the upper 7 ft is paid for on basis of about $1 per cu yd. A 4 by ... 10 ft deep is generally accepted at Butte by U S mineral surveyors as costing $25. ...ures are based on miners' wages at $3.50 per 8-hr day.

..., Mo. 5 by 5-ft prospect shafts, with light cribbing, cost $3 to $6 per ft in loose ... hard, dry ground, $8 to $12. Presence of water adds $5 to $20 per ft. Aver ... to 120 ft.

...ka Co, Nev. Exploratory 2-compartment shaft. Following figures include no ... charges. Shaft was sunk to 900 ft and timbered to 875 ft.

... 5 by 9 ft; hammer drills; 2 8-hr shifts per day.

...r per shift:		Guides	4 by 4 in
...men	@ $4.00	No of holes per round	18 to 22
...man	@ 3.00	Depth of cut holes	6 ft
...neer	@ 4.50	" " breaking out holes	5 ft
...timberman	@ 4.00	Powder, strength	35%
...hoister	@ 3.00	Powder per ft advance	8.4 lb
...r per M	$32	Aver advance per day	2.54 ft
...ize	6 by 6 in		

Cost per ft

Sinking		Sinking		Timbering	
		Materials:		Labor	$ 2.84
...	$3.22	Drilling	$2.52	Materials	2.40
...	3 65	Shoveling	0.92	General expenses	0.20
...pening	0 26	Steel	0.03		
...	0.60	General expenses	0.95	Total	$17.59

..., Nev. Vertical shaft, 6.5 by 10.5 ft rock section, was sunk 100 ft using a wind... ...ng first 20 ft, after which a "whip" was operated by horsepower. First 30 ... gravel, remainder in firm soft shale, drilled with hand churn drills. From ... ft, 8 by 8-in timbers were used; below 25 ft, 6 by 8-in. No water encountered. ...ed 1 shift per day (19).

... shift, 4 men	$ 4.00	Powder per ft advance	3 lb
... per lb	0.18	Powder per ton rock broken	0.55 lb
... ton	20.00	Av advance per day	1.85 ft
... per M	42.50		

Cost per ft

...	$8.56	Cartage	$ 0.06
...	0.647	Sundry	0.028
...	0.544	Timbering	4.75
...	0.398		
...	0.042	Total	$15.02

21. Working Shafts, Metal Mines

Tamarack No 5 vertical shaft, Lake Superior copper district, was sunk through amygdaloids and conglomerates, dipping approx 38°. Dimensions, outside ti 29 ft 2 in by 8 ft 10 in, divided into 5 compartments. Surface force comprised la laborers, hoisting and compressor engineers, firemen, and blacksmiths, their n varying, depending on conditions.

Timber dimensions:	Sinking bucket capacity 1⅝s
Wall plates..... 10 by 14 in	Labor per shift, underground:
Dividers and end plates.. 10 by 12 in	(11 miners, 7 muckers. 5 timbermen, 1 pump tender)
Posts..... 8 by 10 in	2 8-hr shifts per day
Guides..... 5 by 7 in	Aver number holes per round 44
Drills, 4 Rand Sluggers	Aver depth hole..... 4 to
Size sinking rope:	Aver advance per round 5 ft
Surface to 3 500 ft..... 1⅛ in	
3 500 to 4 680 ft..... 1¼ in	

Speed and Cost

Depth sunk, ft	Aver advance per month, ft	Cost per ft, sinking and timbering	Depth sunk, ft	Aver advance per month, ft	Cost p sinking timbe
Surface to 140	28	.....	2 015 to 3 000	82.08	84 8
140 to 986	70.5	$83.13	3 000 to 3 818	62.5	105 8
986 to 2 015	85.75	86.14	3 818 to 4 580	63.5	132.1

Following items were included in cost: captain and assistant; timbermen; mi men; muckers; engineers and firemen; blacksmith; carpenter; machinists; team; der, fuse, caps; oil, candles, and wick; wire rope, hardware, and iron; pipe fittin machine parts; lumber, timber, and lagging; engine house supplies and fuel.

New Anvil vertical shaft, Bessemer, Mich, was sunk through hard rock havin ding planes parallel to long axis of shaft. Bonus was paid for all work in excess of per month. Following figures cover work from surface to 960 ft. Lining, concret

Rock section, 12 by 18.5 ft:	Time analysis	
8 hammer drills used	Work	% of t
3 8-hr shifts per day		
Aver number holes per round..... 40	Drilling.....	23 4
Aver depth of hole..... 4.6 ft	Blasting.....	9.6
Aver advance per shift..... 1.5 ft	Shoveling.....	57.9
Powder per ft advance..... 22 lb	Delays.....	9 1

Cost per ft for labor and supplies, sinking only

Labor, 9-men per shift.....	@ $3.50	$21
Powder.....	@ 11 5¢ per lb	2
Fuse.....	@ 3½¢ per ft	
Drill steel (estimated).....		
		$24
Total cost, finished shaft.....		$85

Corbin, Mont. A 3-compartment shaft was deepened 213 ft below 400-ft level th dark hornblende granite. Bonus paid to sinkers hastened work (23).

Labor per shift:			Cost per
Miners.....	$4.00	Wall plates..... 10 by 10 in	Labor ...
Engineers.....	4.00	End plates..... 10 by 10 in	Lumber.
Shaftmen.....	4.00	Dividers..... 8 by 10 in	Supplies..
Blacksmith ...	4.00	Rock section, 17 by 7 ft:	Power. ...
Helper.....	3.50	Drills..... 2 2½-in Sullivan	Total...
Carpenter.....	4.50	Powder, strength.... 40%	
Topmen.....	3.50	3 8-hr shifts per day	
		Aver adv'ce pr day 3.23 ft	

...chusa, Ariz. A shaft was deepened from 879 ft to 1 017 ft, through heavy water-... while the mine was in operation. Size, 8 ft 10 in by 23 ft 8 in, outside timbers. ...partments. Timbers, 10 by 10-in Oregon pine. Water, 2 200 to 2 400 gal ... following costs do not include hoisting.

Cost per ft (26)

.....	$3.39	Lumber.....	$12.85	Labor, miners..	$58.63
.....	0.18	Supplies.....	9.84	Labor, general..	1.68
.....	1.03	Compressor and drills.	4.78	Total.....	$92.38

...theastern Missouri lead district. Shafts in this field are in hard magnesian ... and are untimbered except stulls supporting cage guides and pipe lines. Fol... examples are all vertical shafts (20).

.....	6 by 18	18 by 20	10 by 20	7 by 10	6 by 14
.....	407	338	418	320	400
...ke per day, ft.....	1.69	2.25	2.32	2.66	3.33
... per min.....	450	.....	dry	dry	wet

Vertical Shafts, Wisconsin Zinc District (21)

	No 1	No 2	No 3		No 1	No 2	No 3
.....	6 by 9	5.5 by 9	5 by 11	Machine sinking, ft.....	.....	63	.....
ft	145	153	73	Hand sinking, ft	115	90	73
...dvance per ...	1 42	1.80	1.40	Cost per ft			
... per m'th	125.00	125.00	100.00	Labor.....	$20.50	$14.54	$14.05
... per day...	2.00	1.75	2.00	Coal.....	4.47	2.64	.....
... per day.	1.75	1.75	1.75	Explosives.....	1.33	1.10	0.99
...on ...	3 80	4.40	.....	Oil.....	0.05	0.08	.....
...ered, ft	19	30	30	Candles.....	0.10	0.09	0.06
...ft ...	65	85	73	Lumber.....	0.33	....	1.40
... per min.	1000	400	dry	Sundry.....	0.64	1.21	.....
... per lb	12.5	13.5	11.5	Total.....	$27.42	$19.66	$16.50
... strength ...	30	40	30				

... pumps used in sinking. Rock, limestone and shale.

... No 3 3-compartment shaft, on Mesabi range, Minn (39), was sunk to a depth ... months, through broken taconite, paint rock, and solid taconite. Water to ... was pumped. Lined with steel sets 6 ft by 18.75 ft inside dimensions. ... plates were 5-in 18.7-lb H-pieces; dividers, 10-in I-beams; sets spaced 4 ft ... held together by 8 3.5 by 3.5-in angle studdles. Lagging, 2-in pine, held in ... by 2-in angles, shop-riveted to back of wall and end plates.

... boiler house expense, $28.20; pumping, $12.67; labor, $58.39; explosives, $4.17; ... $.17; miscel, $17.96; steel sets, $14.73; total per ft, $134.49.

Lucius shaft, Globe, Ariz (11), 1 017 ft deep, 12 ft 4 in by 8 ft 6 in in rock section, ... of partly burned timbers, relined with concrete of 8-in minimum thickness, ... by an 8-in reinforced curtain wall. Contract price, $28 per ft.

...ton shaft, Bisbee, Ariz, 27.25 by 6 ft in clear, 1 536 ft deep, was lined with ... reinforced 3 curtain walls. Cost of concreting for 9 months, ended Aug 24, ... laborers (11). Supervision, $1.40 per ft (approx).

Monthly cost per ft

...onth, ft...	60	140	170	185	180	200	170	235	196
...ering pipe	....	.....	.....	$ 0.07	$ 0.10	$ 0.15	$ 0.12	$ 0.01	$ 0.00
...rs	$ 3.24	$ 6.29	$ 4.72	4.05	2.93	2.15	2.25	1.60	1.85
.....	53.83	43.52	26.00	30.31	21.33	23.68	24.93	18150	19.66
...rs	27.17	16.37	11 63	10.18	8.93	8.57	8.20	7.11	7.00
...crete.....	9 20	6.64	4.06	3.17	2.77	2.25	2.73	1.90	2 50
.....	$93 41	$72 82	$46.41	$47.78	$36.06	$36.80	$38.23	$29.12	$31.01

Argonaut, Cal. Following data are based on deepening a 60° inclined shaft from 3 492 to 3 675 ft, while mine was in operation. Rock penetrated was a s schists and slates, dipping with the shaft but at a steeper angle; slates were soft small, caved, and swelled on exposure; schists were hard and broke large. E operated air hoist at bottom level of mine; boss timberman supervised placing sets. Powder, 40% gelatin, with a few sticks of 60% at bottom of holes. Bonu per ft was divided among crew for all sinking in excess of 50 ft per 31-day month. below do not include drill repairs, steel, tools, or overhead charges (25).

Labor per shift:		Posts	10
2 machine men	@ $3.00	Timber, per M	$21
2 helpers	@ 2.75	Powder, per lb	$ c
1 engineer	@ 3.00	Power per hp month	$ 4
3 8-hr shifts per day		Size of shaft, over all	18
Drill, Sergeant	2¾-in	Aver advance per day	1 8
Air press	80-90 lb	Aver advance per round	5-6
Wall plates	20 by 20 in	Aver number holes per round	32
End plates	14 by 20 in	Depth of holes	6-9
Dividers	10 by 20 in	Powder per ft advance	34

Cost per ft

Drilling and mucking	$19.94	Fuse	$0.48	Track	
Engineers	5.54	Caps	0.10	Power for hoist	
Timberman	0.78	Candles	0.56	" " drills	
Powder	3.74	Timber	8.13	Total	

North Star shaft, Grass Valley, Cal, in its main inclined section, dips at ab Ground is hard granodiorite, requiring little timbering. Following data are bas 5-month period of sinking below the 4 000-ft level. Machine men made up an in drilling or mucking by working overtime. Four 1-man 2¼-in Torpedo dri mounted on 2 bars. In hard ground, the holes were shorter than 6 ft (28).

Labor, day shift:		Size of shaft, over all	18
3 miners	$ 3.00	Aver advance per day	3
2 trammers	2.50	Aver advance per round	3
Labor, night shift:		Aver number holes round	22
4 machine-men	4.50	Aver depth of hole	6
Timber per M	20.00	Water per min (est'd)	15
Powder 40%, per lb	0.12		

Cost per ft

Labor	$ 8.90	Timbering	
Powder and candles	2.98	Hoisting	
Drills	2.90	Blacksmiths and mechanics	
Tramming	8.25		
Tools	1.41	Total	

Giroux shaft, Kimberley, Nev (22), was raised 948 ft from the 1 000-ft le joined to a shallow pit already completed. Raise was carried at full cross sec After the 770-ft level was reached, operations were conducted from there. R divided into 4 compartments, 2 chutes, cageway, and ladder and pipe way. Pe shaft timbering was installed subsequently, 75% of the temporary raise timb salvaged.

Rock sec of raise, 13 by 21 ft		Aver no holes per round	
Drills, 2 Waugh stopers		Aver depth per hole	
Aver advance per day	5.27 ft	3 8-hr shifts per day	

Cost per ft of raise

Labor:		Labor:	
Raising	$16.97	Supplies	
Tramming	5.09	Hoisting	
Timber framing	1.76	General	
Blacksmith	1.19		
Machine shop	0.11	Less 75% timber recovered	
Teaming	0.54	Total cost per ft	
Air compressor	10.00		

Negaunee No 3 shaft, Negaunee, Mich (38), 17 ft diam, was begun by an 8.5 by [illegible] from the 806-ft level to top of rock, 68 ft below surface. Raise was then [illegible] to full section, concreted, and steel dividers placed. Following costs per ft are [illegible] ft of rock section; raising, $18.68; enlarging, $17.33; steel dividers, $10.06; [illegible] concrete forms, $1.26; temporary plant, $6.74; concrete, $17.22; compressed air, [illegible] per ft, $72.17. Aver thickness of concrete, 16.2 in.

Gordon shaft (45), Ducktown Sulphur, Copper and Iron Co, Tenn, was excavated [illegible] of 1 175 ft in comparatively dry, very hard, silicified mica schist, by a com- [illegible] of sinking, raising and reaming. Size of shaft, 7. by 14 ft. Shifts per day, 2 [illegible].

[illegible] ft of shaft was made up as follows: collar 8 ft above surface; 63 ft reamed [illegible] old shaft and 99 ft sunk. Upper 84 ft were timbered and lagged; shaft sets, 12 [illegible] dividers, 8 by 10 in; sets, 5-ft centers. Rest of shaft was in solid ground and [illegible] omitted; dividers and end plates necessary to carry guides were set in hitches. [illegible] 170 to 629 ft, drifts were run from 5 levels to line of shaft and a raise, about 5 ft [illegible] was started at 5 points, sinking being carried on simultaneously from surface. The [illegible] were subsequently enlarged to full section. Shaft was put in operation to a depth [illegible] (including 15-ft sump) in 13 months; approx monthly aver, 49 ft. Drills per [illegible], 2 hammer stopers; reaming, 4 3-in reciprocating. Number of men per [illegible], 2 drillers, 1 nipper, trammers as needed; reaming, 8 drillers.

[illegible] to 860 ft, the shaft was sunk, as there were no workings beneath to permit [illegible]. Although every facility then available was employed, the sinking and equipping [illegible] section proceeded at an aver monthly rate of less than 17 ft. Drills per crew, 4 [illegible] reciprocating. Number of men per crew, 1 boss, 8 drillers, 1 hoistman, 1 lander.

[illegible] 860 to 1 175 ft, the shaft was sunk with 6 jackhamer drills, and with increased [illegible] and hoisting facilities. A maximum of 80 ft was sunk in a month of 47 9.5-hr [illegible] the aver speed being 63.5 ft per month. The 300 ft of divider timbering and hitch [illegible] (5 ft of the 50-ft sump being untimbered) was done in 1 month.

[illegible] costs apply to first 860 ft.

	Total cost	Cost per ft	Aver advance per mo, ft
[illegible] ground and temporary structures	$ 760.03	.....	.....
[illegible]	5 005.48	.....	.....
[illegible] frame	2 245.01	.....	.....
[illegible] plant	20 607.70	.....	.....
[illegible] general equipment (*a*)	$28 618.22	$33.28	.....
[illegible], 358 ft	17 691.11	52.38	31
[illegible] 412 ft	6 877.40	16.67	34
[illegible] 475 ft	12 905.86	27.18	90
[illegible] 9 stations	15 495.57	.....	.....
[illegible] hitches in 774 ft of shaft	3 285.76	4.24	.....
[illegible]	9 645.71	11.18	150
[illegible] and aver (including stations)	$94 519.63	$109.90	.....

(*a*) [illegible] equipment largely sufficient for shaft when sunk to depth of 2 700 ft.

22. Working Shafts, Coal Mines

[illegible] River, W Va. Elliptical shaft, 180 ft deep, 17 ft 4 in by 33 ft in clear, was lined [illegible] aver thickness 1 ft (37).

Costs per ft of Concrete Lining

[illegible]	$ 5.90	Forms: Lumber, $13 per M	$ 1.83
[illegible]	1.77	Making, $21 per M	2.95
[illegible]	19.18	Placing	4.83
[illegible] Mixing	3 83	Platform for starting	0.92
[illegible]	3.40	Superintendence	3.03
[illegible] and pumpman	2.19	Plant	0.28
		Oil, 0.21; tools, 0.24; sundry, 1.06	1.51
		Total per ft	$51.60

Pocahontas colliery shaft, W Va, 14 by 22 ft, was sunk 180 ft through hard sand carrying little water. 3 8-hr shifts per day underground, 2 12-hr shifts on surface

Wages: Foreman	$3.00	Coal per ton	$ 1.25	Cost per ft	
Drill runners	2.50	Timber per M	20.00	Installing tem-	
Helpers	2.25	Wall plates	8 by 10 in	porary plant	$
Muckers	2.00	Dividers	6 by 10 in	Coal	
Blacksmith	2.75	Lagging plank	2 in	Powder	
Engineer	2.00	Aver advance per day	3.33 ft	Labor	
Compressor man	2.00			Timber	
Carpenter	2.50			Sundries	
Helpers	1.75			Total	$

Cost of Shaft Sinking in Pennsylvania Anthracite District

Size, ft	Depth, ft	Date	Cost per ft, $	
14 by 47.3	1 175	1912	56.67	Sinking only
15 by 50	1 650	1912	150.00	Soft ground (*a*)
14 by 37.2	596	1905	115.70	(*b*)
14 by 24.2	952	.....	65.06	(*c*)
15.2 by 33.5	1 009	1911	54.28	(*d*)
14.2 by 40.5	1 550	.....	73.03	Sinking only
14 by 18	1 000	1905	45.70	.....

(*a*) Aver advance p 46 ft. (*b*) Cost dis tion: labor, $54.75; su $26.25; temporary $13.45; timbering, concreting, $13.50. (vance, 60 to 90 ft per m Cost distribution: si $57.73; preliminary, concreting, $1.55; t rary plant, $4.47. (*d* thro' mine dump and @ $82.50; 675 ft con $50; 277 ft contr'd @

23. Rand Shafts

Cost and Rate of Sinking Vertical Shafts on the Rand (30)

Name	Size rock section, ft	Total depth, ft	Rate per month, ft	Cost per ft, $	Name	Size rock section, ft	Total depth, ft	Rate per month, ft	C p
Rose No 1	23 by 8	911	....	151.18	Vogelstruis	18 by 7.5	1055	89	9
" No 2	18 by 8	714	....	130.50	"	18 by 7.5	960	86	7
Glen No 1	18.5 by 8	1005	91	108.08	Knights Deep	28 by 8	1244	55	17
" No 2	18.5 by 8	1017	66	117.48*a*	" "	28 by 8	1217	40	23
Durban Roodepoort	18 by 8	1444	87	95.00	Simmer East " "	28 by 8	1925	96	12
Robinson No 1	22 by 8	2389	69	125.20*b*		28 by 8	1802	70	14
" No 2	18.5 by 8	1923	69	117.66*b*	South Rose	28 by 8	2392	78	10
Vogelstruis	18 by 7.5	891	65	100.73*c*	Simmer West	28 by 8	3408	130	10
					Jupiter	28 by 8	3752	135	10

Costs given include overhead charges. (*a*) Mostly in dyke. (*b*) Hand drilling; includes deprec on sinking plant. (*c*) Hard dyke from 250 ft to bottom. (*d*) drilling. (*e*) Very wet.

Government Gold Mining Areas, Rand (35). Southeast 7-compartment shaf sunk 233 ft, from 1 738 to 1 971 ft, during March, 1912. Rock: 155 ft quartzite, shale. 7-lb hammers were used in double-hand drilling (benches drilled single-han Ventilation by brattice. Water was hoisted.

Labor per shift:		Depth holes	3 5
1 White foreman		Aver advance per mo	194.3
1 White assistant		Water, gal per min	21.0
82 Native drillers		Timber, pitch pine:	
4 White timbermen (*a*)		Wall plates	9 by 9
11 Native assistants		End plates	9 by 9
Size, rock sec	45 by 10 ft	Dividers	7 by 9
Number holes per round	40 to 45	Guides	4 by 8

(*a*) Timbermen worked 1 shift per day, drillers 3 8-hr shifts. (*b*) Aver of Jan, March, 1912.

Brakpan mines (31). Two vertical shafts were sunk in 1905–1907; drilling by hand; 12 by 9 ft, rock section. Ventilation maintained by brattices, no mechanical assistance to air currents being necessary below 300 ft. Timbermen worked morning General mine force, carpenters, blacksmiths, etc, were used as needed.

per shift:	Timber:	
...ver...	Wall plates...	9 by 9 in
...ers...	End plates...	9 by 9 "
...ermen...	Corner posts..	8 by 8 "
...ing en-		
...er...	Studdles.....	12 by 4 "
...er...	Dividers.....	7 by 10 "
...er...	Brattice.....	6 by 1¼ "

...plete Cost per ft of Brakpan Shafts

	No 1	No 2
...g	$ 82.86	$ 73.76
...ring	11.70	15.40
...ar	56 03	55 73
...e handling	6 56	7 35
...ng and bailing	5 91	7.77
...st per ft	$163 06	$159.75
...ft	3 129	3 777
...dvance per mo, ft	126	152
...er per ft adv, lb	32.91	.....

Supplies, No 1 shaft	Cost	Cost per ft of shaft
Blasting gelatin, per lb	$0.211	$6.96
Caps, per M	5.770	0.21
Fuse, per M ft	4.709	2.14
Drill steel, per lb	0.070	1.48
Shovels, each	0.745	0.15
Picks, each	0.545	0.06
Hammers, each	0.427	0.11
Sundry tools	.....	0.29
Carbide, per cwt	7.490	0.32
Paraffin, per gal	0.285	0.17
Sundries, lamps, etc	.....	0.24
Shaft sets, per M	106.12	17.91
Blocks, lagg'g, wedges	.....	5.20
Brattice, per M	173.28	0.86
Guides, steel	.....	10.40
Hanging bolts, per set	14.00	2.02
Steel and iron, per lb	0.045	0.46
Sundries	.....	0 54
Total cost supplies per ft		49.52

Brakpan compound shaft turns from vertical to incline at depth of 3 096 ft. In June, ... incline was advanced 223 ft, from 3 453 to 3 676 ft, and in August, 261 ft, through ...gh shale. Sills placed at 7-ft centers were the only timbers used.

... rock sec	19 by 7 ft	Aver no holes per round	30.6
... of incline	13°	Aver depth holes	5.06 ft
...lb. 6 Holman	3¼-in	Powder per ft advance	25.5 lb

New Modderfontein circular shaft (34), 18-ft diam, was sunk 2 258 ft at an aver speed ... ft per month. Drilling was by hand and small machines; 32 hand holes breaking ... Lining: concrete collar, 36 in thick; from collar to 100 ft, ordinary brick 2 ft ... 100 to 600 ft, brick 1 ft thick. Below 600 ft concrete blocks were used. Cost per ...clusive of general charges: sinking, $70.41; lining, $37.02; total, $107.43.

New Kleinfontein, East shaft (33) was sunk at a pitch of 30° through slates. Sills ... the only timber used. In May, 1903, the shaft was sunk 213.5 ft. Following data ... months' work:

... rock sec	6 by 21 ft	Aver advance per round	5.95 ft
...er holes per round	28	Gelatin per ft advance	24.36 lb
...depth hole	7.18 ft	Aver advance per mo	171.6 ft

Cost per ft

...ges	$18.67	Traveling	$ 0.02	Dog spikes	$ 0.04
...	1.56	Candles	0.75	Fish plates	0.21
...ages	6 75	Caps	0.10	Nails and screws	0.04
...d expense	3 50	Fuse	0.24	Oils, grease	0.04
...	0.62	Blasting gelatin	8.30	Pipe	0.12
...ance	7 67	Iron and steel	0.29	Tools	0.07
...ials	14.87	Rails	1.83	Timber	0.40
...ing	0.24	Sleepers	0.10	Sundry supplies	0.51
...portation	0.08	Bolts and nuts	0.10	Total cost per ft	$67.12

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SECTION 8

SHAFT-SINKING IN SOFT WATER-BEARING SOILS

BY

FRANCIS DONALDSON, M.E.

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NOTE. — Numbers in parentheses in text refer to Bibliography at end of the section.

1. Difficulties in Sinking in Soft Ground

Underground water is the prime cause of the special difficulties met in sinking through workable soils, consisting of sand, gravel, mud, or a partially cemented mixture of these. Some pumping capacity is usually required at once, for it is impossible to check inflow by sealing fissures with cement grout, as can be done in water-bearing rock. The water must either be pumped continuously, or excluded permanently by a lining strong enough to resist external pressure corresponding to head of water or of semi-fluid material. The cost of such lining, expense of pumping during construction, and general engineering difficulties involved in sinking under these adverse conditions, have led to the invention of methods for excavating and sinking the lining WITHOUT PUMPING. The ground water is allowed to stand at its natural level, and all excavation is done under water. Finally, the lining is sealed to underlying rock, or other impervious stratum; after which the water is pumped out, and sinking is continued by ordinary means.

Running ground is a term applied to unstable materials, ranging clean dry sand to semi-fluid silt. Presence of water increases instability, is a more serious obstacle to sinking than the water itself. In fine, sand, a relatively small quantity of water acts as a lubricant, converti material into a slow-moving fluid which will penetrate the smallest leaks lining, will flow under and around lower edge of lining and rise in the but can not be pumped. Continued removal of such material from shaft slips and falls in the ground outside the shaft, and produces irregular a cessive stresses in the lining, often distorting it and causing surface subs around shaft mouth. In some cases this subsidence has affected an a more of surface, wrecking neighboring buildings and foundations. Her preparing to sink through unstable ground, the possible result of subs must be considered in laying out the plant and its foundations. Boilers pressors, and engines should be set well away from shaft mouth; derrick have a boom at least 60 ft long and should be erected on a spread foun which can readily be jacked up. If a headframe be used it should be mo on long trusses spanning the shaft; independently supported derricks a ferred to headframes for soft-ground work.

Boulders in soft ground cause much trouble, and their probable presence or a often determines method of sinking to be adopted. When sinking in running gr is always necessary to force down the lining somewhat in advance of excavat minimize flow of material into shaft; this may be difficult or impossible, if a bou imbedded in the ground directly under edge of lining. As an attempt to dig do remove the boulder is liable to result in a violent inrush, it is usually necessary t the boulder in place.

External pressure, which affects the design of shaft lining, depends on: depth cavation, nature of ground, and quantity of water. Where seepage is to be allo drain into the shaft, the lining at any given depth must be strong enough to re corresponding hydrostatic pressure. In heavy, running ground this pressure may exceed hydrostatic pressure, since it is due to a depth of fluid composed of water an or mud, of greater specific gravity than that of water alone. The lining must als stand severe localized stresses caused by irregularities in sinking.

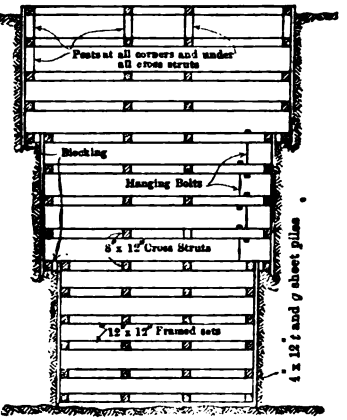
Methods of sinking in soft, water-bearing soils: (*a*) Linings construc the shaft as excavation progresses, including various types of timbering, poling and spiling, and the shield method. (*b*) Linings constructed at s and sunk as excavation advances, including open and pneumatic caisso "drop shafts," sunk by their own weight or forced down. (*c*) Solidificat the material in advance of sinking, including the freezing process for sand, and the cement grouting process for wet, seamy rock.

2. Timbering and Piling Methods

Characteristics. To resist normal earth pressures to depth of, say, requires merely the setting of sufficiently heavy timbers close enough tog but in unstable ground, to build a timber lining strong enough to resist dencies to distortion, due to shifting and falls of soft material behind the li is practically impossible. The solution of the problem is to prevent in that cause falls, by driving a continuous row of sheet piles or poling b around area to be excavated, in advance of excavation, or by forcing do shield. The tendency of the surrounding material to flow under lower ed lining into the shaft is thus checked by the weight of material inside.

Limiting conditions under which timbering is applicable in soft ground are: (*a*) of shaft at which external pressure is not greater than a practicable timber *linin* safely carry; (*b*) soil which permits piling to be driven far enough ahead to prevent in at bottom, but, at the given depth, will not exert enough fluid pressure seriously to d

...ing by bending lower ends inward. The behavior of different soils is described in ... given later. To a depth of about 80 ft, timber lining is cheapest; but, since ... lacks durability, a permanent shaft in soft ground must be excavated of large ... section to include a masonry lin... adding added cost.


Posts at all corners and under all cross struts
Blocking
Hanging Bolts
8" x 12" Cross Struts
12" x 12" Framed sets
4" x 12" 6 and 9' sheet piles
Fig 1. Vertical Sheet Piling

Vertical sheet piling, supported ...ntal sets of framed tim... is common for shallow shafts ... In starting a shaft two sets ...ted, one 5 or 6 ft above the ...; these serve as a guide frame for ...heeting, which is driven around ... The top soil is usually firm ...ugh for these sets to be placed be... surface in a temporarily unsup... soil; if not, they are set above ... with outside waling pieces ... through sheeting to top set, ... sheeting in line. In non... ground piles are driven as ... soon progresses. Sets are sep... by posts, and are usually tied ...ther with vertical rods. In very ...y and the whole lining must ...ended from trusses bridging shaft mouth. The bottom is kept level and ...nal sets are placed as depth is gained.

... must be driven well ahead all around the shaft. To keep them in line, they are ... Fig 2), or tongued and grooved (Fig 3). Wakefield timber piles (Fig 4), or inter-

Fig 2. Splined Sheet Pile

Fig 3. Tongued and Grooved Sheet Pile

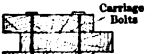
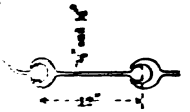


Fig 4. Wakefield Sheet Pile

... steel sheet piles (Fig 5 and 6) may be used; these also form a nearly water-... If rock lies too deep to be reached with a single course of piling, the shaft



... Carnegie Steel Sheet Pile

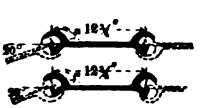


Fig 6. Lackawanna Steel Sheet Pile

... started of large enough section to allow for two or ... successive courses, the second erected inside the first ... The loss of area is roughly 20% at each step. As ...ween courses are weakest places in the lining, and the ... of lower course is troublesome because of the cross ... courses above, the number of courses should be mini... using long heavy piles that will stand hard driving. ... 3 courses should never be attempted. The lower ... are made to present a continuous driving edge by using ...-shaped piles under cross braces (Fig 7). In ground ...ing few boulders, 4 by 10-in tongued and grooved piles can be driven or jetted in ...ths, making a net depth, including necessary lap, of about 12 ft per course

Fig 7. Detail of Joint of Inner Course of Sheet Piling under Cross Struts of Fig 1

Splined 6 by 10-in piles can be driven in 26-ft lengths by using a pile driver (pref steam hammer), making net depth of about 20 ft per course; but at this depth tion of lower ends and consequent leaks in lining are probable. When, after sinkir larly, a pile suddenly stops, presence of a boulder is indicated; the adjacent pil then be driven home, and boulder dug out or broken up at once. If ground run and yet contains many boulders, no type of sheet piling will give good results.

Interlocking steel piling. Among well-known designs are those of linger, U S Steel Corp (Carnegie) (Fig 5), Lackawanna Steel Co (Fig 6 Friestedt. Some are of channels and angles, others of special shapes. I as to cross-sections, sizes, and weights can be obtained from makers' catal Interlocking 12-in steel piles can be in lengths of 40 to 50 ft without sepa and a depth of 75 ft has been reache courses (see Example III, below).

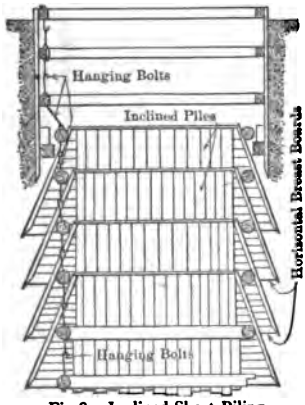


Fig 8. Inclined Sheet Piling

Inclined piles or poling boards (are sometimes used in deeper sha avoid sacrificing area. This is a mo tion of forepoling, common in tunneli ground (Sec 6, Art 19), but is not adapted to shafts. Having carried the shaft with vertical sheeting unt ground is reached, a set of timbers s than those above is so placed on the b that a uniform space, slightly wide thickness of proposed poling boards, between the backs of these timbe the vertical sheeting. The points of boards are then entered into this spac the boards driven on an outward slant. Length of boards must be mor double the out-to-out spacing of timber sets; when driven home they bear a outer edge of bottom set of timbers and inner edge of set above. The completed course thus forms a flaring wall, within which another set of timbers can be placed without reduction in shaft section. Especial care is taken in excavating at corners and opposite cross braces, where gaps are necessarily left to be closed by short horizontal BREAST BOARDS. The timber sets are tied together with hanging bolts, and the whole structure may well be suspended from a surface truss or heavy collar set. A modification is shown in Fig 9. Inclined piling is adapted to hand work, but is not suitable for very soft ground nor great depths, owing to difficulties of keeping tight a lining with so many joints, and the multiplied stresses in timbers, due to lever action of poling boards.

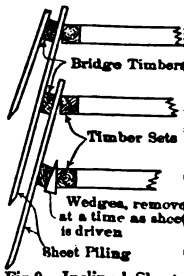


Fig 9. Inclined Sheet Modified

Shields. In very soft ground, iron shields with doors, that can be opened to excavation or closed to stop an inflow of material, are sometimes used (Fig 10). are forced down with jacks bearing against the lining, and are provided with a v tail plate inside which the lining is built up, usually of heavy horizontal plank la wise. The lining is tied together by spiking on vertical planks, and to prevent ment it is usually suspended from heavy beams or trusses across collar of shaft; shoe of the shield must also be suspended, so that, if one end hangs on a boulde other is in soft material, it will not be distorted or fall away from lining. Depth is l usually to about 80 ft, since below that the upward pressure of running ground

...ack a shield against. Fig 10 shows method of hanging shield from a suspension ... is advantageous also with other types of timber lining in bad ground.

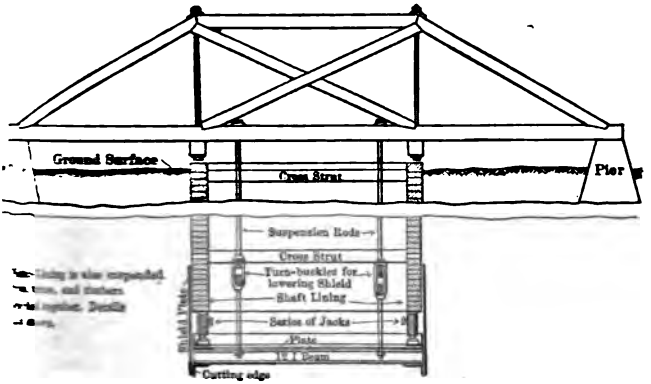


Fig 10. Shield Suspended from Truss

Excavation. Boom derrick, equipped with double-drum engine, has super- ...headframe for sinking in soft ground, either by timbering or by open or ...caisson methods, because of its lightness and flexibility. The boom ...raised or lowered to cover entire area of shaft, a useful feature in placing ... Boom should be at least 60 ft long, and derrick set well back from ...and so constructed that it can be readily raised if ground settles. Ma- ...usually dug and loaded by hand into shaft buckets (Examples I and ...); but in large shafts, where piling is driven well ahead, and in open ... an orange-peel or clam-shell bucket may be economically used (Ex- III). These require a third drum on the engine. While removing ... from beneath points of sheet piles, or cutting edge of shield or cais- ...must be taken to prevent an inrush of soft ground. It is often neces- drive short sheet piles around a boulder projecting into shaft, until its ...be reached, and then to break it with light charges of dynamite. In ..., suction pipe should be kept well below general level of excavation, ...t be necessary to drive a small sheeted sump in advance. In several ...quicksand has been successfully drained by driving into it perforated ... to which the pump suction was attached. On completing a stretch of ...t is essential to its stability that all water coming through it shall be ...clear, carrying no sand or mud. This is best attained by liberal use ...rammed behind the lining and into every crack and crevice. Sealing a ...lining to rock is not difficult, as the piles, if driven home, fit them- ...irregularities of rock surface. Subsequent excavation of the first 10 ... of rock must be done with extreme care; the first cut should be line- ...channeled entirely around the shaft.

Examples

Shaft 5, Wallkill siphon, Catskill aqueduct, was sunk through 40 ft of wet silt ...sand with 3 successive courses of 3 by 12-in by 16-ft tongued and grooved yel- ...plank. Top course, 19 by 30 ft outside; second, 16 by 27 ft; third, 13 by 24 ...shaft, 9 by 20 ft, inside timbers. Material excavated by hand and raised ...ck. A serious run of soft ground occurred under first course of sheeting, which ...aft out of line and made subsequent steps slower, more difficult, and expensiv

Average progress, 9 in per day. Cost per ft: timber (1200 ft b m), $30; labo supplies and miscellaneous, $24; total, $200. (Compare Example I, Art 3.)

II. Shaft 5, City tunnel, Catskill aqueduct, was sunk through 30 ft of fir bearing sand, close to brick conduit of old Croton aqueduct, and great care was t to prevent undermining the latter. Finished shaft is 14 ft diam, with reinfor crete lining, 2 ft 6 in thick, through soft ground. One course of 4 by 12-in tongued and grooved sheet piling was driven; supported by 5 sets of 12 by 1 bers, 35 by 35 ft over all, braced diagonally across corners to leave unobstructe onal opening. These outside dimensions were adopted on account of the shaft c A second course of 12-in U S Steel Corp sheet piling (Fig 5) 16 ft long, was d rock; supported by 6 octagonal sets of 10 by 10-in timbers, 20 ft 8 in across and for added stability. Material excavated by hand and raised with derrick; pilin with steam hammer. Some trouble was experienced by the separation of lowe steel piling, due to boulders, and an oxy-acetylene torch was used to cut off sev which projected into shaft. Progress, 9 in per day. Cost of first lift per ft: tim piles, $37.50; labor, $97.50; supplies and miscellaneous, $20; total, $155. second lift: timber, $8; piles, $35 ($15 allowed for salvage); labor, $115; supp miscellaneous, $25; total, $183 per ft.

III. Shaft for pier foundation, Tunkhannock viaduct, (21) was sunk through 75 ft of earth and sand by 2 courses of Lackawanna Steel Co sheet piling The 30-ft pilings for first course were assembled around a rectangular timber fra driven successively 2 or 3 ft at a time, until they reached their full depth, materi excavated meanwhile with derrick and grab bucket. Sets of 12 by 12-in timl placed as excavation advanced. A course of 45-ft piling was then assembled a frame concentric with first course, and 4 ft 8 in outside of it, and driven grad about 15 ft. The earth between it and the inner course of sheeting was ex outer course being braced against outside of inner course and by timber sets ab of inner course. Driving the inner course was then resumed, the sets of interior being shifted downward as excavation advanced, new sets of bracing for outer cou added above top of inner course. Thus the inner course was driven to rock a course to its full depth, without interference between the two sets of bracing.

3. Open Caissons (Drop-shafts)

Characteristics. Strong heavy caisson, of inside dimensions equal greater than those of proposed shaft, is constructed on surface, and s excavating within. This caisson or lining may be circular, rectangular, other desired shape. The first section is built on a shoe or cutting ed when sunk to nearly its full depth another section is started on top an ing is resumed; the process is continued until cutting edge reaches rock VANTAGES: (*a*) lining is built above ground, where construction is cheap easiest; (*b*) since weight is desirable in sinking, a permanent masonry can be built at once, of sufficient strength to resist any probable stress; cavation is done in a comparatively open space, unhindered by cross likely to be struck or knocked out by a bucket, and consequently wi risk to workmen; (*d*) completed lining is waterproof; (*e*) weight and s of lining are sufficient to crush or push aside boulders that would stop an of sheet piling. DISADVANTAGES: (*a*) under certain conditions it is diffi keep caisson vertical; (*b*) skin friction increases rapidly, and at consi depths lining must be heavy and expensive, to overcome friction; (*c*) in r ground, large excess of material is always removed through caisson, panied by settlement of surrounding surface and damage to adjacent stru For a temporary shaft, the cost of materials for a lining heavy enough t and strong enough to resist strains of sinking, is much greater than co timber sheet-pile lining which will serve the purpose if it can be plac cessfully. On other hand, in a permanent shaft the masonry caisson p the permanent lining, and cost of temporary timber is eliminated.

Materials. Open caisson was probably first used for sinking water wells, the caissons ... stone or brick masonry, as timber is usually too light to sink. The masonry ... upon a timber curb or SHOE, with steel cutting edge. Applying the idea to mine ... masonry stone masonry proved to be too weak to stand the uncertain stresses ... Hard-burned brick, well laid in cement mortar, is better, but still too weak ... caissons. The next step was a compound caisson, an outer sheet-steel shell, ... riveted to cutting edge, and lined with brick. Compressive stresses are borne ... and tensile by the steel, and caissons so constructed have been sunk to great ... difficult material. The widely different elastic properties of a thin steel shell and ... brick lining prevent the compound structure from acting as a unit, and serious ... in the brick are possible without rupturing the steel. Also, with compound, as ... stone or brick caissons, the only possible shape is circular. Development of ... concrete has produced a material so well fitted for caisson construction that ... used almost exclusively for important work. Concrete is as heavy as brick or ... a tougher and cheaper. With thorough reinforcement, and bonding between ... courses, a concrete caisson acts as a monolith; since its walls will bear both ten... compressive stresses, it may be rectangular, oval, or circular in section.

Design and construction. Factors to be considered are: (*a*) probable skin ... at different depths, and weight necessary to overcome it; (*b*) stresses ... sinking; (*c*) permanent exterior earth pressure. The last is least im... for any caisson heavy and strong enough to sink safely will stand the ... There are no exact rules for calculating stresses. Examples are ... of successful caisson work, with statement of skin friction exist... different depths. FRICTION of circular caissons is generally less than that ... angular, under similar conditions, because of the arch action of sur... soil. Friction is reduced by making outer surface true and smooth; ... well braced concrete forms are essential and for important work these ... be built or faced with steel plate. The outer surface is often given ... inward batter, so that as the caisson sinks it will tend to pull away from ... ing soil; though this action takes place in fairly firm material, in run... it is doubtful if anything is gained. Another device for reducing ... is to build occasional horizontal pipes into and extending through ... walls, through which water or air under pressure can be pumped ... ground to loosen its hold. Friction increases with depth, but not ... it is sometimes high around upper part and absent at bottom, so ... caisson is practically hung by the top. Severe tensile stresses may ... to resist which the successive masonry courses must be thoroughly ... and in deep caissons vertical reinforcement is also necessary. The ... p action is greatest when caisson cuts into running ground after ... a thick stratum of firm material.

... should be so constructed as to cut into the ground with minimum re... and to provide a firm base for the walls, so that "hanging up" of ... a boulder or irregular rock ledge will not seriously crack or damage ... 11 shows a common form of shoe, as used at Shaft 2, Rondout siphon ... 11. Fig 12 and 13 show other forms, in which the plates are con... through the wall by gusset stays and are bonded to masonry with ⅞-in ... bars. The latter may be extended to form the vertical reinforcement ... above, which, in any case, should be attached to shoe.

Excavation and sinking. Excavation within the caisson is done by hand, ... grange-peel or clam-shell grab bucket. The grab bucket is cheaper ... digging, though for a single shaft its cost, with the additional rigging ... handle it, outweighs the saving. But if the sinker owns, as part of ... equipment, a grab bucket with derrick and engine adapted to handle ... economical to install them rather than the lighter and cheaper equip... and work. Aside from cost, the character of the soil often determines

the method of excavation. An important point is to KEEP the CAISSO TICAL. In uniform ground this is done by excavating evenly around un shoe; in variable ground, when caisson deviates from vertical, it is recti cutting more material from under the shoe on the high side. When th strikes a boulder big enough to support the caisson, it must be remove fully to prevent an inrush (Art 2), or it is drilled and blasted into piece enough for the shoe to cut through. Thus, even when a grab bucket for most of the excavation, some hand work is necessary. When depth ting edge below bottom of excavation is less than width or diameter of deviation from vertical is easily corrected as stated above. At greater correction is difficult or impossible; but in this case the caisson is unli deviate, provided boulders are properly handled and inrushes of grou vented by keeping cutting edge well in advance. This is accomplished: starting with heavy caisson, having smooth outer skin and sharp cuttin

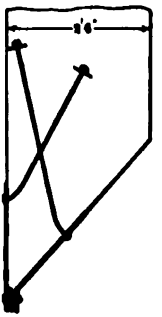


Fig 11 Roundout Caisson

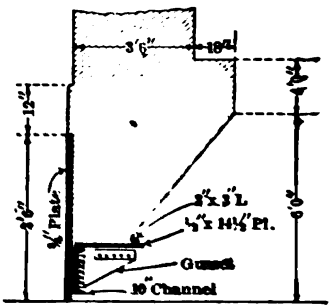


Fig 12. Colorado River Caisson

Fig 13

Cutting Edges of Shoes .All Rivets through Cutting Edges countersunk

) by weighting caisson; (c) by reducing skin friction. For extra w invert is best since a great weight can be piled on a relatively small top of caisson.

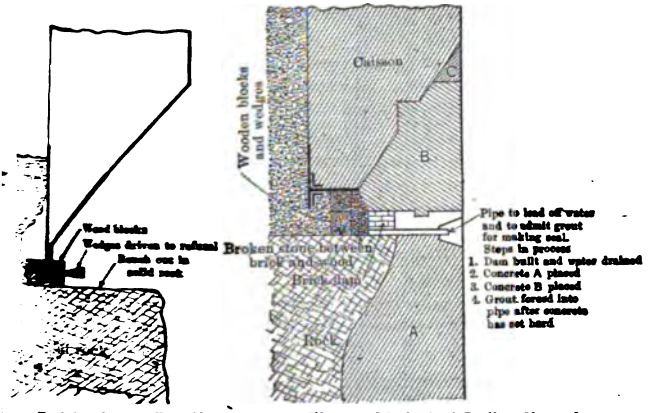
For deep caissons, extra weight should be provided for in the design Excavated material itself is the cheapest weight. Skin friction can sometim above by pumping air or water into the surrounding ground through holes (Example I, below), or by allowing the water to rise in caisson to balance a grab bucket being used for excavation under water. In latter case, since flow of water impedes it, are reduced and caisson is more readily lower the other hand, boulders are particularly troublesome and divers have sometimes required to remove them. Example III, below.

Sealing caisson to rock. When shoe reaches rock a permanent be made. This is done by [illegible] rock surface is level, so that cutting come to it over bearing [illegible] of rock is partly dissolved a depth of several inches. The disintegrated layer permits cutting and [illegible] it serving as a guard to exclude the sand. rock is hard, a tight joint can be made between shoe and rock by [illegible] sawdust will [illegible] and shape [illegible] will wonder wedges. Fig 14 having been wedged, rock excavation is commenced. As in sinking [illegible] the last rock cut should be [illegible] of diameter, a [illegible] to prevent [illegible] from breaking rock under cutting edge.

...ing may be done more freely, as there are no cross braces to be knocked ...

...en rock surface is sloping or irregular, and ground immediately over ...ad, the shoe may be rigidly supported at one point of its perimeter, the ...t being in soft ground. The caisson is thus thrown out of plumb, a con... which is aggravated by runs of soft material caused by attempts to break ...e rock under point of contact. This rock must be blasted and removed ...reat care, a little at a time, breast boards or short sheet piles being used ...k back adjacent soil.

...round be very soft, it may be necessary to support the free side of caisson on posts ...small sheeted pits driven to rock, and to hold back the ground temporarily by in... ...ding boards driven under flare of shoe. In this case the permanent seal is usually ...ructed as follows: The shaft is sunk 8 or 10 ft into solid rock, of a section as large



...Wedging for Sealing Shoe ...f Roundout Caisson

Fig 15. Method of Sealing Shoe of D, L & W Caisson

...ing above will permit, a small ledge being left to support the cutting edge, or ...s and sheet piles (Fig 15). All water coming in under shoe or through the piling ...at this ledge, and led through pipes to interior of shaft. A back form is con... from ledge to caisson, just inside cutting edge, and a concrete lining is built ...ft bottom to caisson. After concrete has set the drain pipes are plugged or ...with cement grout forced in under pressure (Art 7). Since the lining meets the ...order side of caisson it should be bonded to it with rods or bolts. If ground is ...t and rock surface irregular, the difficulties of founding and sealing the ... may be so great as to require use of compressed air to hold back the water (Art ...ergency to be considered in the design.

Examples

Shaft 2, Roundout siphon, Catskill aqueduct. The caisson for soft-ground por... ...shaft was cylindrical, 26 ft outside diam, walls 2 ft 6 in thick. Shoe (Fig 11) ...2 of 0.5 by 20-in steel plates, with 4 by 1-in filler at cutting edge, and was an... ...concrete by 80 ¾ by 30-in rods, attached alternately to inner and outer plates. ...consisted of 1 cement, 2 sand, 5 stone. Inner and outer forms were of 2 by 6-in ...wood lagging, supported by angle-iron rings tied through walls with ⅝-in rods. ...was built in 5- and 10-ft lifts to full depth of 55 ft, lifts being bonded together by ...vertical reinforcing rods, 4 ft c-c.

Finished shaft was 10 ft 8 in by 22 ft in clear. Borings showed 60 ft soft gro upper 6 ft being sandy loam, and the rest a material resembling blue clay when was completely saturated in place, flowing "like cold molasses and very stick bed rock, and surrounded by soft soil, were numerous hard boulders of all siz shaft site was leveled, shoe assembled upon short planks laid on ground, and forms started. 5 ft of concrete was placed and allowed to set for a week; 10 was then placed and when sufficiently set, sinking was begun. Mud was loa shaft buckets, sometimes with shovels, sometimes with water buckets, by men on plank rafts. Concrete was added as caisson sank. At a depth of 45 ft a laye

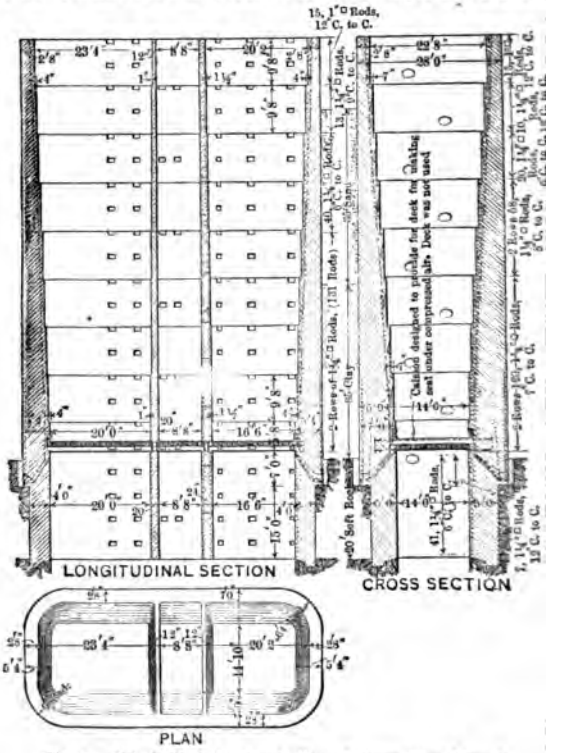


Fig 16. Del. Lack & Western Colliery Caisson (Eng News)

soft mud was encountered, which ran in under one side of shoe, throwing cai out of plumb. A trench was therefore dug through surface loam on high side o the material from it was piled against low side, and when sinking was resumed th straightened up. About 6 ft from rock, the shoe was stopped by boulders for lon time to allow mud to stick to caisson walls, so that after boulders were blast was necessary to load caisson with 200 tons of clay, and also to agitate the compressed air blown through 1.25-in pipes built into the wall. A layer of har found just over the rock, into which the cutting edge sank deep enough to seal son automatically. Average progress, from building shoe to sealing of caisson ft per day, including concreting and all delays. Cost per ft: concrete and s

...rating, $39; general expense, $32; total, $133. Before sinking was stopped by ... at depth of 50 ft, skin friction was less than 380 lb per sq ft; afterward, at same ... 500 lb.

Colliery shaft for D, L & W RR Co, near Wilkes-Barre, Pa (Fig 16).[1] Caisson ...ncrete, rectangular section, with rounded corners, and divided into 3 compart- ...cross walls. End compartments were arranged to permit further subdivision ...ermations. Outside dimensions, 28 by 59½ ft; total height 90 ft. Shoe is ...g 13. Thickness of walls at bottom: sides 7 ft; ends, 5 ft 4 in; outer surfaces ...ber surfaces stepped, in lifts of 9 ft 8 in; thickness at top, 2 ft 8 in; main ...forced vertically and horizontally with 1- and 1.25-in rods. At 7 ft above ...sson was closed by an air-tight deck, for sealing caisson to rock under air pres-

...d was leveled, shoe assembled, and 20 ft of concrete placed. Sinking was ...day and night, each shift consisting of a foreman and 16 men in shaft. After ...ched rock, the soft ground was held back temporarily with timber blocks wedged ...r under horizontal portion of the shoe (Fig 16). As this stratum was not firm ...o make a permanent seal, the shoe was undercut and shaft excavated 4 to 5 ft ...d around than inside of caisson at bottom. In blasting, great care was taken ...eak the ledge under shoe and blocking. Sound rock was found 15 ft below ...ace, and a wall was built up to underside of caisson. Drain pipes disposed of ...reaking through blocking, and were grouted after wall was finished and con- ...ete. During construction of this wall, water was led to the pipes by building ...ck dam upon the ledge. Total depth of soft ground, 70 ft; average progress, ...building of caisson and construction of temporary seal, about 7 in per day. ...ton, somewhat less than 700 lb per sq ft.

Colorado River siphon, Arizona (22). Shaft 30 ft outside diam. Walls ...k except for 10 ft above cutting edge (Fig 12). Shoe was assembled on bot- ...t 10 ft deep, and concrete walls were carried up 10 ft before sinking began. ...ed by hand to depth of 73 ft, making 62 ft sunk and about same height ...alt in 71 days. Pumps were used for lower 45 ft. When inflow amounted to ...00 gal per min, inrushes of ground under shoe prevented further progress; the ...as then flooded and dredged with a ½-cu yd clam-shell bucket. Ground was ...ft, and after caisson had sunk 5 ft further it stuck, although all material that ...uld reach was dug out to depth of 10 ft below shoe. Advance of 11 ft was ...lowering water level inside caisson, and a further advance of 2 ft by exploding ...e charges in pipes jetted down on outside to depth of 5 ft below shoe. 16 ft ...e gained by blasting underneath shoe, with dynamite placed by divers. Cais- ...stuck fast, being held by skin friction of over 400 lb per sq ft. The 34 ft done ...ng to this point took 50 days. Successful attempt to relieve skin friction by ...ing around the outside was now made; by which, and with use of dynamite, ...n was sunk to final depth of 139 ft, a further penetration of 32 ft in 38 days. ...pletion of first 10 ft of wall above the shoe, 128 ft of caisson was built and sunk ...s. Maximum skin friction was about 460 lb per sq ft, even when caisson was

4. Pneumatic Caissons

General design of pneumatic caissons is similar to that of open caissons ...cluding materials and construction; chief difference is in method of ... In the pneumatic caisson, an air-tight deck is constructed 7 or 8 ft ...e, and ground water is entirely excluded by maintaining under the ... air pressure equal to or greater than that of the water outside. A ...t is therefore required. Men are admitted to working chamber, and ...d materials removed, through one or more air locks fitted to openings ... General vertical section of a caisson is shown in Fig 17. ADVANTAGES: ... water is excluded and shaft bottom is held down by air pressure, in- ...der shoe are prevented, even in very fluid materials; (*b*) caisson may ... without "pulling in" surrounding soil, and hence may be kept con- ...vertical; (*c*) boulders or other obstructions under shoe can be removed

with greater ease and safety. DISADVANTAGES: (*a*) expensive air plant quired; (*b*) use of a grab bucket is precluded; (*c*) cost of hand excav is greatly increased; (*d*) po depth of sinking is limited to more than 100 ft below gr water level.

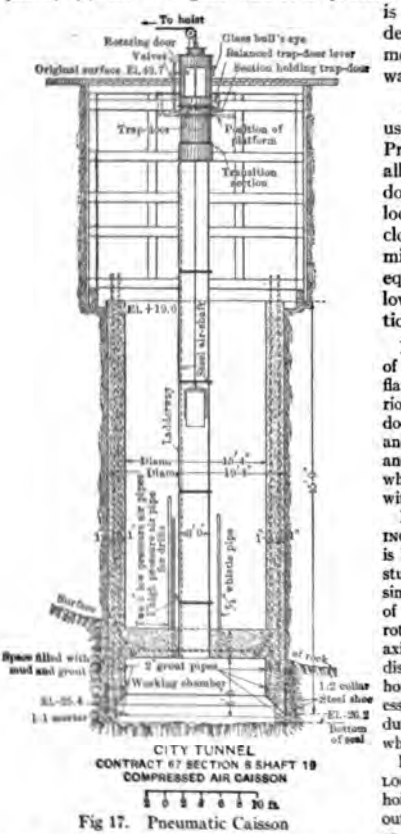


Fig 17. Pneumatic Caisson

Air locks of types most used are shown in Fig 18 Principle of operation is sa all: for admission to caisson, door is closed, upper one o lock is entered, and upper closed; compressed air is th mitted to lock until press equal to that in caisson, lower door is opened. The tion is reversed in passing o

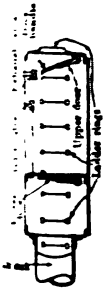
Fig 18 shows a MAN LOCK, for of men only, and fitted with flap doors. It is controlled fro rior, the lock tender standing o door plate and manipulating air and lower door. Upper door is and closed by an assistant out whom lock tender signals by with a hammer.

Fig 19 shows a MATTSEN E ING LOCK, into which a muck is hoisted by a rope passing th stuffing box in top. Lower d simple flap; upper door is in t of lock and opens by sliding s rotating at same time about th axis. The bucket to be rem disconnected from rope and horizontally, and as this is a slo ess the bucket is usually arra dump its load of material into while hanging in the lock.

Fig 20 and 21 show STRAIGHT LOCKS, by which the bucket hoisted or lowered through caiss out detaching from rope. Lo on both are simple flaps. POT-U (Fig 20) has an upper door which can be entirely disconnected, the hoisting ro ing through a stuffing box in the middle. While bucket is in caisson, several manipulated swing bolts hold lid in place. Bolts are loosened after bucket is hoi lock, lower door is closed, and lid is then hoisted away with the bucket. M (Fig 21) obviates necessity of tightening and loosening bolts, and is therefore qu operate. Upper door is a simple flap, so located that when bucket is hanging free in the rope passes through a groove in upper door frame, just clearing edge of door. Th is fitted with a special stuffing box, so designed that, by rotating it 180°, it clasps o the rope. In operation the bucket is lowered into lock through upper door, and pushed sidewise until rope enters groove; stuffing box is rotated, upper door closed a opened, and bucket lowered into caisson. All excavating locks are operated from

Excavation and sinking. Material from caissons is usually sho hand into 30-in cylindrical buckets and hoisted through lock. If a sc

si compressed air be available, loose sand can be blown out through a SUCTION PIPE (Example I), by shoveling it into the end of a suction hose at- to pipe. The work itself is simple, the difficulties of the process being effects of compressed air upon laborers. Up to pressures of about 20 lb equivalent to depth of 46 ft below ground-water level), men can pass



Ordinary Lock

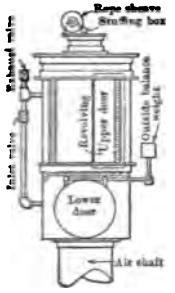


Fig 19. Mattsen Lock

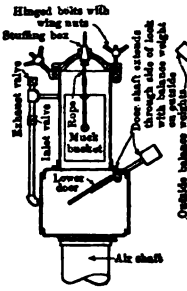


Fig 20. Pot-lid Lock

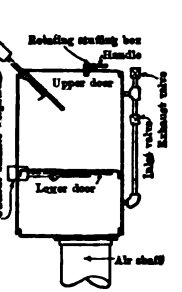


Fig 21. Moran Lock

lock in 4 or 5 min, and can work 8-hr shifts under pressure without in- but, as pressure increases above this point, the time of locking through must reased and length of shifts decreased, to avoid caisson workers' paralysis (Sec 15, Art 34.) Rate of wages also increases with air pressure. human endurance is about 50-lb pressure. Table 1 shows requirements York law governing caisson work, and wage scale in force in 1914:

Table 1. Requirements of New York Law for Caisson Work

Pressure, lb per sq in	Hours per shift		Wages per shift	Pressure	Minimum time of decompression
	Total	As worked			
				Up to 10 lb	1 min
10 to 21 lb	8	½ hr for lunch	$4.50	10 to 15 "	2 "
21 " 30 "	6	3 on, 2 off, 3 on	4.75	15 to 20 "	5 "
30 " 35 "	4	2 on, 4 off, 2 on	5.00	20 to 25 "	10 "
35 " 40 "	3	1.5 on, 4.5 off, 1.5 on	5.25	25 to 30 "	12 "
40 " 45 "	2	1 on, 5 off, 1 on	5.50	30 to 36 "	15 "
45 " 50 "	1.5	As arranged	5.50	36 to 40 "	20 "
	.....	.....	.....	40 to 50 "	25 "

Wages are increased 50¢ per shift on concrete work.

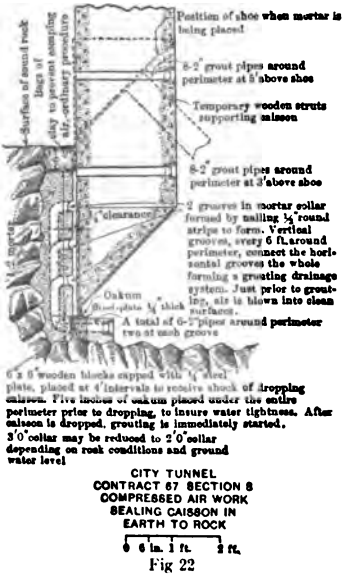
supply. Sufficient air must be pumped into caisson for adequate venti- Good practice requires 30 cu ft per min of free air per man. When less amount escapes in lockages and leakages, the remainder usually flows the shoe.

pressure required, theoretically, should suffice to balance hydrostatic ting at the level of working chamber, due to depth below water level. ead may be greater or less than depth below surface of soil. Pressure, sq in per ft head, is 0.433 lb in fresh water, 0.445 lb in sea water. If should pierce a stratum of stiff impervious clay, it might be possible t

reduce working pressure much below that theoretically required (as at hattan bridge, N Y), and thereby increase the attainable depth. To ma a vertical face of soil under the unsupported side of a caisson resting at ot on sloping or irregular bedrock, the pressure should correspond to the head on the face.

Weighting. In sinking pneumatic caissons, upward pressure of comp air must be overcome, as well as skin friction; hence, they are usually h per unit of horizontal area than open caissons. On other hand, the bu up of air escaping under shoe stirs up the ground and reduces its adhe caisson. Pneumatic caissons can easily be loaded by filling space over th between walls and air shaft, with excavated material; pig iron may b on top, if necessary. A sudden release of air pressure (the men having l working chamber) is an effective method of starting a stuck caisson.

Sealing to rock of a pneumatic caisson is similar to that of open c (Art 3), except that, instead of inflowing water, outflowing air must l


Position of shoe when mortar is being placed
8-2" grout pipes around perimeter at 8' above shoe
Temporary wooden struts supporting caisson
8-2" grout pipes around perimeter at 3' above shoe
2" grooves in mortar collar formed by nailing ½" round strips to form. Vertical grooves, every 6 ft. around perimeter, connect the horizontal grooves the whole forming a grouting drainage system. Just prior to grouting, air is blown into clean surfaces.
A total of 6-2" pipes around perimeter two at each groove
6" clearance
Oakum
6 x 8" wooden blocks capped with ½" steel plate, placed at 4' intervals to receive shock of dropping caisson. Five inches of oakum placed under the entire perimeter prior to dropping, to insure water tightness. After caisson is dropped, grouting is immediately started.
3'0" collar may be reduced to 2'0" collar depending on rock conditions and ground water level
CITY TUNNEL
CONTRACT 67 SECTION 8
COMPRESSED AIR WORK
SEALING CAISSON IN EARTH TO ROCK
0 6 in. 1 ft. 2 ft.
Fig 22

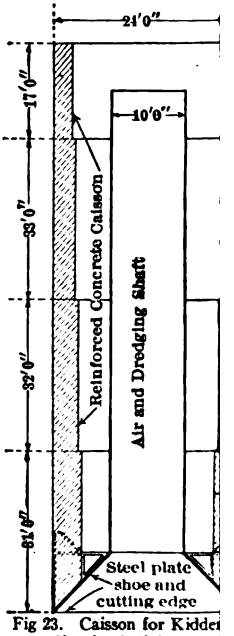


Fig 23. Caisson for Kidde Cleveland-Cliffs Iron

vented in making the seal. Leveling of the rock ledge and blocking up are done in same way. A strip of waterproofed canvas laid over th assists in preventing escape of air, but provision should be made for gr

Several caissons of the City tunnel, Catskill aqueduct, were sealed as follows (a) ledge leveled; (b) shaft excavated to depth of 3 ft (about 1 ft larger all arou outside of shoe), caisson being supported on timber posts; (c) excavated porti with a 1 to 2 mortar wall, of an inner diam 1.5 in greater than outside diam

...ring ½-in clearance all round; (*d*) horizontal ledge cut at bottom, and grout ... into the lining, with their ends projecting inward through the ledge; (*e*) thick ... sacking placed on ledge, and posts supporting caisson shot out, allowing it to ... sacking forming a watertight joint between shoe and mortar ledge; (*f*) holes ...

Table 2. Details of Sinking 5 Reinforced Concrete Caissons, Catskill Aqueduct

	Shaft 19	Shaft 20	Shaft 22	Shaft 23	Shaft 24
...de diameter, ft	19½	19½	19½	24	24
...diameter, ft	15½	15½	15½	18	18
...height, ft	45	105.3	100	123	105.6
...er of locks	1	2	2	2	2
...rete proportions, cement: sand: stone	1½ : 2 : 4	1½ : 2 : 4	1½ : 2 : 4 to 1 : 2 : 4	1½ : 2 : 4 to 1 : 2 : 4	1½ : 2 : 4 to 1 : 2 : 4
...sank under compressed air:					
...sand, ft	11	77	50.3	105	59.5
...rock, ft	14.2	33.7	13.7	14	10
...rage progress:					
...ncreting, ft per working day	3.75	2.85	3.56	3.23	3.08
...ing under pressure in sand, ft per working hr	0.0275	0.0346	0.0393	0.0591	0.028
...ing under pressure in rock, ft per working hr	0.0035	0.0075	0.0056	0.004	0.0025
...hours constructing	64	35	51	51½	51
...rage progress, ft per ...elapsed time, from ...of shoe to completed total depth	0.82	1.18	1.25	1.20	1.22
...mum air pressure, lb ...sq in	17	39	28	46	29.5
...ght of caisson, tons	460	1 050	978	2 323	1 780
...ght of caisson, with ...load of sand and ...tons	700	2 100	2 470	4 612	4 046
...ional resistance, lb ...per ft of outer surface at various depths	300 to 400	630	630 at 81 751 at 95	1 411 at 45 1 202 at 86 872 at 116	1 685 at 49 1 101 at 79 945 at 93.5
...struction price per ft for ...finishing only (concrete ...and reinforcing steel ...separately)	$466	$471	$456	$735	$614

Note.—All these caissons were weighted with excavated sand piled on top of deck ...over shafts. In each case, a pit was excavated and timbered square to depth of ... ft. The shoe was set on bottom, and caisson built to its full height before sinking ... started. Contract price allowed a fair profit to contractor.

Examples

1. **Shaft 19, N Y aqueduct** (Fig 17). For structural reasons not related to sinking, ... and horizontal reinforcement of concrete was made unusually heavy. The ... ckness of 2 ft, for a required inside diam of 15 ft 4 in, was at least 1 ft less than ... required for a caisson of same depth. The deck was a 3-ft slab of reinforced ... cast integral with caisson walls, and was cut out after seal was made. Ex... the concrete deck, the design was typical: a 36-in circular opening was provi...

in deck and a vertical line of 36-in flanged steel pipe (air shaft) was led from open top of caisson, where air lock was attached. Opening was formed by casting bottom of air shaft into the deck; (where deck is of wood or iron, the lower flange of air bolted to it). Air shaft was long enough to keep lock always above ground-water l that, in case of accident to lock or to air-compressing plant, the caisson men would trapped by rising water. Air shaft had ladder rungs so arranged as not to in with operation of bucket. Besides the air shaft, one or two 3-in inlet air pipes at bottom with check valves, a 0.75-in whistle (signal) pipe, a high-pressure air conduit for electric wires, and sometimes a 4- or 6-in discharge or "blow" pipe, through deck. In deeper caissons, 2 air shafts were provided, fitted respectively material lock and a man lock.

II. Kidder shaft, Cleveland-Cliffs Iron Co, Mich. Caisson was 24 ft diam (Fig 23). Air shaft, 10 ft diam, was used first as a dredging shaft for a cla bucket, which excavated to a depth of 87 ft. As it then became necessary to us pressed air, a deck and air lock were bolted to the top. Ledge rock was reac 104 ft. Shoe was sealed to rock at 113 ft. Average progress, 0.72 ft per day time.

5. Forced Drop-shafts

General description. For penetrating depths of quicksand and oth stable, water-bearing material, beyond the limit of open or pneumatic cai a method has been developed in Germany of jacking down a telescopic se iron drums, inside of and against a preliminary cylindrical concrete ca The latter is strong and heavy, and built into it, near the top, is an in cast-iron flange, the REACTION RING. This is anchored by vertical rods e ing to the shoe, for resisting thrust of the jacks, which are attached to an against under side of ring. The caisson is sunk by dredging in the ope depth of 50 or 60 ft; then a concrete floor, PLUG, is placed in the bottom (water if necessary), and shaft is unwatered. A cast-steel shoe, with an o diam slightly less than inside diam of caisson, is set on the concrete floor, cylindrical drum, of flanged and bolted cast-iron segments (similar to tubbing, Sec 7), is built up from shoe to under side of the jacks. The conc then broken, thus admitting water to natural level, and the drum is jacked the material being excavated under water by a grab bucket, mammoth or sack borer (described below). Finally, the jacks are removed and add segments of lining added as required. By using hydraulic jacks, a drum outside diam can usually be forced down 250 to 300 ft before it sticks. I lies still deeper, the bottom is again covered with concrete, the shaft watered, a second drum of smaller diameter is built inside the first, the are shifted inward to bear upon it, and sinking is resumed. Frequently a drum has been necessary; less often, a third (4).

Details of construction of caissons, drums and sinking plant are sh Fig 24 and 25. A headframe is provided to handle the machinery, whi cludes: trepans, similar to those used in the Kind-Chaudron boring meth rock (Sec 7), for breaking up the concrete floor and any boulders or par mented ground that may be met; the caisson, with reaction ring and hy jacks; and sinking drums. The outer drum (Fig 24) is the patented Pa compound, the cast-iron shell of which is lined with 22 in of strong brick crete, for additional weight and stiffness; the second is a simple iron-s drum. The compound drum is made necessary by the great earth pr at depths of 300 or 400 ft, a number of shafts having been lost by colla unsupported iron drums, notwithstanding use of segments 3.5 in thick. segments are about 5 ft high, flanged and bolted on both horizontal and v joints, and 8 to 10 of them make up a ring. The shoe must be very stro heavy, and anchor and reaction rings and all bolts must be designed to

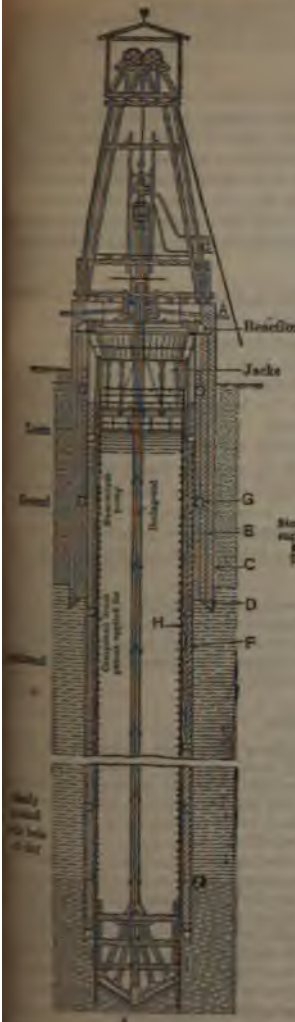


Fig 24. Compound Drum, Mammoth Pump ... Shafts IV and V, Rheinpreussen ... Germany. *A*, Suction Pipe and Over- ... Compressed Air; *C*, Caisson; *D*, Shoe; ... Rods; *F*, Compound Lining; *H*, Cast- ... Lining (Riemer)

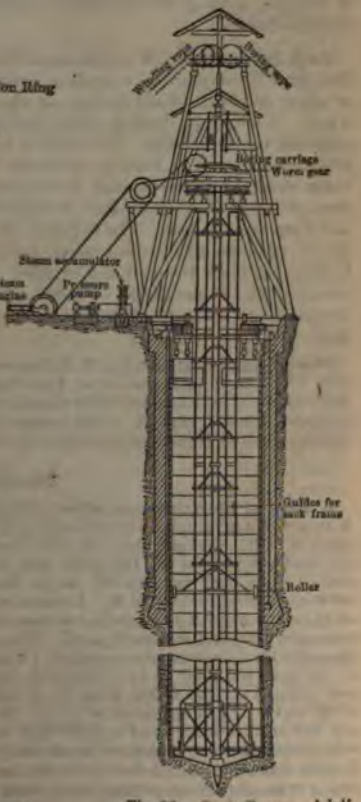


Fig 25. Sack Borer. Adolf Shaft, Eschweiler Mining Co, Germany (Riemer)

safely the full thrust of the jacks. For considerable depths, the grab b used in ordinary caissons has been superseded by the mammoth pump in fi and the sack borer in softer, soils.

Mammoth pump (Fig 24) comprises the hollow rods of the trepan or b tool (through which water is pumped under pressure to cutting edge of tre in combination with an air-lift pump, the pipes of which are also attach boring rods. The suction opening of the pipes is at the middle of the tr just above its point. The trepan should make short rapid blows, to agitat broken fragments of ground, so that they may be more quickly sucked i discharged by the pump. As compared with the sack-borer, and other rot or dredging devices, the mammoth pump shows less tendency to deviate vertical direction.

Sack borer (Fig 25) is a large auger-like tool, with its stem in center of The stem is composed of a series of lengths of heavy flanged pipe, termi at upper end by a splined section, on which is mounted a large horizontal wheel. A wire rope, from hoisting engine drum to swivel link at top of suspends the sack borer. The stem is rotated through the gear-wheel b other engine, and is lowered gradually by hoisting rope. New sectio borer stem are added as shaft is deepened. Cross arms are attached t stem at intervals, having rollers at their ends which bear against sides of and keep stem in line. The material cut by rotating borer is swept int heavy, open-mouthed canvas sacks, fastened to backs of cutters. From to time the borer is raised and emptied. In an improved form, the sack mounted on a frame sliding on guides attached to cross arms on stem, an hoisted by an independent engine. The sack borer is best adapted to cla sand.

Sassenberg process of hydraulic flushing reduces skin friction and adhesion in soils. Shoe and 4 lining rings above it are made of about 1.5 in larger outside dia than the rest of the lining, and in the shoulder thus formed are water passages, whi connected through pipes to a high-pressure pump. By operating this pump sinking, the drum is partially surrounded by a film of water.

Examples

I. Shaft 5, Rheinpreussen colliery, Homburg am Rhein, Germany, was s in 1901 with a brick caisson 29.2 ft inside diam, walls about 3.5 ft thick. This rea depth of 65 ft. Concrete plug, 9 ft thick, was then placed on the bottom, under the shaft was pumped out, the anchor ring, rods and reaction ring (designed for a sure of 3 000 tons) were erected, and an inner, truly vertical, brick lining was reducing inside diam to 25.68 ft. A compound sinking drum with outer and inner of 25.52 and 21.32 ft respectively was then constructed, and sinking was begun percussion borer and mammoth pump. The concrete was bored through in 4 day thereafter the average progress was about 5 ft per day. The compound drum stuck ft, and shaft was filled for 60 ft with sand and gravel (instead of concrete). Sha next pumped out and an iron drum, 3.5 in thick and 19.35 ft inside diam, was bu to the jacks. This drum struck at 315 ft; the shaft was again partly filled and pu out, and another drum, 17.38 ft inside diam was forced to a depth of 343 ft, whe shoe entered clay firm enough to permit shaft to be pumped out. A fourth drum ft inside diam, was finally forced to the coal measures, at a depth of 508 ft. The si took 3 years, the average progress being about 6 in per day (4).

II. Sterkrade shaft, near Holten, Germany, was started with a brick c 24.6 ft inside diam, which was sunk to a depth of 59 ft. The excavation was cont by hand to 131 ft, where an iron sinking drum, 22 ft inside diam, was constructed. drum was forced to 264 ft; a second drum, 19.3 ft diam, to 433 ft; and a third, 16 diam, to 448 ft. Here the water was found to be successfully shut out, and sinkin continued by hand (4).

Table 3. Cost of Sterkrade Shaft

Brick curbing, 59 ft @ $285		$16 800
1st iron drum sunk 133 ft @ $502		
Segments	$32 900	
Labor and supplies	33 900	66 800
2d iron drum sunk 169 ft @ $715		
Segments	75 300	
Labor and supplies	45 900	121 200
3d iron drum sunk 15 ft @ $6 060		
Segments	71 500	
Labor and supplies	19 300	90 800
Depreciation of plant and fittings, 50% of new value		43 300
Total		$338 900
Less salvage on tubbing recovered from inner drums		35 700
Total cost for 376 ft		$303 200
Cost per ft		807

Average cost of sinking-drum method in Germany, at various depths, is given by Henry Louis, as follows:

Depth	Cost per ft
52 to 164 ft	$260
164 to 328 ft	593
328 to 492 ft	817
492 to 656 ft	1 040

6. Freezing Process

General principles. This process, invented by F. H. Poetsch in 1883, consists in solidifying soft watery ground by freezing, prior to excavation, so that the frozen ground itself makes a water-tight wall around the shaft, permitting safe excavation. Freezing is continued until rock is reached, after which the shaft is lined. A number of vertical holes are bored from 2 to 4 ft apart, in a circle outside the periphery of proposed shaft, and cased through the soft strata into rock. A freezing plant is erected, and the freezing solution is circulated down interior pipes and up through the bore-hole casings, until the surrounding ground is frozen solid. The process is adapted to soft ground or wet rock, and requires the material to be such that holes can be bored with reasonable ease and accuracy; ground water must also contain little salt in solution, and not be in active circulation. Strong circulation of ground water will hinder or may absolutely prevent freezing. The holes should be vertical, for a slight deviation may result in so wide a gap at bottom, between two adjacent pipes, as to prevent thorough freezing of the ground between them. To overcome this difficulty in sinking to great depths, freezing in successive lifts has been suggested. Some saline solutions freeze at a lower temperature than plain water, a leak or break in a freezing pipe is serious; the solution, by escaping into the surrounding soil, will cause a weak spot, if not a hole, in the frozen wall and make necessary a longer period of freezing. Furthermore, at great depths ice tends to flow under pressure, and the thickness of ice wall required to resist this tendency at a given depth must be studied. Frozen sand is much stronger than pure ice. Alby's experiments (23) show that frozen saturated sand crushes under 1 500 lb per sq in. at temperature of $-25°$ C, and at about 1 700 lb per sq in at $-12°$ C. It should therefore carry safely a pressure of about 300 lb per sq in, and assuming full hydrostatic pressure on outside of ice cylinder, the thickness of ice wall for a shaft 300 ft deep and 20 ft diam outside the permanent lining would need to be 7.5 ft.

Freezing tubes are sunk by methods ordinarily used for casing bo (Sec 9); in soft ground by washing the material up inside casing by water forced down a smaller interior pipe (with occasional use of sand and in hard ground by drilling with a percussion well drill, the casin forced or hammered down in both cases. The holes should be surveye time to time, so that one with too great a deviation may be abandoned, new hole started. Absolute verticality is impossible; good practice lim deviation to not over 3 ft in 300 ft. (Holes are surveyed by method in Sec 9.) Difficulties caused by breaking of casin to temperature stresses during freezing, make it ad to use independent freezing tubes. Fig 26 shows used at Dawdon colliery, Durham, England, in 1902 4-in heavy pipe, with its lower end welded shut, is l into the 6-in bore-hole casing, tested hydraulicall casing is then withdrawn. The freezing solution is p down through the 1-in interior pipe and returns t the other. The inner and outer pipes are connec headers as shown, valves being provided to regulat of solution and cut out any hole quickly, in case o age.

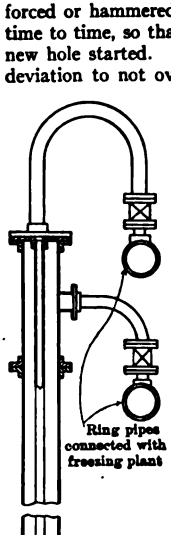


Fig 26. Freezing Pipes, Dawdon Colliery, England

Freezing plant and freezing. Ammonia compr freezing plants are generally used, the Linde type of m being particularly satisfactory. The freezing soluti brine, is chilled by the expansion of compressed am gas, and is then circulated through the freezing pip magnesium chloride solution is recommended, as th has less tendency to precipitate at low temperature the calcium chloride commonly used in ice plants, and is consequently less danger of clogging the pipes. required capacity for a given shaft may be worke theoretically, but good European practice prescri capacity of from 1 to 2 tons of ice per hr for dep 200 to 300 ft, and a freezing period of 3 to 5 month a rough approximation, 1 ton of ice per hr requires at compressor engine. The total hp of different plant ranged from 50 to 250. The condition of the ground being frozen is indicated by temperatures of inflowin returning brine. The brine leaves machine at a temper of about $-20°$ C, and at first returns comparatively w As the measures become frozen, the temperature of the re ing fluid drops until the difference is only $2°$ or $3°$ C, excavation can be safely begun. Throughout excavatio freezing plant must operate at full speed, to maintain t wall against thawing by circulation of warm air in the s

Sinking, after freezing, is done by hand. It is usually necessary to blast the f sand with light charges; black powder, gelignite, and dynamite having been used cessfully. Care must be taken not to crack the frozen wall, for the smallest leak is tain to enlarge itself by thawing until a disastrous inrush may occur. Presence of in a drill hole signifies that the soil is insufficiently frozen, and sinking should be sto

adly near the bottom, new holes must be kes necessary a longer freezing period.

nking by Freezing

h oy	Time, months			Average progress, ft per month
	Boring	Freezing only	Sinking and lining	
	17	5.5	7	10.5
	6.5	5.5	4.5	10.9
	6	2.5	6	18.1
	10.5	6	12	9.1
	10	13	6	3.9

es

ine, Leopoldshall, Stassfurt, Germany. .25 ft diam and cased to depth of 325 ft. le 4, consumed 17 months. Freezing was gun. After only 30 ft had been sunk, a flooded shaft. Sinking was stopped and ich a progress of over 60 ft per month was was lined with iron tubbing (Sec 7), the h water containing calcined soda. Sinking he tubbing was sealed to rock at 325 ft. king was entirely successful (4).

wdon, County Durham, England, were k to depths of 350 and 204 ft respectively. ing through underlying sand to the coal les were bored around each shaft, includ- drills being used. The holes were 8 and ezing plant comprised two 135-h p steam shown by Table 4, the freezing of Theresa before sinking could be begun. Both ars (24).

by Freezing (Henry Louis)

Anzin (2 shafts)	Flinez les Rache	Lens
Vicq, France	France	France
12.8 and 16.4	13¾	15¾
20 and 16	.....	.....
772*	246	492
$6 360	.....	.....
3 800	.....	$7 780
14 310	22 530	4 080
54 780†	.....	6 610
21 320	3 010	18 470
37 400	9 460	21 380
$137 970	$35 000	$58 320
$ 178.75	$ 142.25	$ 118.55

ezing only, $6 435.

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III. Chapin shaft, Iron Mountain, Mich (10), was sunk in 1888–9, throug sand, gravel and boulders, with water level 10 ft below surface. Twenty-six 10-i pipes were put down in a 29-ft circle; inside of these were 8-in freezing pipes, i 1.5-in circulating tubes. Casings were then withdrawn. Ice machine was a (ammonia type) of 50 tons daily refrigerating capacity (1 ton equals cooling effect melting ice). Freezing fluid was saturated solution of commercial $CaCl_2$; vel flow in the 1.5-in pipes, 2 ft per sec. In about 21 days after starting ice mach frozen ring was complete and excavation was begun. Rock was reached in 1 (including a 30-day stoppage, when shaft was allowed to fill with water). Th some trouble and delay from leakages through ice wall, the entire elapsed time, to at rock, being about 200 days.

7. Grouting Process (for rock)

Characteristics. The general principle of solidifying water-bearing in advance of sinking, accomplished in soft ground by freezing, is app hard rock by injection of Portland cement grout into the fissures; the gr process was used for this purpose in France, as early as 1882. It has sinc used many times with entire success (13), is rapidly gaining favor, and is recognized as the cheapest method of sinking in wet rock.

The advantages of completely stopping, or limiting the flow of water into a sh great: pumps and piping are reduced, fewer boilers are required, and operating cos throughout. Against this saving must be placed the cost of the cement and time o in grouting. In case of a fissure with a comparatively small flow, say 100 gal per m depth of 100 ft, with ordinary boilers and sinking pumps, and coal at $3 per ton, cost alone for pumping will amount to about $70 per month. Such a fissure would pr not require more than 100 bbl cement, which at $2 per bbl would be paid for in 3 by saving in coal. The cost of hindrance to sinking cannot be stated definitely, b flow of 500 or 600 gal per min into the bottom of a shaft will usually reduce rate of to one-fourth of that in a dry shaft, it would appear that three-fourths of the tot could be profitably spent in grouting, provided it were necessary to keep the sha Grouting is difficult in water-bearing fissures carrying sand or mud (Example I, and is not applicable to soft ground. Two common variations are practiced:

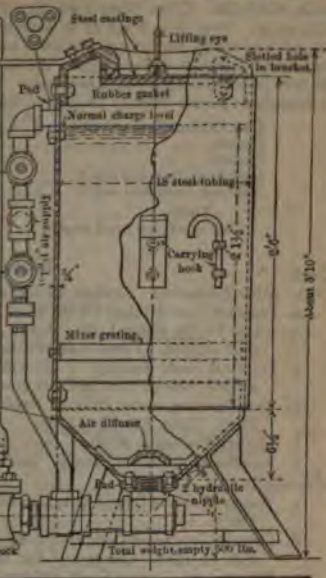
(*a*) Deep-hole method, the purpose of which is to fill the fissures in one operat a considerable depth of water-bearing rock; (*b*) short-hole method, by which fissu treated successively as they are reached. The former has been used in Europ number of instances; the second is becoming common American practice.

Deep-hole method. A number of holes are drilled through the fis ground, either inside or outside the proposed shaft periphery, pipes are nected to them and grout is poured or forced from surface until the holes to admit more. Operation of injecting is begun at bottom of holes, the gro pipes being withdrawn by stages, as fast as fissures are filled. To aid lation of grout, a second series of holes may be drilled outside the grout and suction pumps attached to them, thus removing from the fissures all or other materials likely to clog the grout holes. The holes should be bored a core drill, rather than a percussion drill, as operation of the latter is lik obstruct fissure with cuttings. When hole is completed, rapid pumping its bottom for several hours will remove refuse from rock crevices.

Short-hole method. The same holes drilled for blasting are used for g ing, if they cut into a water-bearing fissure; grout is forced in by a pneu grout tank. The Canniff tank and mixer (Fig 27) is a cylinder 18 in dia 3 ft long, with a door on top through which cement, sand, and water introduced, a 2-in discharge opening at bottom, closed by a stop cock, an connections top and bottom, each controlled by a valve.

Mode of procedure. Grout connection pipes are first prepared, each consistin 3-ft piece of 2- or 2.5-in pipe, tapering gradually to 1.5- or 2-in diam at one end, thre and fitted with a heavy iron stopcock at other end. When a drill hole cuts a water- ing seam, drill is withdrawn and one of the pipes is driven into it, the tapered end

to make a watertight joint. Stopcock is after pipe is firmly sealed. After all flow-



ig 27. Tank Grouting Machine

ples

uct, 17-ft diam, was sunk in 1911 without illed below this depth cut an open fissure, This was plugged temporarily with a wooden e following sump holes cut same fissure and en begun with a Canniff tank (Fig 27) and es up to 150 lb. Allowing the last batches issure was found to be completely sealed. cut, filled with finely crushed material, due cles ranged in size from sand to large pebbles, oles in large quantities by the water. As al, drilling of holes and pumping grout con-batches, nearly all neat cement, were placed was reduced to less than 50 gal per min, and ost of grouting both fissures was:

.....	$1 078.85
.....	839.45
.....	288.00
.....	767.05
	$2 973.35
.....	596.80
.....	562.00
.....	1 323.00
.....	30.00
.....	15.00
.....	$5 500.15
.....	86.00

STANFORD LIBRARIES

II. Lens Mining Co, France (13), sunk a shaft by direct cementation in Ordinary sinking, with pumping, carried depth to 126 ft before the inflow becan great to handle. Grout pipes were then inserted in fissures, and shaft allowed with water. Four 8-in holes, 13 ft outside the shaft, were bored through the fi ground, and sludge pumps were attached to them. Cement grout was run into th ices from a mixer at surface, and at the same time mud and water were pumped the bore holes, to assist grout to penetrate. Injection of 500 bbl cement sealed the f sufficiently to permit shaft to be sunk to solid rock at 492 ft. The cost was:

Installation of plant, etc	$12 800
Pumps	5 040
Cement	1 550
Labor, sinking and lining	21 770
Tubbing (material only)	17 100
Total	$58 260
Cost per ft for total depth	118.30

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ON 9

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TAGGART

SCIENTIFIC SCHOOL, YALE UNIVERSITY

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r to Bibliography at end of this section.

Y HAND

gs (Drivepipes)

ployed for sampling soft strata over- ock below surface, and to sink stand- or depths exceeding 100 ft.

in Art 10. For testing superficial pent in moving from point to point; as follows.

of 4 by 4-in timber, complete with wind- uck wheels should be heavy, about 40-in 1⅛-in hoisting rope. Flush-joint casing,

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2.5-in, 5-ft lengths, for the probable maximum depth of hole; also a few lengths, 2, 3 and 4 ft. (Sleeve couplings can be used, but are troublesome in sinking and p

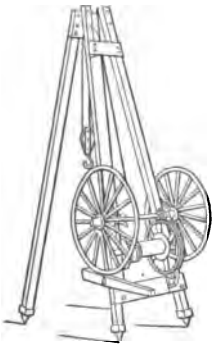

Fig 1. Derrick for Wash Boring

1 casing driveshoe; 1 casing drivehead; 1 casing tee. joint, 1⅛-in drill rods, in 10-ft lengths, with coupling ing 10 ft longer than casing. Add two 5-ft lengths. gas pipe can be used in light sandy material.) 1 l water swivel and coupling; 1 hoisting swivel; 3 cross cl bits; 1 chisel bit; 1 bushing, water swivel to casing; for taking dry samples; 1 worm auger; 1 hand forc 4-in cyl by 4.5-in stroke; 15 ft, 1.5-in suction hos coupling and strainer; 50 ft, 0.75-in, 4-ply, rubber p hose, with couplings for pump and water swivel. 2, 5-ft crowbars; 15 ft of ⅜-in chain, with hook an 1 coldchisel; 2 hose couplings; 1 150-lb drive-block, 2 cornered 6-in files; 2, 10-in flat files; 1 machinist's ha 1 oiler; 1 pick; 1 galvanized pail; 1 saw; 1 screw 1 shovel; 1 tape line; 2, 36-in pipe tongs; 1 tool box; unions; 3 Stillson wrenches, 10, 14 and 24-in; 2 r wrenches, 6 and 15-in; sample boxes and 4-oz wide bottles; 1 firing battery; dynamite, electric detonat 400 ft insulated copper wire; 1 pair sister hooks for rope; 1 shovel; 1 pair combined nippers and pliers; with bits 0.25 to 1 in; 4 pairs Brown's patent pipe

2 No 3 and 2 No 4; 1 pair pipe clamps for 2.5-in casing; 2, 10-ton jack screws; extra valves, liners and packing for pump; 4 balls candle-wick; 1 hank sash cord; 1, 1-gal oil can and lubricating oil; 1 wagon. This outfit, for 50-ft holes, costs $350 to $400, and will last several years.

Operation. A pipe of the required diam is sunk, the core is broken up by a jet of water, or, if necessary, by a chisel bit, and the disintegrated material is brought to surface by the current of water. For penetrating SOFT GROUND, a smaller pipe carrying water under pressure is worked ahead inside the drivepipe as fast as the loosened stuff is carried to surface. In such material the drivepipe sinks of its own weight, or can be made to do so by rotating it by cross-bars (brace head), weighted by old carwheels, if necessary. In HARDER MATERIAL, a chisel bit is attached to lower end of wash pipe, and churned up and down to cut a hole below the drivepipe. Cuttings are washed to surface by a stream of water issuing from holes in the sides of bit. In this case the rig shown in Fig 2 is used, ropes *A* and *B* passing through a double block in the derrick. In GRAVELS the fine material is washed out, leaving coarser pebbles in the hole. If small enough, these are bailed out with a sand pump. If too large, they are broken with a cross chopping bit before pumping. BOULDERS are drilled through ahead of the casing, then broken with dynamite. The charge, made up for electric firing, is lowered to the proper depth by the leading wires. Before shooting, the casing is raised several feet to prevent injury. After shooting, casing can usually be driven through the broken rock. For testing overburden 2.5-in drivepipe and 0.5 or 0.75-in water pipe for drill rods are common sizes.

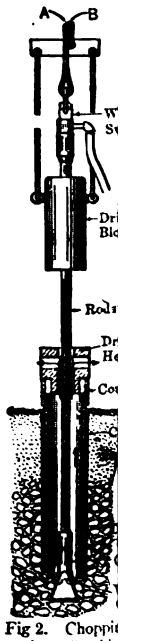

Fig 2. Choppi in Combi with Drive

For deep holes or holes in bad ground, several strings of pipe may have to be sunk, before reaching bedrock. This contingency must be allowed for in sinking diamond-drill standpipe, as the last pipe must be enough for the diamond bit (Fig 50). Minimum size of standpipe for diamond dr

oles have been started with 18-in pipe at
boring, but cost increases rapidly with
ons, and when samples are not required,
water swivel and drivehead to the pipe
while pumping down water. The water
es outside of the pipe. Crew of a wash-
and a team.

m test holes after they have served
attaching windlass rope to the pipe
ipe is twisted with tongs. If neces-
raised by jacks or levers; a bight of
e place of a clamp. By proper ar-
e caused to give an upward blow on
t than placing it, especially in clay.

erienced crew the speed of advance,
f soil and depth of hole.

e to depths of 50 ft. In stiff clays, 5 to
p, in sand, 12 to 15 ft per day is average
Tables 41, 49, 50 and 52.

eed of advance, size of crew, wages
country, water supply, and climate.

Canal, 666 holes were sunk by wash bor-
an. Aver depth of holes drilled in any
per ft was $0.18, in Sept, 1906, when 49
soil. Highest monthly cost, $2.70 per
were sunk in clay, sand, and hardpan.
per ft. Crew: 1 foreman, 3 laborers.
d team ranged from 87 to 98% of total
the Panama Canal, aver cost of 115-ft
arse gravel, was $0.83 per ft, including
15 for plant.

st-hole Diggers

ecting in soft ground. Fig 3 shows
outh, consisting of derrick, windlass,

Four Types of Earth Augers

ods. Rotating may be done by a
e power drives both rods and hoist.
ying in diam from 6 to 24 in. Form
and is present, *C* in sand, and *D* in

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clayey ground containing boulders. Holes to 700 ft deep have been bo
this type of rig. On Long Island, wells 250 ft deep are bored with th
rig more cheaply than with power machines. SMALL AUGERS, 1.5 in
with post-hole diggers and chisel drills as accessories, have been large
for prospecting and geological work.

C. Catlett lists following outfit as used by him in prospecting soft, superfic
ore deposits: (*a*) 2-in auger bit of steel or Swedish iron with steel point, twiste
spiral, thickness of blade not less than 0.25 in, length 13 in, pitch of spiral 4.25 i
was welded to one end of an 18-in length of 1-in iron pipe which was threaded
end. (*b*) 2-in chisel bit, made of 1 ft of 1⅜-in octagon steel, welded, like the
an 18-in length of 1-in pipe threaded for connection. (*c*) 10 ft of 1.25-in iron rod,
both ends for connection with 1-in pipe. (*d*) Lengths of 1-in pipe with couplin
Iron handle, length 2 ft, with central eye and set screw. (*f*) Sand pump or slud
diam, 2 ft long, with a leather flap valve. (*g*) 2 pipe tongs. (*h*) Oil can, 25
flat file, spring balance, water bucket. The auger was turned by 2 men, sta
opposite sides of hole. Enough water was added to soften the material. Hard
was penetrated with the chisel bit. Two men operated to depth of 25 ft; 3 me
35 ft, a rough frame 15 to 20 ft high being built for the third man. Table 1 rec

Table 1. Prospecting with 2-in Auger and Chisel Bits

Hole	Material	Total depth, ft	Number of men	H w
1	Sand, gravel, clay, and ore	16	2	
2	Surface, ore, clay	40	2 3	
3	Clay, sandstone, and flint	18	2	
4	Mostly clay	29	2	
5	Clay and ore	26	2	
6	Sand, ore, clay, and sandstone	19	2	
7	Sand, sandstone, clay, and flint	52	2 3	
8	Sand, sandstone, clay, ore, and flint	63	2 3	

work done by one of these outfits in favorable circumstances. A similar outfit w
in prospecting residual iron deposits at Moa and Mayari, Cuba. B. Halberst
a similar outfit for coal prospecting, costing about $25.

At Toronto, Can, in similar work, the cost of 30 to 70-ft holes with labor at $2 p
was 29 to 55¢ per ft, respectively in clay and made ground. Supplies and bla
repairs comprised 15% of total cost per ft.

E. Low states (2) that boring 450 6-in holes, averaging 13.26 ft deep, with pod
in earth, clay, sand, and gravel, 3 men to a crew, the aver progress was 40 ft p
length of moves 200 ft; cost about 18¢ per ft.

Post-hole diggers are used alone or in connection with earth augers.

New Market Zinc Co, Tenn, used an ordinary post-hole digger for prospecting a
of zinc carbonate and silicate, in tough, residual dolomite clay, 10 to 75 ft thick
strap by which the handle was attached to the cutting blade was reinforced. Upp
of handle was threaded and attached to a 10 or 12-ft piece of 1-in gas pipe, making
handle. From a ring at upper end of gas pipe a rope passed over the pulley in an or
3-leg derrick, by means of which the digger was churned. In ordinary soil half-a
drops filled the tool, which will extract 3 or 4-in lumps of ore. Boulders were bro
dynamite and then churned up. Two men made 20 to 40 ft per day.

3. Spring-pole Rig. Hand Churn Drill

Spring-pole rig is a primitive form of churn drill, useful where fuel and
are scarce, transportation difficult, labor cheap, and where few holes are

of low first cost, maintenance, and ill in cost per ft of hole.

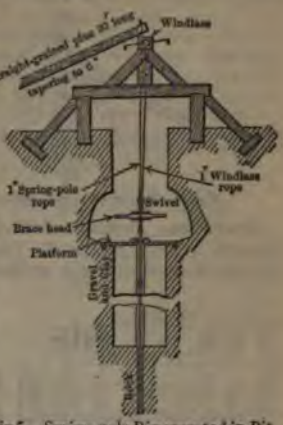

Fig 5. Spring-pole Rig operated in Pit

long; 1 iron-bound oak pipe driver, 4 ft 2-in drivepipe, 1, 2, 3, 4 and 5 ft long; d and 1 driveshoe; 1 pipe lift; 1 chain ch for pipe 5 in outside diam; 1 alligator ch for pipe 5 in outside diam; 2 screwjacks osening casing; 2 clamps for holding and g pipe with jacks; 1 rod 1 ft 8 in, 1 rod 8 in and 1 rod 5 ft 8 in between shoulders; ds 11 ft 8 in between shoulders (all rods or 1.5-in round iron, with 1.25-in square on near joints). 6 3-in drills, of 2-in octa- steel; 2 swivels, for windlass and spring- ropes; 1 brace head; 1 reamer; 1 foot ch; 1 hand or disconnecting wrench; 1 sand p for light drillings and 1 for coarse or pasty ings; 1 grab for drills and rods; 1 grab for p; 2 water barrels; black oil for rod and joints; blacksmith's and pipe fitting and ading tools; carpenter's tools. Fig 6 shows ral of the tools used. Cost of a rig for 250 ft not exceed $400 (3).

peed of drilling varies with size and th of hole.

shales and soft sandstones, 1.5 and 2-in s, 8 to 10 ft per shift for first 100 ft, and 8 ft for second hundred, are fair averages. he same rocks 3-in holes will advance 7 to per shift for the first 100 ft and 4 to 5 ft for nd hundred.

Hand churn drill is used for prospect- nber of holes to be drilled, or inaccessi- the portable power churn drill (Art 7).

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Outfit for a drill used in prospecting on Seward Peninsula, Alaska (Fig 8): 1 1 set driving blocks; 1 set wooden, and 1 set iron pipe clamps; 1 sand pump; head; 1 driveshoe; 1 tripod; 50 ft, 1-in Manila rope; 2 16-in jacks; 1 Stillson 1 shovel; 5 5-ft len drivepipe; 1 singl double block; 1 bit-h lb hammer; 1 monke 3 pipe tongs; 1 axe chisel; 1 can grea iron work requires hr labor by a machi blacksmith. Cost of fit at Nome, Alask about $200.

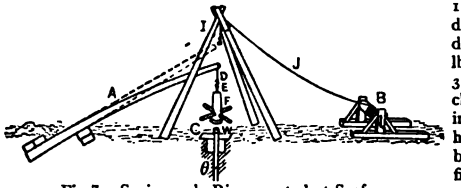

Fig 7. Spring-pole Rig operated at Surface

Operation. Drilling and driving tools are suspended by a 1-in rope through a block in tripod. Driving is done by ordinary Keystone blocks, clamped near top of drill. Total falling weight of the hamm

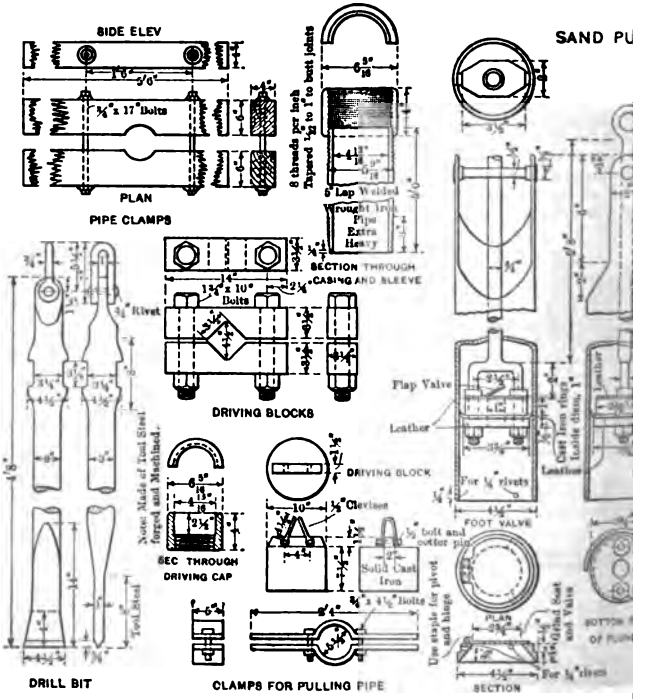

Fig 8. Details of Hand Churn Drill used for Prospecting Shallow Gravels at Nome,

formed is 125 to 175 lb. Hammer is raised 2 to 3 ft per stroke by 2 men ing on free end of the suspending rope. About 20 blows per min are st First joint of casing, with driveshoe and drivehead, is set in a shallow hol tamped plumb. It is then driven some multiple of 6 in, according to pros

off, core drilled up, and pumped out g.)

ulled. It is first loosened by turning with d a long pole used as a lever. One ft to rding to position of fulcrum. The chain ck when clamps are removed. For short k up the tripod by the legs and carry it. moving 100 ft is often done in 20 min.

rs' costs at Nome with this drill, as ne drill on similar work.

tting with Hand Churn Drill at 1909–1911

Drilling		Pulling & moving		Delays		Cost, $ per ft
Total hr	% of total hr	Total hr	% of total hr	Total hr	% of total hr	
1018.0	84.0	123.7	10.3	68.3	5.7	2.02
755.1	78.7	139.1	14.5	66.1	6.8	1.79
356.3	68.1	96.2	18.4	70.6	13.5	1.95

1910. [2] Drilled in 1911.

work in 1908 averaged 1.2 ft advance per pth, 560 ft. Time distribution: moving casing, 6.8%; delays, 7.3%. Wages, $5

, but using a 3-in by 5-ft auger stem and e required on the rope. In testing placer tives, he averaged 21 ft per working day

Banka Drill

ing especially useful in remote regions, difficult, and labor cheap.

eavy flush-joint casing, shod with a pper end, projecting from the ground, men. To this platform is attached a are revolved by man or horse power. ming and weight of men; for ordinary by a driving ram, weighing about 200 casing is being rotated. The core is y a drill pump (Fig 9) on the end of screw joints. Special pumps, drills, nditions. For depths exceeding 30 ft, to handle, and a spring counterbalanc- d with a lever, as described in Art 3; s operation. Complete equipment for about 2 500 lb and costs $1 400 f o b, lb, cost $2 500. This equipment can ing not over 75 lb each.

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Cost and speed. C. M. Rolker, prospecting on ISLAND OF BANKA in toug 11 to 32 ft deep, with raw crew, 4 men on platform and 6 on ground, averaged ft of 3.5-in hole per hr. Actual boring occupied only 8 to 30% of total time

Following is a record of work done in COLOMBIA, S A, in a jungle where could not be used. Day 1 : Moved across river and made 14 ft in top soil a gravel. Day 2: Finished hole 1, 2.5 ft to bedrock, 27.5 ft total depth. Pulled moved 100 ft before noon. Sunk 17 ft to bedrock in 4 hr. Pulled casing and moved to hole 3 and made 9 ft in overburden. Day 3: Finished hole 3, 24 ft total depth. Pulled casing and started hole 4 by 2 p m. Drilled 12 ft overburden and 10 ft sand and gravel by 5 p m. Day 4: Finished hole 4, total depth 28 ft to bedrock. Pulled casing, moved 300 ft to hole 5 and started by noon. Made 22 ft by 5 p m, passing through buried tree. Day 5: Finished hole 5 to 28 ft. Pulled casing and began hole 6. Made 14 ft in overburden and 9 ft in gravel. Day 6: Finished hole 6, to depth of 32 ft. Moved across and 1 mile up river and at 2:45 p m started hole 7. Made 6 ft in overburden and 9 ft in gravel. Day 7: Finished hole 7, total depth 29 ft. Moved 50 ft, sunk hole 8, 22 ft to rock. Started hole 9 after 50-ft move and made 6 ft in top soil. Summary: 7 days; 213.5 ft drilled; 30.5 ft per day.

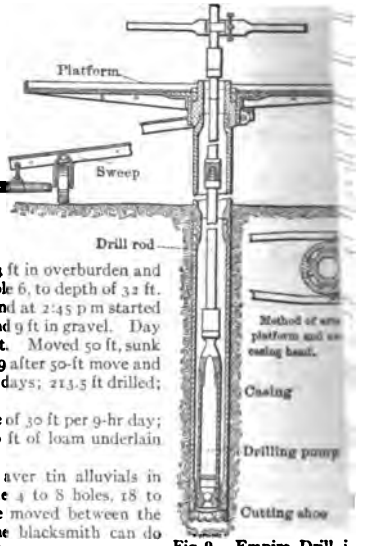

Fig 9. Empire Drill i ation

J. Chisholm reports aver advance of 30 ft per 9-hr day; 5 to 30-ft holes; 6-in drill; 9 to 10 ft of loam underlain by gravel.

E. Middleburg reports that, in aver tin alluvials in Banka, native crews of 8 men make 4 to 8 holes, 18 to 30 ft deep, in 8 hr, when distance moved between the holes does not exceed 60 ft. One blacksmith can do repairs and sharpening for 8 outfits.

R. E. Smith and H. G. Hann (4) give the following figures on work with two 4-in Empire drills in Siberia, when the thermometer re 0° to −45° F. Total depth, 6 696 ft; total time, 193.5 days. Percent total time 81.1, as follows: drilling, 63.7; putting on pipe, 2.9; pulling pipe, 5.0; clearing sn brush, 1.8; moving from hole to hole, 2.8 (aver time, 17 min); moving from line t holes over distance 6 083 ft (aver time per move, 2 hr 5 min), 3.1; lost time while 1.8. Percent total time lost, 18.9, as follows: moving camp (twice), 1.8; storm holidays, 5.2; resting screws, 3.1. Crew of 7 men on each drill. Aver time p 8 hr 20 min. Aver advance per hr of total time per drill, 2.1 ft.

CHURN DRILLING

5. Oil-well Rig and Its Modifications

Early forms of drills employed solid rods between the bit and surface anism. But, in deep holes, the weight to be lifted becomes excessive, wit bility to buckling or breakage of the rods on striking the blow. To pr this difficulty numerous free-fall mechanisms have been introduced in E such as those of Dru, Kind and Fabian, by which the rods are kept alw tension and the striking weight is constant (5). In America, developmer followed 3 lines: (*a*) substitution of a cable for the metal rods; (*b*) use o ing link or jars between rod and bit; (*c*) use of wooden rods. The first t these features are found in the standard rig; the third, in the Canadian rig

...mmonly
...gas wells.

...; A_1, A_2, A_3, A_4, sandlife; ...; E, bracket post; F, tail post; ...; H_1, H_2, jack-post braces; I, sampson

Oil-well Rig

...lking beam; J_1, walking-beam cap; K, ...d handle; L, bandwheel; M, pitman; ...sills; P, P_1, P_2, P_3, P_4, P_5, derrick floor ...and brace; S, headache post; T, U, V, ...er; X, crown block; Y, sandline pulley

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block; *a*, flanges; *b*, shaft, crank and wristpin; *c*, saddle and side irons; *d*, stirrup
lever; *g*, brake band; *i*, crown pulley; *j*, sandline pulley; *k*, reverse lever; *l*, engi
l_1, l_2, engine pony-sills; *m*, m_1, mudsills; *n*, engine-block brace; *o*,
sandline; *r*, cable; *s*, bull rope; *t*, t_1, telegraph cord and wheel;
cord; *v*, bandwheel tug pulley; *w*, sandreel friction pulley.

Fig 11. String of Tools for Standard Oil-well Rig

Rig irons consist of: bandwheel shaft, with crank and flanges
jack-post boxes, saddle and side irons for supporting the walking be
rup for connecting pitman to walking beam; sandline and derrick
gudgeons for bullwheel and sandreel. Size of the outfit is denomi
the diam of bandwheel shaft; standard sizes are 3.5, 4, 4.5 and

String of tools (Fig 11) is attached to walking beam,
feed regulated, by means of the temperscrew (Fig 12). T
ing rope is wrapped with cloth or raveled rope to prevent
and wearing in the clamp by which it is suspended.

Table 3. Details of String of Tools, Oil-well Rig

Tool	Length, ft	Diameter, in	Weight, lb	
Bit.....	3.5 to 6	4 to 20.5 a	170 to 3 750	a is
Auger stem.	16 to 48	2.75 to 6	20 to 98 per ft	of
Jars.....	about 5.5	3.5 to 8	100 to 800	ed
Sinker bar..	6 to 16	2.75 to 6	20 to 98 per ft	ga
Rope socket	2.5 to 4	2.75 to 6	40 to 350	bit

Derrick is high enough to allow string of tools to swing clear of hole. Standard height, from top of sills to under side of crown block, is now from 72 to 84 ft. Base is generally 20 ft square.

Load on drilling cable, under ordinary conditions, is that of the string of tools, or a string of casing. Possible max load on the derrick is twice the ultimate strength of drilling cable. This load must be distributed evenly by the crown block to the derrick legs. Fig 12 gives sizes for the important timbers in an 84-ft, spike-joint derrick. Other designs by the oil-well supply houses are built of bolted wooden members, gas pipe or structural steel. These derricks can be set up and dismantled with little, if any, destruction of material, and in less time than the spiked derrick.

Fig 12. Method of pending Drilling R

Hoisting and transmission machinery. Bandwheel shaft, which distributes the power, is actuated by belt from the engine pulley to the bandwheel. From it the bullwheel is driven by a crossed rope running in grooved pulleys, the sandreel is driven by friction from the bandwheel, and the walking beam is actuated through crank, wristpin and pitman. Bullwheel carries drilling cable and casing line. It is thrown out by throwing off its crossed driving rope. When not connected with the engine it is controlled by a band brake. Sandreel is used to run the bailer in and out of the hole. When the sandreel is pulled forward, the friction pulley engages the bandwheel and bailer is hoi

ng the friction pulley against a friction
nd shafts for bullwheel, bandwheel and
ly houses.
graph cord running from the telegraph
lever is controlled by a rod running

wire-rope, is used for drilling cable.
er, where the buoyancy of hemp rope
string of tools.

xtra strong cast-steel hoisting rope. For
e rule. For weights and strengths of ropes

slide valve, and link reversing gear;
n. Crankshaft carries a pulley on one
nd a heavy flywheel on other end. An
akes the belt pull.

type, with locomotive fire-box; rated
the common fuels. Little attention is

re essential in most wells. These terms
pipe is the common name for the lining
lining the well to any
drivepipe is a special
driven into the hole.
mmonly used. Butt-
eavy strains to which
e casing is of 2 layers
surface of outer layer
nk to allow a close fit.
o the well, the pipe is
d pick, to prevent pulling apart. Riveted
ctor pipe. It can be driven lightly, but
l wooden pipe, made of beveled edged strips
vith iron straps, is often used for conductor
is also used. Sleeve-coupled pipe (Fig 13)
all depths. Subjected to hard driving the
the ends of the lengths do not butt together
gives sizes and weights. Drivepipe has
cut that the pipe ends butt in the couplings.
to prevent break-
weights. Besides
lush-joint and in-
times used (6). A
ide diameter than
n lower end of a
pe, to protect the
is large enough for
to, a driving head
screwed on the upper end of the string, to
and protect the pipe.

Fig 13. Sleeve-coupled Casing Pipe

Fig 15. Driving Shoe

up. To frame and erect a standard rig with
derrick requires about 4 days' work with a
men, timbers and all supplies being on the
To dismantle and load the same rig for mov-
To unload and erect again, about 3 days.
d costs about $200. Operators rarely move

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Table 4. Dimensions and Weights of Sleeve-coupled Casing Pip

Size, in	Diameter		Thickness, in	Weight per foot		Threads per in	Couplings	
	External, in	Internal, in		Plain, lb	Threads and couplings, lb		Diam, in	Length, in
2.0	2.25	2.050	0.100	2.296	2.34	14	2.714	2.625
2.25	2.50	2.284	0.109	2.759	2.82	14	2.964	2.625
2.5	2.75	2.524	0.113	3.182	3.25	14	3.214	2.875
2.75	3.00	2.768	0.116	3.572	3.65	14	3.464	2.875
3.0	3.25	3.010	0.120	4.011	4.10	14	3.771	3.125
3.25	3.50	3.250	0.125	4.505	4.60	14	4.021	3.125
3.5	3.75	3.492	0.129	4.988	5.10	14	4.271	3.125
3.75	4.00	3.732	0.134	5.532	5.65	14	4.521	3.125
4.0	4.25	3.974	0.138	6.060	6.20	14	4.771	3.625
4.25	4.50	4.216	0.142	6.609	6.75	14	5.021	3.625
4.25	4.50	4.090	0.205	9.403	9.50	14	5.021	3.625
4.5	4.75	4.460	0.145	7.131	7.25	14	5.271	3.625
4.5	4.75	4.364	0.193	9.393	9.50	14	5.271	3.625
4.75	5.00	4.696	0.152	7.870	8.00	14	5.521	3.625
5.0	5.25	4.944	0.153	8.328	8.50	14	5.828	4.125
5.0	5.25	4.886	0.182	9.851	10.00	14	5.828	4.125
5.0	5.25	4.886	0.182	9.851	10.00	11.5	5.800	4.125
5.0	5.25	4.768	0.241	12.892	13.00	11.5	5.800	4.125
5.0	5.25	4.648	0.301	15.909	16.00	11.5	5.800	4.125
5.187	5.50	5.192	0.154	8.792	9.00	14	6.078	4.125
5.187	5.50	5.044	0.228	12.837	13.00	11.5	6.050	4.625
5.187	5.50	4.892	0.304	16.870	17.00	11.5	6.155	5.125
5.625	6.00	5.672	0.164	10.222	10.50	14	6.664	4.125
5.625	6.00	5.620	0.190	11.789	12.00	11.5	6.636	4.125
5.625	6.00	5.552	0.224	13.818	14.00	11.5	6.636	4.125
5.625	6.00	5.450	0.275	16.814	17.00	11.5	6.636	4.125
6.25	6.62	6.287	0.169	11.652	12.00	14	7.308	4.625
6.25	6.62	6.255	0.185	12.724	13.00	14	7.308	4.625
6.25	6.62	6.257	0.184	12.657	13.00	11.5	7.280	5.125
6.25	6.62	6.135	0.245	16.694	17.00	11.5	7.280	5.125
6.25	6.62	5.913	0.356	23.835	24.00	11.5	7.280	5.125
6.625	7.00	6.652	0.174	12.685	13.00	14	7.692	4.625
6.625	7.00	6.538	0.231	16.699	17.00	11.5	7.664	4.625
6.625	7.00	6.538	0.231	16.699	17.00	10	7.642	5.125
6.625	7.00	6.450	0.275	19.751	20.00	10	7.699	6.125
6.625	7.00	6.334	0.333	23.711	24.00	10	7.699	6.125
7.25	7.62	7.263	0.181	14.390	14.75	14	8.317	4.625
7.625	8.00	7.628	0.186	15.522	16.00	11.5	8.788	5.125
7.625	8.00	7.528	0.236	19.569	20.00	11.5	8.788	5.125
8.25	8.62	8.249	0.188	16.940	17.50	11.5	9.413	5.125
8.25	8.62	8.191	0.217	19.486	20.00	11.5	9.413	5.125
8.25	8.62	8.097	0.264	23.574	24.00	11.5	9.413	5.125
8.25	8.62	8.097	0.264	23.574	24.00	8	9.358	6.125
8.25	8.62	8.003	0.311	27.615	28.00	8	9.358	6.125
8.625	9.00	8.608	0.196	18.429	19.00	11.5	9.788	5.125
9.625	10.00	9.582	0.209	21.855	22.75	11.5	10.911	6.125
10.0	10.75	10.192	0.279	31.201	32.51	8	11.958	6.625
10.0	10.75	10.146	0.302	33.699	35.00	8	11.958	6.625
10.625	11.00	10.552	0.224	25.780	26.75	11.5	11.911	6.125
11.625	12.00	11.514	0.243	30.512	31.50	11.5	12.911	6.125
12.5	13.00	12.482	0.259	35.243	36.50	11.5	14.025	6.125
12.5	13.00	12.278	0.361	48.730	50.00	8	14.085	7.125
13.5	14.00	13.448	0.276	40.454	42.00	11.5	15.139	6.125
14.5	15.00	14.418	0.291	45.714	47.50	11.5	16.263	6.125
15.5	16.00	15.396	0.302	50.632	52.50	11.5	17.263	6.125

Permissible variation in weight is 5% above and 5% below.

a nailed derrick more than 4 times, though there are records showing 6 or 7 wells d
with one rig where great care was used in taking down the rig and placing it on p
foundations. Time to erect a rig with bolted wooden or steel derrick is less than

d Weights of Drivepipe

foot †	Threads per in	Couplings		
treads coup-gs, lb		Diam, in	Length, in	Wt, lb
.730	11.5	2.923	3.625	2.380
.906	8	3.486	4.125	3.748
.705	8	4.111	4.125	4.493
.294	8	4.723	4.125	5.973
.995	8	5.223	4.125	6.740
.758	8	5.723	4.125	7.439
.989	8	6.410	5.125	11.871
.408	8	7.473	5.125	13.956
.021	8	8.474	5.125	15.955
.495	8	9.588	6.125	24.343
.303	8	9.588	6.125	24.343
.334	8	9.882	6.125	31.320
.711	8	10.588	6.125	27.035
.631	8	11.950	6.625	40.108
.628	8	11.950	6.625	40.108
.785	8	11.950	6.625	40.108
.953	8	12.950	6.625	43.664
.358	8	13.950	6.625	47.220
.067	8	13.950	6.625	47.220
.849	8	15.438	7.125	66.024
.005	8	16.438	7.125	70.533
.161	8	17.438	7.125	75.043
.000	8	18.675	7.125	91.746
.000	8	19.913	7.125	109.669
.000	8	21.913	7.125	121.298

in weight is 5% above and 5% below.

A crew of 4 men can erect a bolted wooden are placed. Placing the sills will not take is not excessive. Four men can erect a steel day. Time n both cases.

to 200 ft of l down on a ctuating the one or two ie bullwheel ng free end

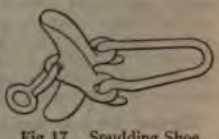

Fig 17. Spudding Shoe

tightening and loosening the cable on ools are raised and dropped; (*b*) end of o the bullwheel and a "jerkline" passed SPUDDING SHOE (Fig 17), placed on the oove the bullwheel. The weight of the eel band brake and the churning motion the drilling cable.

18) is used for spudding through surface soil. rock the bit shown in Fig 19 is used. While to the hole to make a mud which will hold en sludge becomes so thick that fall of the bit vn, and hole cleaned out with BAILER (Fig 20). r bar and jars. If surface soil will stand with- s drilled "open." When rock is reached, the

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tools are withdrawn and conductor pipe is lowered into hole and seated firmly in rock vent subsequent entry of surface soil. If soil caves, conductor must be driven ahea

Driving pipe. A shallow hole is excavated and in it is placed th length of drivepipe with a driveshoe on lower end and drivehead on upp The pipe is plumbed and earth filled in around it. Driving i by a drive block running in guides in the derrick or, more com by means of drive clamps (Fig 8) bolted to a square on the Weight is raised and dropped by either of the methods de under Spudding. When the pipe has been driven to refusal th is churned up and hole bailed out. Additional lengths are to the pipe as necessary.

Fig 19. Spudding Bit for Rock

Drilling. After the hole has been spudded to such depth there is about 100 ft of cable in the hole, the walking beam is br into play. The drilling cable is wound onto the bull-wheel, pitman connected with one of the inner holes on crankshaft, temperscrew hung on forward end of walking beam, the tools rested on bottom of hole, and the cable let out until there is about 4 in slack in the jars. The cable is wound with marline at the point where the temperscrew clamps grip it, temperscrew is clamped on, and the bullwheel slacked off, thus transferring the weight to the walking beam. About 25 ft of cable is run off the bullwheel and thrown on the floor to prevent lashing. On starting engine, the tools are picked up with a shock as the links of the jars come together, thus preventing the bit from sticking in the hole. Downward stroke of the string of tools is quicker than the movement of the end of the walking beam, because of the spring in the cable, action of the tools being that of a weight suspended by an elastic cord.

As hole deepens, the bit is fed downward by the temperscrew. For the first 200 ft or so the cable is twisted to keep the hole round. Beyond this depth, the twist in the cable itself is sufficient to bring the bit to a new seat at each stroke. When temperscrew has run out, it will usually be advisable to pump out the cuttings. Slack of the drilling cable is taken up on bullwheel, temperscrew clamps loosened, temperscrew run back, pitman thrown off the wristpin, and rear end of walking beam lowered. Tools are hoisted, swung to one side, and the bailer run in. After hole is pumped out, the tools are again lowered, attached to walking beam, and drilling is resumed.

Fig 20. tail B

Casing the hole presents varied problems. In hard rock that stands well, the only casing necessary is the lining for the finished well; this is put in after drilling is completed. Casing block hung in the derrick and the casing line run onto the bull-w A casing shoe is screwed on the lower end of a length of pipe, a coupling on the other end, an ELEVATOR (Fig 21) is locked ar the pipe below the coupling and the casing lowered through a RING or SPIDER (Fig 22), resting on the conductor, until only a few inches project. Slips are then dropped into the pipe ring, the elevator is unlocked, and another length picked up on the casing line and swung into place. This length is screwed into the projecting coupling and tightened by chain tongs. Casing clamps are then loosened and the string lowered by the bullwheel brake. This opera is repeated until the full string is in the well.

Fig 21. Casing-pipe Elevator

Fig 22. Casing-Ring

If the hole will stand for a few hundred
lescribed; after which boring is continued
a number of succes-
he hole becomes too
nder-ream to allow
ther.

g a hole already
ing, the reamer
backed by pow-
e reamer. While
he casing the bits
eir recesses. On
he bits are forced
le.

Fig 23. Under-reaming Bit

tools from the hole.
icking of the tools,
s; (*b*) breakage of
ing of a joint in the
tion; (*d*) breakage
(*f*) sticking of the bailer; (*g*) "freezing" or
ring of casing; (*i*) loss of some small article
arring with the drilling jars fails to loosen
the hole on a separate line and churned
Failing this, the rope may be cut off at the
d to the sandline through pipe jars and a
d on a string of tools consisting of long-
et. The slip socket (Fig 27), like most
hole (6). When the slip socket has en-
e string, but the blows are much heavier
the longer stroke (12 to 36 in) and the
KEN DRILLING CABLE or sandline may usu-
or the rope grab (Fig 29). If part of the
be picked up by lowering a section similar
ing, and screwing up the joint again; or,
rt. If the upper end of the unscrewed or
walls of the hole, a spud, similar to the
lost part, so that it stands straight in the
11) is used to prepare the end for the grip-
ocket may then be used to pick up the lost
by heavy downward blows of the jars; it
d. A LOST BAILER can often be picked up
he boot latch pulls off the bail, a mandrel
overy. The upper portion of the bailer is
ownward blows of the fishing jars, and is
n the tool is hoisted. FROZEN CASING can
ad hoisting with jacks. When this fails,
safe strain with jacks, then taking a grip
ly. SMALL ARTICLES in the hole are often
the common fishing tools above described,
ly houses, and many more have been de-
r details of fishing operations, see Bibliog
it may often be drilled up, or blown into
or the hole may be diverted at some point
by obstructing the hole at the desired point
ock being drilled (flint, steel or even old
string of tools of smaller diam than the
pened to a chisel edge. A hole is rarely

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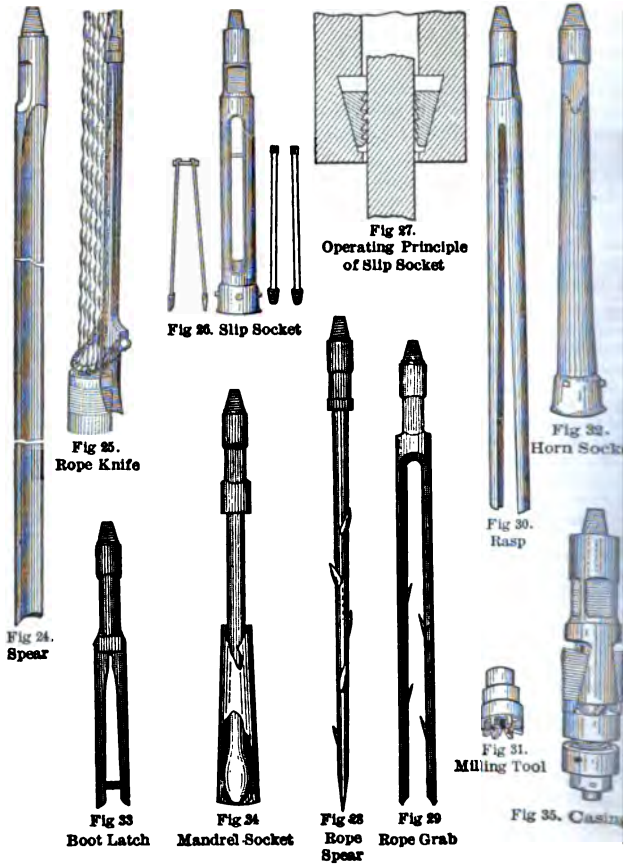

Fig 24 to 36. Oil-well Fishing Tools

California rig (Fig 37) is a modification of the standard rig to n special conditions in the California oil fields. Long strings of heavy sometimes have to be handled rapidly to prevent caving. This is done addition of the calf-wheel *A* and 2 to 5 extra pulleys *B* in the crown b The casing line, a 6-strand 19-wire rope, passes from *A* to one of the pu and is reeved from them through a multiple block *D* carrying the casin Strings of casing weighing over 90 000 lb have been handled in this w carry the heavy loading the derricks are generally strengthened by d the corner posts for a part or all of their length, and by increasing size and braces; or a steel rig is used.

Canadian rig differs from the standard in the arrangement of hoisting an mission machinery, and the use of wooden rods instead of rope. Formerly, n

Fig 37. 80-Ft California Drilling Rig

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used also for the sand pump, but a reel and wire rope are now employed. Fig 38 Canadian rig, with drum *A* for hoisting rods, sandreel *B*, and having a bullwheel f cable-tools when desired. wheel *C* is belted to engine. pulleys on the driving sha belts are run to pulleys on th which carry spools *A* and operating the spools belt ti are applied by levers *D* and rods are attached to walkin by a chain wrapped sever around end of beam, and the a windlass, which is controll ratchet. Rods are of white a

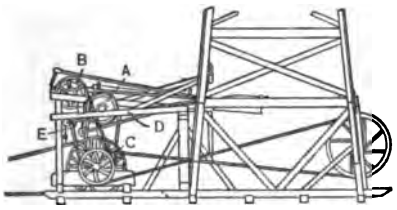

Fig 38. Canadian Well-drilling Rig

2.5 or 3 in diam, about 18

Two poles, joined by straps and fitted at ends with taper-box and pin-joints, for about 36 ft long. String of tools is the same as that of the standard rig. Cana can be dismantled and moved more rapidly than the standard, and pole tools are tageous for fishing; but it is now rarely used except for shallow wells, where frequen are necessary.

6. Speed and Cost of Oil-well Drilling

Controlling factors: (*a*) character of formation to be penetrated a vious knowledge concerning it; (*b*) depth and diam of hole; (*c*) type a of rig and weight of tools; (*d*) cost of fuel, water, and drilling supplies; (pose of hole; (*f*) general considerations, as location, experience of driller, number of shifts per 24 hr, climate, and accidents, discussed under D Drilling. Character of the formation affects speed of advance, and det amount of casing necessary. If much water is present, the buoyant lessens force of the falling tools, and in an oil or gas well the water n excluded. The pioneer wells in any district are drilled more slowly, an cost more than the later wells. Character of formation and depth of being unknown, it is difficult to choose the rig best suited to the work depth to which casing of a given size must be carried, to bottom the the desired diam is unknown, and the progress of the hole must be wat

General considerations. Cable rigs drill faster than pole rigs, bec the time saved in handling tools and pumping out the hole.

R. N. Boyd gives following data for a 1 542-ft well in Galicia, using Canadian rig ing rods, 25 min; letting down sand pump with rods, 14 min; drawing up sand 24 min; changing bit, 3 min; letting down bit, 14 min; connecting to beam, total, 85 min. Using a sandline for the bailer, the time can be reduced to 55 min. operations, with a cable rig to 1 600 ft, consume about 20 min.

Comparative speeds with California and standard rigs in the same for are less important than the fact that with California rig in a caving for the casing may be moved instantly, thus preventing an expensive cave o ing of the casing. This is impossible with standard rig. Heavy rigs an increase speed of drilling and of handling casing. For deep holes t creased speed more than offsets the greater time, labor and money invo erection. In established oil fields, the prices of natural gas or crude o for fuel vary so greatly as to vitiate cost estimates. Where coal must b 3.5 to 10 lb per hr per boiler h p is generally sufficient. Water may cos than fuel. In the California fields, where water at the well has cost as as 1¢ per gal, the daily water charge is $3 to $10. Cost of drilling suppli depreciation on the rig depend on depth of hole and character of servic deep holes, where the service is hard, there is almost no salvage.

Requa gives following figures for DEPRE-, aver time 88 days: bullwheel, bandwheel, ing in excess of water string, 60%; drill

shallow, due to greater time required out of the well, fewer strokes per min, ng. Large diam holes cost more than g. Holes less than 5.625 in diam drill istance of tools and cable in the well. is an expensive process (see Cementing, countered under pressures that make en drilled with the hole full of water, ing in a decreased drilling rate.

were furnished by companies that have nd wells in the respective districts.

the formation comprises 400 to 450 ft of Trenton sandstones and shales. Contract ntractor furnishes the rig and 45¢ when the equipment averages $1 250 to $1 500.

ormation consists of shale, slate, and shells gara limestone extends to 3 200 or 3 400 ft, is $1.00 per ft, the company furnishing the es $8 000.

800 ft, the overlying formation being chiefly stone and limestone. Contract price is 80¢ ost of well and equipment averages $1 800. ational drilling machines (Oil Well Supply nost of them being finished at 6.25-in diam.

are interspersed with soft and hard streaks, sing can be set. Each well bears the total one well to another. "Combination" rig ccording to which is best adapted to char-

ell Drilling in California (1912–14)

n d.	Depth, ft	Days drilling	Aver ft per 24 hr	Cost per ft, labor and supplies	Total cost
5	2 692	185	15	$5.91	$27 486
5	3 021	154	19	6.74	38 370
5 } 5 }	2 081	232	9	8.06	27 302
5	2 220	52	43	4.15	17 679
5 } 5 }	2 104	73	29	4.82	17 856
5	2 517	122	22	6.10	29 292
0	2 504	109	23	5.42	25 720
25	3 100	235	13	6.49	36 202
0 } 5 }	2 691	221	12	7.91	35 792
5	3 628	256	14	7.98	47 931
5 } 5 }	3 557	265	13	8.52	47 822

bination rig at others.

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West Virginia. All wells in sandstones, limestones, slate and coal. Pipe
with wooden housing, boiler and steam engine, or gas engine, charged to each we
to 40 quarts of nitroglycerine per well used for shooting, where desirable. A
drilling time for 299 wells in these counties in 1914 was 46 days; average cost co
$7 200.

Table 7. Speed and Cost of Oil-well Drilling in West Virginia

County	Building rig, days	Drilling, days	Depth, ft	Advance per 24 hr, ft	Contract price per ft, drilling only	Casing		
						Diam, in	L'n'th, ft	
Harrison, No 1	16	60	2 768	49.1	$0.95	10	204	$
						8¼	1 026	
						6⅝	1 922	
						5⅜16	2 638	
Harrison, No 2	6	88	3 168	36.0	1.05	6⅝	2 427	
						5⅜16	2 923	
Kanawha, No 1	12	49	1 884	38.4	1.15	10	28	
						8¼	440	
						6⅝	1 370	
						5⅜16	1 537	
Kanawha, No 2	11	35	2 173	62.1	1.15	6⅝	1 785	
Lewis, No 1....	5	42	2 259	53.8	1.10	8¼	872	
						6⅝	1 867	
Lewis, No 2....	19	53	2 527	47.8	1.10	10	225	
						8¼	1 145	
						6⅝	2 090	
Marion, No 1...	7	91	3 566	39.3	1.05	10	23	
						5⅜16	194	
Marion, No 2...	6	74	3 277	44.3	1.05	10	215	
						6⅝	2 410	
Monongahela, No 1.....	5	56	3 021	54.0	1.25	5⅜16	2 697	
						10	246	
						8¼	1 285	
						6⅝	2 198	
						5⅜16	2 557	
Monongahela, No 2.....	6	111	3 182	28.7	1.25	4½	270	
						13	37	
						10	414	
						8¼	1 465	
						6⅝	2 206	
						5⅜16	2 885	
						4½	268	
Roane, No 1...	8	43	2 028	47.1	1.00	10	215	
						8¼	810	
						6⅝	1 886	
Roane, No 2....	11	39	2 163	55.5	1.00	10	696	
						8¼	1 084	
						6⅝	2 036	
Wetzel, No 1...	26	80	3 033	37.9	1.05	6⅝	2 141	
						5⅜16	2 585	
Wetzel, No 2...	11	64	3 383	52.9	1.05	....	....	

All struck oil except Marion No 1 and Wetzel No 2. All 10-in casing was pull
Lewis No 2, and Monongahela No 2; all 8¼-in from Harrison No 1, Kanawh
and Monongahela No 2; all casing was pulled from Wetzel No 2.

ell Drilling in Mid-Continent Field
ary formation)

used	Diam hole completed, in	Aver ft per 12 hr	Cost per ft, ¢	Total cost, drill'g and equip't
able	6.25	54	65	$ 700
able	6.25	44	75	1 575
able	6.25	47	85	2 200
dard	6.25	63	70	3 100
dard	6.25	34	70	3 000
dard	5 3/16	21.5	150	9 000

l Drilling in Texas and Oklahoma

	Size finished, in	Advance per shift, ft	Total time, from rig on ground to water string set, days	Cost per ft, labor and supplies, ¢	Total cost, including equipment, $
	4	125	3	13	862
	5 3/16	15	46	24	7 916
ol	4	15	57	50	14 384
ol	8 1/4	15	32	44	4 812
	4	150	25	10	1 165
ol	6 1/8	15	46	47	8 767
ol	8 1/4	15	34	53	4 205
	5 3/16	15	60	22	8 407
r-)	5 3/16	25	16 (j)	38	949

le, and gumbo. *c.* Shale, limestone, slate, le. *e.* Shale and sandstone. *f.* Sandstone, nd sandstone. *h.* Sandstone, shale, lime- *j.* One shift.

Churn Drills

ed for shallow wells, prospecting, and the standard rig mounted on a strong

andard rig, but are lighter. The string consists of rope socket, stem and bit. better to use jars, though they are often y be done with a 5⅝-in bit. With this with portable rig can be used, and prog- naller sizes. Common bit sizes are 10, part of a hole with 5⅝-in bit, holes are tem used, larger bits than 10-in tend to smaller than 4.5 in limit the tools to is greatly decreased by rope friction on

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sides of hole, and progress is lessened. Rock bits are made blunt and
to crush the rock. Placer bits are long and thin and should be kept

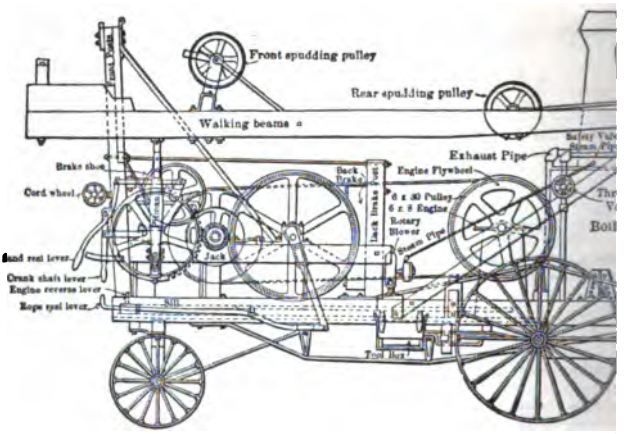

Fig 39. Keystone Portable Drilling Rig

the object being to loosen rather than to disintegrate gravel. Mother H
bits, blunt, and almost filling the hole, are used to keep holes straight.
holes become crooked they can
straightened by throwing in ca
hard boulders, or rope, to a poin
the crook and re-drilling. It
about 1 hr to sharpen and temp
About 0.5 ton blacksmith coal
sumed per month for this purp

Table 10. Comparison of Costs of Blast-hole Drilling with Steam and Gasoline Churn Drills

Year	Month	Steam drills, cost per ft	Gasoline drills, cost per ft
1912	Jan	$0.354	$0.270
	Feb	0.388	0.260
	March	0.266	0.277
	April	0.343	0.283
	May	0.306	0.307
	June	.....	0.321
	July	0.866	0.288
	Aug	0.513	0.300
1913	Aug	0.450	0.407
	Sept	0.372	0.362
	Oct	0.449	0.354
	Nov	0.440	0.358
	Dec	0.551	0.437
1914	Jan	0.485	0.387
	Feb	0.679	0.470
Aver (monthly)		0.462	0.339

Note. — Rock mostly shattered quartzite and porphyry.

Power. Steam, gasoline, electric
compressed air are used for portab
Steam has the advantage that th
can exert full force from rest; a
feature in the case of stuck tools o
In general, the steam drill is co
more reliable than the others. A
engine, giving a resultant vertical vi
is better than a horizontal engine, th
tions of which are at right angles
caused by the tools. Substitution
pressed air for steam at Panama
in increased drill duty. Electric
convenient and does away with firen
unless the voltage is constant the
adversely affected by speed changes.
line drills reduce amount of fuel to b
as compared with a steam drill.

duction in haulage costs has a direct effect on the cost of drilling as shown in T
Gasoline and electric drills are lighter than steam drills and hence usually less d

uld weigh 12 000 to 14 000 lb. For shallow good service. Friction hoist is best for shal- re the machine may be used for tamping and and out of the hole. For ordinary prospect-

H. P. Bowen, based on experience at oles averaging 600 ft deep in silicified below 400 ft. Repair parts listed are without showing sufficient evidence of nent from a distance. Bits used were ols are f o b, Miami, except in case of iis.

lling Outfit. H. P. Bowen

iler, derrick, 2 crown pulleys, sing	$2 206.28
arings; screw 1.75 in by 5 ft; ools	36.50
ns	13.50
anged to fasten to derrick	35.00
No 4 Barrett lever jacks; 2 2 aulic jacks	348.00
1 coal shovel, 2 oil cans	3.70
elt punch, 6 strings lacing	24.68
pin	11.00
supports, gear for bullwheel,	45.50
on	7.25
with drum	13.50
arm, 2 wristpins for crank -groove pulley, wristpin for	90.60
	4.75
8.25 in and 6.25 in casing.	249.00
illing cable, 0.75 flattened- sandline, with 2 connections	348.66
xtra connections for 2-in rope	998.41
	561.00
see below	2 301.81
" "	39.67
" "	29.60
" "	146.81
" "	118.14
	$7 633.36

ints; 2 rope sockets, 5.5-in OD box; 1 rope ft, 2 with 5.5-in OD box, 1 with 5-in OD box; its; 6 7⅝-in bits; 4 6.25-in bits; 1 4.5-in OD by 14-ft and 4-in OD by 16-ft bailers.

oints: set 5.5-in OD by 24-in stroke jars; s steel-lined slips, 1 medium and 1 heavy; edium and 1 heavy; 1 horn socket each for nter rope spear, with rings for 7⅝ and 10-in uding 1 swivel tap, 1 set pipe jars, 2 sinkers h ball-bearing swivel top concaved on back; ⅝ hole spuds by 10 ft in clear; 1 each 7⅝-in

of 8.25-in steel casing, 11.5 threads, 20-lb; 30 ft of 11¾-in casing, plain ends; 2 each

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casing nipples for 8.25 and 6.25-in casing; 6 6.25-in and 8 8.25-in recessed cou 600 ft of 8-in and 800 ft of 6-in drive pipe, 8 thread, with 6 each extra couplin cessed, and 2 each casing nipples, recessed; 1 each drivehead and 2 each drivesh 6-in and 8-in drive pipe; pulling ring for 8.25-in casing, with 1 set each slips for 8.25 and 6.25-in pipe; pair 4-in square wrenches; 1 set each Fair patent wroug elevators 8.25 and 6.25-in; 1 each heavy (No 35) and medium (No 33) chain ton

Blacksmith outfit: 1 each bit gages, 10-in, 7½-in, 6.25-in; 200-lb anvil; 2 st lipped tongs, 20 and 22-in; 3 bolt tongs, 0.5, 0.75 and ⅞-in; 1 each 20-in tongs gad tongs, 22-in pick-up tongs; 5 each top and bottom swages, ⅜, ½, ⅝, ¾, and 3 hot chisels, 1.5, 1.75 and 2-in; 2 flatters, 2-in and 3-in; 2 hardies, 1.5 and 2-in; hammers, 2 and 3-lb; wire thread brush; 8-lb sledge.

Carpenter tools: 1 axe; 1 hatchet; 1 28-in hand saw; 1 15-in jack plane; hammer; 1 square; 2 wood chisels, 0.5 and 1.5-in; 1 10-in screw driver; 1 12-in with 7 bits 0.5 to 1-in; 1 draw knife; assorted nails, 100 each of 30d and 20d an each of 10d and 8d, 25 lb each 6d and tin roofing.

Machinist and plumber outfit: 4 12-in and 1 8-in bastard files; 2 10-in round files; 2 each 10-in and 12-in flat mill files; 2 each 10-in cant saw files; 2 8-in tri files; 1 each ratchet drills for holes up to 1 in, with 1 each drills from 0.25 to 1 in teenths to fit same; 1 each 6, 8, 10, 12 and 15-in monkey wrenches; 1 each 8, 14-in Stillson wrenches; 1 each 0.25 by ⅜, 0.5 by ⅝, and 0.75 by ⅞-in open-end wr 1 ball-peen hammer; 1 riveting hammer; supply ⅜, ½ and ¾-in pipe and fittin sortment ⅜, ½, ⅝, ¾ and ⅞ bolts, nuts, washers and rivets; 2 bars 0.5-in roun 1 bar 0.25 by 1.5-in flat iron; 2 bars 1⅛-in octagon drill steel; 1 bar each 0.75 an octagon tool steel, 25 lb waste, 50 lb Babbitt metal, 1 soldering outfit, 1 each pi bolt threading outfit; 1 combination pipe vise; 1 hack saw and 12 12-in blades.

Miscellaneous: 2 gasoline torches and 4 lanterns; 3 600-gal water tanks; 2 0.5-i valves; 2 1.5-in globe valves; 1 0.5-in water gage; 6 gage glasses, ⅝ by 12 in; 50 ft rubber hose; 25 ft 0.75-in suction hose; 2 barrels or distillate tanks; 1 iron pulle and 1 double-swivel hook for 2-in rope; 12 ⅝-in and 6 0.75-in black cable clamps.

Following items may be added to Bowen's list: fan parts, extra boiler tubes, valv brasses for spudding arm, piston rings, ⅝-in cable chain 10 ft long, lubricator, i The price of the machine itself is only 20 to 25% of cost of complete outfit. But the work is done near a large supply center, the added first cost will be more tha in reduced delays. Table 11 gives approx prices and weights of Keystone rigs.

Table 11. Approx Weights and Prices of Keystone Portable Churn Drills, without Tool Equipment

Capacity, ft	Power	Hoist	Motive power	Weight, lb	Pric
250	Steam	Friction	Non-traction	7 500	7
250	Gasoline	Friction	Non-traction	7 500	7
350	Steam	Friction	Non-traction	9 500	8
500	Steam	Geared	Non-traction	9 500	8
350	Gasoline	Friction	Non-traction	9 250	8
350	Steam	Friction	Traction	12 000	1 1
500	Steam	Geared	Traction	12 000	1 1
350	Gasoline	Friction	Traction	11 650	1 5
400	Steam	Geared	Non-traction	10 000	9
400	Gasoline	Geared	Non-traction	9 750	9
400	Steam	Geared	Traction	13 000	1 1
800	Steam	Geared	Non-traction	10 000	9
800	Gasoline	Geared	Non-traction	9 750	9
800	Steam	Geared	Traction	13 000	1 1
1 000	Steam	Geared	Non-traction	14 000	1 0
1 000	Steam	Geared	Traction	16 000	1 2

Setting up. Level the rear wheels transversely by blocking, if nece Take weight off front axle by lifting the body on jacks, and level transv For holes less than 100 ft, longitudinal leveling is unnecessary if the tools the front of machine. For deeper holes some attention should be paid to

he aid of engine. For holes over 300 ft
lank, about 14 by 18 ft, in front of drill.

lar to drilling with standard rig. Holes
s fitted with a special spudding device,
), consists of 2 pulleys mounted between
le passes from the cable reel first under
dding pulley; then over the crown pulley
ols are dropped about 60 times per min.
drilling as it keeps the gravel in suspension
g. Variation in speed has a considerable
noticeable in shallow than in deep holes.
starts on the temper-screw, although with
d to several hundred feet. Worn cable is
ing wears cable quickly, which therefore
part being cut out and the rope spliced as
o ft is an economical length for new cables.
h, and leaves about 60 ft of good cable on
l of a new rope. About 1 000 to 1 500 ft
The sequence of operations depends on the
cting requires that the hole be lined even
to interfere seriously with progress. Blast
rst few feet. Shallow water and oil wells
ortable rig works at a great disadvantage
annot be handled without a special derrick.

ft wide in the solid. A traction drill can make a
r hauling supplies the grades should not exceed
speed, 4 to 5 miles per hr can be made on good
on drill can go over any boulders or ruts that a
s much cheaper to use horses for moving. A 6-
good roads; a 10-horse team, an 18 000-lb drill.
25 mile and set up in 2.5 to 5 hr. In blast-hole
expedients. Because of the short string of tools
tened, and the center of gravity of the machine
th the derrick up. This will save 2 or 3 hr per
lerrick and the tools are dragged under the ma-
ailable, as at Copper Flat, Nev (Art 8), the drill
position and lifted off again. This requires only
inst about 10 men moving with the traction de-
the time is less than 1 hr, as compared with a

les, prospecting, and shallow water-well drilling.
an 4.5 or 4.25 in (the economical limit with cable
ted for the cable, thus reducing falling friction.
s cuttings to surface. With obvious exceptions,
able rig.

Churn Drilling for Prospecting

G. Weber (7), Keystone and Star drills were used;
granite and shattered silicious schist and granite
asing used in caving ground. Table 12 shows time
holes. Monthly average cost was $2 to $3 per ft.

int cylinder oil and one pint engine oil per shift
to 1 000 lb soft coal per 8-hr shift, or 0.5 to 0.75
than 1 000-ft capacity, are required, according to
ls (300-ft capacity) require about 4 gal gasoline

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Table 12. Time Distribution in Churn Drilling 13 Holes at Miami, Ari

Depth, ft	Total time, hr	Ft per 24 hr	Depth to water, ft	Rock	Percentage of total time for:				
					Drilling	Repairs	Delays	Moving	
103	136	18.2	.....	*A*	53.0	8.1	8.8	30.1	
252	303	19.9	.....	*B*	.....	.....	.....	.....	
252	684(*a*)	8.8	.....	*B*	.....	.....	.....	.....	
340	336	24.3	.....	*C*	.....	.....	.....	.....	
421	456	22.1	290	*C*	.....	.....	.....	.....	
427	420	24.5	.....	*B*	.....	.....	.....	.....	
468	580	19.4	320	*B*	63.6	1.4	19.5	8.1	
505	468	25.9	300	*C*	.....	.....	.....	.....	
527	730	17.3	15	*B*	66.7	0.7	16.4	7.1	
555	575	23.3	255	*A*	50.8	1.0	34.3	8.2	
608	636	22.9	465	*A*	75.6	5.8	6.9	6.3	
635	588	25.5	380	*C*	.....	.....	.....	.....	
645	529	29.3	.....	*A*	71.2	6.0	16.2	6.6	

a. 381 hours fishing. *A*. Granite. *B*. Granite and schist. *C*. Schist.

Table 13. Cost of Churn Drilling at Miami, Ariz (7)

	Cost
Labor: 2 drillers @ $6, 2 helpers @ $4.80, 1 sampler @ $4 and ½ foreman @ $6	$1.
Roads: Labor @ $2, 0.50; foreman @ $4, 0.05; explosives, 0.03; tools, 0.01	0.
Miscellaneous: Coal and oil, 0.27; water, 0.01; teaming, 0.10; assaying, office and incidentals, 0.16; int at 5% and deprec at 25% on $6 000 equipment, 0.20	0.
Total	$2.

per 10-hr shift. Water consumption for boiler, hole, and sampling, varies from to 3 000 gal per 24 hr, according to size of boiler and character of formation.

H. P. Bowen (9) gives following data on drilling at Miami copper mine: tot drilled, 44 500; 73 holes; range in depth, 336 to 941 ft; average depth, 605 ft; 36 drilled in silicified schist, 6 300 ft in granite or granite porphyry, 1 700 ft in soft co erate; work very slow in conglomerate after water was struck. Table 14 gives with different sizes of bit and casing. Table 15 gives speeds with a 10-in bit at di depths. Time distribution was as follows: Drilling and sampling, 69.80%; movi setting-up, 11.15%; casing, 6.25%; removing casing, 8.4%; fishing for tools, repairs, 2.0%; extraordinary delays, 0.30%.

Table 14. Speed of Churn Drilling, Miami Copper Co (9)

Bit diam, in	% of total hole drilled	Size of casing, in	Wt of casing per ft, lb	% of total casing	% of casing lost	Aver ft drilled, each hole	Aver vanc drillin
10.00	65	7.625	20	55	4.5	397	2.
7.625	24	6.25	13	31	4.5	157	2.
6.25	8	4.25	9	14	1.0	122	2.
4.25	3	.....	.....	.....	.....	105	1.

Table 15. Drilling with 10-in Bit at Different Depths, Miami Copper Co

Depth, ft	Speed, ft per hr	Depth, ft	Speed, ft, per hr
0–100	3.56	200–300	2.33
100–200	2.98	300–400	2.23

in speed up to about 125 ft; then a gradual crease is marked. In shallow holes less time g, and there is less friction on the cable than ing and removal of casing increase out of pro- med.

10 Churn Drill Holes, Miami Copper Co

d, ft	No shift	Total adv, ft	Daily adv, ft	Aver speed, ft per shift
	16	342.1	19.8	21.68
	17	364.4	22.3	21.72
	18	380.6	16.2	21.36
	19	391.7	11.1	20.74
	20	402.5	10.8	20.20
	21	414.2	11.7	19.72
	22	435.0	20.8	19.77
	23	451.3	16.3	19.63
	24	460.0	8.7	19.18
	25	469.0	9.0	18.65
	26	486.5	17.5	18.72
	27	501.8	15.3	18.58
	28	511.8	10.0	18.28
	29	528.8	17.0	18.25
	30	536.7	7.9	17.90

17 and 18 are based upon the records of a small the Chino Copper Co, N M. Table 18 shows r speed than the day shift; this was also the led in Table 26.

rilling at Chino Copper Mine

on Moving, %	Advance per drilling hr, ft	Number of drills	Costs				
			% of Total			Total, $	Per ft, $
			Labor	Supplies	Teaming		
8.1	1.84	2	....	....	.....	.....	....
9.4	1.16	3	76.8	18.0	5.2	2 742.35	2.05
7.1	1.22	2	73.3	23.8	2.9	2 770.69	2.09
0.9	1.30	2	78.0	17.6	4.4	2 561.38	2.04
6.0	1.37	2	75.9	19.1	5.0	2 594.71	1.85
4.8	1.52	2	....	....	.....	.....	....
7.1	0.95	2	73.4	20.4	6.2	2 484.82	2.50
1.9	1.48	2	70.2	24.5	5.3	2 529.74	2.20
9.5	1.28	3	63.4	33.0	3.6	4 254.93	2.76
7.6	1.58	3	65.5	25.5	9.0	2 805.76	2.45
2.0	1.58	2	64.2	31.2	4.6	2 721.58	1.75
5.3	1.51	2	52.2	42.6	5.2	3 535.52	2.64
3.2	1.42	2	....	....	.....	.....	....
5.6	1.78	3	....	....	.....	.....	....
11.7	1.62	3	....	....	.....	.....	....
6.8	2.13	1	....	....	.....	.....	....
6.9	2.63	2	....	....	.....	.....	....
* 8.5	1.48	Note. — Rock mostly shattered quartzite and porphyry					

ril 1912 to June 1913 inclusive, and Nov and Dec 1913.

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Table 18. Churn Drilling at Chino Copper Mine: Comparison of Day Night-shift Work

Year	Month	Day shift			Night shift		
		Drilled, ft	Working hr	Ft per hr	Drilled, ft	Working hr	Ft
1912	April	998	762	1.31	712	570	1
	June	696	756	0.92	630	618	1
	Aug	702	834	0.84	696	588	1
	Oct	528	714	0.74	468	594	0
	Dec	865	1 153	0.75	677	834	0
1913	Feb	716	642	1.12	841	624	1
	Apr	786	714	1.10	786	601	1
	June	546	516	1.06	522	366	1
	Nov	375	251	1.49	438	234	1
Totals 17 mo		11 748	12 326	.....	10 864	9 183	...
Averages		691	725	0.95	639	540	1

Note. — Rock mostly shattered quartzite and porphyry.

Table 19. Cost of Drilling, Savanna Copper Co, N M

Time drilling:			
From	Jan 31, 1910	Aug 1, 1910	Jan 31,
To	July 31, 1910	Oct 10, 1910	Oct 19,
Feet drilled	12 032	19 072	31 10
Number of holes	29	28	57
Average depth, ft	415	681	545
Cost of drilling per ft			
Runners	$0.329	$0.477	$0.42
Helpers	0.224	0.307	0 27
Assaying and sampling	0.151	0.239	0 20
Teaming	0.044	0.075	0 06
Foremen	0.085	0.093	0 09
Clerks	0.019	0.029	0 02
Road	0.114	0.057	0 08
Fuel and coke	0.229	0.341	0 29
Oils and grease	0 005	0.027	0 01
Road supplies	0.006	0.002	0 00
Cables and sandlines	0.044	0.141	0 10
Drill tools and repairs	0.054	0.133	0.10
Surface tools and repairs	0.004	0.019	0 01
Repairs to machine and boiler	0.016	0.041	0 03
Lighting	0.010	0.007	0.00
Barn supplies	0.023	0.042	0 03
Building material	0.016	0.015	0 01
Chemical and eng'g supplies	0.015	0.038	0 02
Stationery, printing	0.002	0.002	0.00
Telegrams	0.004	0.003	0.00
Transportation	0.028	0.022	0 02
General supplies	0.006	0.032	0 02
Depreciation	0.100	0.172	0 14
Casing	0.005	0.039	0.02
Pumping plant	0.151	0.134	0 14
Total	$1 684	$2 487	$2 17

Burro Mt District, N M. According to I. J. Stauber (10), rock was altered r zonite porphyry, cut by numerous slips and dikes. Mother Hubbard bits (Art 7) c cially long and thick, were used to prevent crooked holes and sticking. Most holes sta with a 10-in bit. Casing was not driven; if it would not follow, a smaller size was u

500 to 600 ft. Wages: drillers and helpers, plers, $110 per month; Mexican labor on uled 14 miles by wagon at $5.50 per ton. uch trouble, especially in freezing weather. time distribution for 20 months, 2 drills.

Drilling, Savanna Copper Co, N M

st time:	Hr	%
No water	782	2.9
Repairs (machine, tools)	1 531	5.7
No help	1 041	3.8
Fishing	1 314	4.9
Casing and pulling casing	1 472	5.4
Moving and erecting	2 904	10.7
Reaming and bad hole	2 482	9.2
Misc (rain, sickness, etc)	231	0.9
Total lost time	11 757	43.3

rmation is schist and porphyry; 9 Star used. String of tools: bit, 175 lb; stem, ope-socket, 75 lb; total, 1 450 lb. Wages: ; sampler, 50¢ per hr. Aver cost per drill ver cost per ft of 89 000 ft of hole, including as distributed as follows: general expense, d collecting technical data $0.57; grading al $2.90.

shows effect of caving ground and bad cites a record of 54 ft in 8 hr in rhyo-porphyry.

illing, Copper Flat, Nev (8)

Case I		Case II	
t	Cost per ft	Total cost	Cost per ft
	$0.300	$107.50	$0.672
	0.264	108.80	0.679
	0.251	72.10	0.451
	0.327	72.10	0.451
	0.007	3.18	0.019
	0.017	8.80	0.055
	0.528	100.73	0.629
	$1.694	$473.21	$2.956

a stands well. Good weather.
ich caved badly. About 33% of time was

at Silver Bell, Ariz (11)

Gal water per day (a)		2 200
Hr consumed moving	(7.6%)	10.58
" " drilling	(62.5%)	9 071
" " repairs	(8.2%)	1 147
" " fishing	(2.9%)	393
" " casing	(1.4%)	186
" idle	(14.7%)	2 041
Total hr		13 896

st in the hole and in sampling.

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Table 23. Cost of Churn Drilling at Silver Bell, Ariz (11)

	Aver cost per ft		Aver cost
Labor on drills	$1.25	Renewals	
Supplies	0.17	Roads	
Pipe line	0.09	Total	
Repairs	0.07	Aver cost cable per ft hole	
Fuel	0.35	" " coal " " "	

Placer prospecting. R. G. Smith furnished data for Tables 24 and 25 concerning prospecting at Natoma, Folsom district, Cal. Work was done with a No 3 Keystone

Table 24. Time Distribution in Placer Prospecting, Natoma, Cal

Group	1	2	3	4	
No of holes	25	12	13	33	
Total depth, ft	1 343	467	385	1 574	
	% of total time				
Drilling	57.6	58.4	58.4	63.9	
Pulling	19.6	11.5	14.4	14.8	
Moving	9.1	14.3	11.5	6.8	
Delay	13.7	15.8	15.7	14.5	
Total time, hrs.	970	426	243	1 105	
Maximum depth, ft	72	50	40	70	
Minimum " "	21	9	12	16	
Average " "	53.8	38.9	29.6	47.7	
Ft per drilling hr	2.4	1.8	2.7	2.2	
Ft per working hr	1.4	1.1	1.6	1.4	

Table 25. Cost of Placer Prospecting, Natoma, Cal

Property	1	2
No of holes	201	38
Total depth, ft	8 400	1 505
Number of shifts	700 (*a*)	98 (*c*)
Cost:		
Labor on drill	$9 970 (*d*)	$1 265 (*e*)
Superintendence	1 200	600
Supplies, material, additional labor	5 210 (*b*)	995 (*f*)
Total	16 380	2 860
Cost per ft	$1.96	$1.90

(*a*) Includes about 15 shifts. (*b*) Includes $200 drill. No rent or deprec on company drills. (*c*) shifts. (*d*) Driller at $4; panner at $3.50; fire $3; team and teamster at 9-hr shifts. (*e*) Driller panner at $3.50; fireman teamster at $2.50; all 9-hr (*f*) Includes team at les $2.50 per day.

Notes on drilling costs. The cost $1.90 to $1.96 at Natoma is somewhat lowe the aver for California. N. C. Stines gives 12 to 40 ft per shift as reasonable ress, cost per ft ranging from $1.40 to $3.50; $2.50 a fair average. D'Arcy Weat gives costs per ft for Oroville, Cal: 13 holes at $3.48, 7 holes at $2.50; for 5 holes, average depth 93 ft, aver cost, $3.85, winter rains, heavy roads, slow repai high fixed costs; for South Oroville, 50 holes, 35 ft average depth, $0.97 aver cost summer, all conditions favorable, 2 drills under one foreman and panner; for Or 9 holes, total 258 ft, 1 machine, 33 days single shift plus 4 for moving. He takes per ft as a fair aver for Cal work. J. E. Dick (12) gives $2.50 per ft as aver fo and about the same for Colo. J. P. Hutchins, writing in favor of hand dri prospecting in cold and tropical climates, gives daily cost of a steam drill in A in 1907, as $59 to $91 per shift. He sets 15 ft per shift as a good rate in gravel, using pipe; 50 ft or more in frozen ground without drivepipe, making a cost per foot of $4 to $6. T. A. Rickard cites 40 holes, 25 ft deep drilled in Alaska in August, for $1.01 per ft, including all costs, with wages as follows: driller $8, panner $6, fi

ystone traction machine, in Hot Springs
, 1908, sunk 9 054 ft of holes at aver cost
was 115 ft, shallowest 10 ft, aver depth,
rs and moving, 49.5 ft. The ground was
o $2 per ft in frozen ground and $3 to $5
or estimating.

sconsin, cost of drilling holes 100 to 300 ft
ft. A 1 300-ft hole cost $3 per ft, but
nce of crew and unsuitableness of the rig.
drill holes, through drift, sandstone, and
t $1 per ft.

paratively shallow, moves short, and

er shift, depending on kind of rock drilled.
ile in the porphyry at Copper Flat, Nev,
holes in a shift is the record for the pit.
drilling at the Chino copper mine, N M.
quarry. The discrepancy in favor of the
ished, though it has the advantage of be-

at Chino Copper Mine

er ce r l- ft	Number of drills	Costs			
		% of total		Total, $	Per ft, $
		Labor	Supplies		
0	5	83.7	16.3	2 937.30	0.372
3	5	81.6	18.4	2 212.60	0.300
7	3	81.7	18.3	1 129.82	0.321
6	4	81.2	18.8	1 908.40	0.325
7	4	82.4	17.6	1 791.20	0.305
2	5	.....	.....	.....	.....
3	7	.....	.....	.....	.....
0	7	.....	.....	.....	.....
1	6	.....	.....	.....	.....
5	6	.....	.....	.....	.....
4	6	.....	.....	.....	.....
1	6	.....	.....	.....	.....
9	6	.....	.....	.....	.....
5	8	.....	.....	.....	.....
5	8	.....	.....	.....	.....
0	8	.....	.....	.....	.....
0	8	81.9	18.1	4 092.97	0.407
1	8	82.0	18.0	5 340.28	0.365
6	8	77.6	22.4	6 120.01	0.377
7	8	77.1	22.9	5 671.22	0.376
9	8	68.0	32.0	6 402.31	0.461
4	8	76.2	23.8	6 560.98	0.407
9	8	70.1	29.9	6 490.53	0.507
3	9	.....	.....	.....	.....
3	9	.....	.....	.....	.....
4	Note. — Rock mostly shattered quartzite and porphyry				

April 1914 inclusive.

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Table 27. Blast-hole Drilling in Limestone Quarry of Dolese and Sh Co, Chicago, Ill

Drill	Power	Total ft	Total hr	Average ft per hr	D
A	Electric	1 593	274.5	5.8	
B	Steam	916	286.5	3.2	
C	Steam	817.5	278.5	2.9	
D	Steam	746	241.0	3.1	
E	Steam	947.5	235.5	3.9	
F	Steam	1 041	261.5	4.0	

Holes 5⅝ in; 40 to 45 ft deep in limestone; spaced 10 ft apart along the bench tric power cost $1.80 to $2 per day. 30-day run.

Table 28 is representative of the work of a No 3 non-traction Keystone drill blast holes in the quarry of the Staso Milling Co, Castleton, Vt.

Table 28. Blast-hole Drilling, Staso Milling Co, Castleton, Vt

Depth of hole, ft	Time drilling, hr	Time pumping, hr	Aver time each pumping, min	Advanced	
				Per drilling hr	Per in
42	11.25	1.08	12.33	3.7 ft	
41	11.67	0.92	12.59	3.5	
50	17.08	1.08	18.16	2 9	
48	13.92	1.25	15.17	3.4	

Rock, slate. Diameter of bit, 6 in.

At the New Haven Trap Rock Co's quarry, Keystone drills putting in 30-ft bl do not average more than 10 ft per shift. They make as high as 25 or 30 ft under exceptionally favorable conditions, but vertical cracks in the rock cause delays and lost holes that average is greatly reduced. One drill broke 4 stems in

ROTARY DRILLS

9. Rotary Hydraulic Drill

Construction and operation. With this drill, for boring in soft rock, r is loosened by a rotating bit on a line of drill pipe. Cuttings are ca surface by an upward current of water which is pumped down through p

Surface rig (Fig 40) consists of a derrick, 72, 88 or 106 ft high, similar to standard (Art 5); rotary-table *A*; draw-works *B*; engine *C*; and 2 pumps *D*. *A* is provided w for turning the drill stem, while allowing free vertical motion. It is driven by be and clutch on a horizontal shaft; which, in turn, is driven by chain from a sproc counter-shaft in derrick. Counter-shaft carries other sprockets, to take power fro and operate hoisting drum. Pumps are usually 10 by 5 by 12-in duplex, fitted for l heavy mud and cement grout at pressures up to several hundred lb. Typical borin naming parts from bottom upward, comprises: 8¼-in fishtail bit (Fig 41); couplin nect bit with drill pipe; a length of 6-in pipe; reducing coupling; 4-in drill pipe to drill stem; water swivel. When pipe is drawn from hole, joints are broken every 100 ft, according to height of derrick; at these points tool joints (Fig 42) are ins save wear on pipe. For drilling through thin layers of hard rock, Sharp & Hu (Fig 43) is used. For thick strata of hard rock, combination rig (Fig 44) is em

Speed of rotary drilling in soft formations has reached 1 000 ft per In such ground it is always faster than with standard rig. But, in ha

43. Sharp & Hughes Cone Bit

and Churn Rig

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the speed, even with the Sharp & Hughes bit, falls below that with st rig. Table 29 gives progress in wells drilled in Coalinga district, Cal. It a gradual decrease in speed with depth.

Table 29. Depth of Wells on Different Days

Days	I	II	III	IV	V	VI	VII	VII	IX
10	90	.....	.....	.....	.....	.....	.....	.....	.....
15	540	500	100	.....	50	400	100	170	150
20	800	780	400	70	200	800	300	300	400
25	1 050	1 150	900	350	500	1 100	600	650	670
30	1 300	1 380	1 300	750	800	1 340	800	880	950
35	1 440	1 550	1 500	1 100	1 050	1 500	1 000	1 150	1 070
40	1 550	1 700	1 680	1 450	1 280	1 700	1 280	1 400	1 180
45	1 680	1 850	1 900	1 650	1 420	1 810	1 550	1 600	1 330
50	1 780	2 010	2 050	1 780	1 590	1 830	1 750	1 690	1 530
54	.....	.....	.....	.....	.....	1 846	.....	.....	.....
55	1 880	.....	2 150	.....	1 700	.....	1 950	1 760	1 650
56	.....	.....	.....	1 902	.....	.....	.....	.....	.....
60	.....	.....	.....	.....	1 850	.....	.....	.....	.....
62	.....	2 123	.....	.....	.....	.....	2 151	1 872	.....
63	.....	.....	.....	.....	2 036	.....	.....	.....	.....
65	.....	.....	.....	.....	.....	.....	.....	1 711	

Rigging-up time as follows is included in above figures: I, 7 days. II, 11 day 12 days. IV, 14 days. V, 8 days. VI, 11 days. VII, 7 days. VIII, 10 day 8 days. X, 10 days.

Cost depends largely on speed. Crew of a large rig consists of 5 m shift as against 2 or 3 for standard rig, and labor costs are therefor Nevertheless, M. L. Requa shows cost in Coalinga field, Cal, to favor the rig, although some hard strata are encountered and the field is one in standard rig has been largely used (Table 30).

Table 30. Cost per Ft (Dollars) of Drilling by Rotary Method, Coalinga Fie

	I	II	III	IV	V	VI	VII	VIII	IX	X
Setting up	0.143	0.183	0.217	0.194	0.141	0.135	0.186	0.160	0.212	0.292
Tearing down	0.023	0.047	0.023	0.026	0.049	0.054	0.046	0.027	0.053	0.029
Stores to rotary	0.224	0.230	0.397	0.374	0.325	0.343	0.203	0.095	0.177	0.333
Extra materials on rotary	0.109	0.096	0.095	0.108	0.100	0.111	0.095	0.109	0.108	0.119
Extra labor on rotary	0.024	0.021	0.021	0.023	0.022	0.024	0.020	0.022	0.023	0.025
Fuel oil	0.377	0.221	0.193	0.237	0.295	0.150	0.256	0.194	0.305	0.235
Water	0.029	.....	0.047	0.029	.....	0.046	0.037	0.070	.....	.....
Drilling labor	1.316	0.869	1.000	1.027	1.367	1.156	1.192	1.395	1.376	1.725
Standard crew casing	0.197	0.210	0.531	0.378	0.184	0.336	0.250	0.283	0.254	0.087
Rig depreciation	0.298	0.266	0.260	0.294	0.377	0.302	0.259	0.331	0.295	0.326
Casing depreciation	0.274	0.270	0.220	0.276	0.274	0.247	0.218	0.219	0.108	0.218
Overhead charges	0.242	0.102	0.079	0.211	0.205	0.094	0.170	0.081	0.157	0.194
Deprec on tools, etc	0.149	0.134	0.131	0.149	0.139	0.153	0.130	0.151	0.150	0.164
Extra pump connection	0.013	0.012	0.016	0.014	0.011	0.014	0.010	0.013	0.012	0.014
Totals	3 418	2.658	3.231	3.340	3.389	3.165	3.084	3.122	3.242	3 766
Depth, ft	1 880	2 123	2 150	1 902	2 036	1 846	2 151	1 872	1 884	1 711
Days drilling	55	62	55	56	68	54	62	62	63	65
Aver ft per day	32.2	34.2	39.1	33.9	29.9	34.2	34.6	30.2	29.9	26.3

Note. — Costs common to both rotary and standard methods (as derrick, e boilers) not included.

the thin mud commonly circulated

adily drilled 700 ft of 15-in open hole, plain, before setting any casing. In He cites another hole 1 880 ft deep then cased with 6⅝-in pipe, the only . The fluid is made from any clayey apidly, allowable rate being a matter errick, overflows into a pump pit from of a trough in which a horizontal shaft e pit. Overflow from the well casing rflows into the storage pit. The mud nsistency.

to the objection that, unless the depths ney may be overlooked in drilling, as the well, except when under unusual heads. r oil and gas. When so employed the f oil tested by casing and bailing down. gh the casing, or the casing is removed. th the standard rig, for well drilling.

fluid with churn drilling. A special f string of casing. Through this the ugh the casing and returns to the sur- amed, and an extra large shoe is used ng. This gives good clearance for the e circulating fluid, and allows the cas- spended freely in the hole.

Lombardi (14) records a 15.5-in at 2 300 ft, and a 12.5-in string at this method, both of them being ee when landed.

ting is necessary to shut out water roductive oil sands. The water cas- ented into an impervious stratum

iller Method nenting

Fig 47. Perkins Method of Cementing

horizon. Fig 45, 46, and 47 show r-reamed 4 to 6 in larger than the

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water casing for 40 to 80 ft from the bottom, and circulation is estab
raising this casing a foot or so from the bottom, and pumping water or m
casing is free.

In the TUBING METHOD (Fig 45) casing is then raised 2 to 6 ft from bottom, an
of 3-in tubing with swedge nipple and cast-iron packer on lower end is lowered
about 2 ft of the casing shoe. An amount of cement estimated to fill the re
portion surrounding the casing is pumped into the tubing. When this is all in
a wooden plug is slipped in and forced down by the pump on top of and follo
cement. Pump pressure rises sharply when the plug reaches the swedge nipple, a
ing is stopped. The water casing is then set on the bottom, and the cement i
to set for 10 to 28 days. Cement must be placed rapidly and allowed to settle b
period of initial set has passed. Neat cement is commonly used, proportion o
to water being about 10 to 6. In BAILER METHOD (Fig 46) after circulation is est
water casing is raised about 10 ft from bottom, cement is mixed and run into a la
arranged to dump on reaching bottom. Usually 2 loads of a 40-ft bailer are put
the casing is capped and dropped, forcing the cement out and up outside the
PERKINS METHOD (Fig 47), cement is pumped down through the water casing b
plugs, the lower being made so that it will rest on bottom of hole without fallin
the casing. Under the pump pressure the cement is forced past this plug and u
the casing. When the rise in pressure indicates that upper plug has reached the
pumping is stopped and the casing lowered to bottom. The cement is then allow

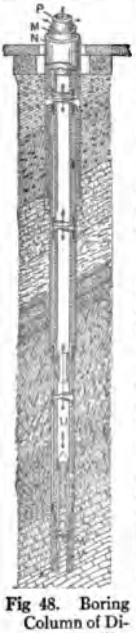

Fig 48. Boring Column of Diamond Drill

10. Diamond Drill

Construction. There are 2 parts: boring column and
plant. Boring column (Fig 48) consists of: bit X, set w
monds; core-lifter shell V, containing core lifter W; core b
and rods P. Surface plant (Fig 49) consists of: eng
derrick N, pump R, the boiler with steam rig, and the ne
connections and accessory tools. Following list is of a co
steam-driven outfit, with such repair parts as should be p
for work in remote regions. When operating near a supply
or for a small amount of work, many parts can be omitte

Outfit. Drill complete, with frame. Boiler and pump co
16 ft suction hose, with couplings and strainer. 75 ft pressu
with couplings for 3 lengths. Pipe, valves and fittings, for ste
water connections of boiler, drill, and pump. 1 derrick. 2
hoisting rope, with hooks. 250 blank bits. Carbons for 3 bits.
shells. 36 core lifters. 1 20-in and 2 10-ft core barrels. For dee
1 long core barrel in place of 10-ft barrel. Rods and couplings
times the capacity of the machine (include a few short lengths).
swivels, with couplings. 1 lifting bail and clevis. 1 water
1 hoisting swivel. Casing, with protectors, of length equal to c
of drill (include a few short lengths). 6 casing bits. 1 bushing.
casing. 1 reamer and guide and 2 extra reamer faces. 2 sets rec
taps. 1 derrick sheave. 1 bail and bolt, or shaft and pillow blo
sheave. Material for floor and shelter. Sludge and core boxes.
hole surveying outfit. Pipe clamps for drivepipe and casing.
diamond-setting tools, including: bench vise, breast drill, setting
setting hammer, pair 6-in dividers, inside and outside calipers, 1
rule, 12 chisels and punches, 10 twist drills $\frac{1}{16}$, $\frac{5}{32}$, $\frac{3}{16}$, $\frac{7}{32}$,
1 hack-saw frame and 12 blades. Safety clamp and extra set of j
same. Set solid wrenches and spanners for drill. 1 pressure g
drill. Sight-feed lubricator and valve for drill. Extra boiler tu
grate bars. Flue cleaner, poker, and ash hoe. Extra oil-cup gla
drill and pump. Supply of cylinder and lubricating oil and grease. 6-in single
6-in double block. 100 ft, 0.75-in Manila rope. 15 ft, $\frac{3}{8}$-in chain, with hook an
1 sledge. 1 machinist's hammer. 2 hand saws. 1 screwdriver. 1 tape. 1 brac
bits, 0.25 to 1 in. Extra valves, packing and lining for pump. 4 balls candle w

Tank or barrels for water. Lanterns or gaso-
6-in adjustable level. Rubber, hemp and
2 jackscrews. 1 one-gal oil can and 2 hand
ags. Supply of round and strap iron. Set
ter, taking up to 2-in pipe. Machine taps
75 in, and small bolts and nuts. Assorted
couplings, to twice the greatest estimated
drive shoes. 1 driving weight. 3 Stillson
es, 10 and 15-in. 1 15-lb crow bar. Axe,
galvanized iron pails. 1 firing battery and
sing. 1 chisel bit to fit drivepipe. 2 pair
waterproofed. Portable forge and small

izes of bits, core barrels and rods in
which they bore. (For Diamonds and

ing Fig 50. Double Core Barrel

nd *N*, of Sullivan Mach'y Co, are the
heir letters. SAW-TOOTH BIT is some-
SOLID BIT, making no core, has been
gh barren strata above ore horizon.
with diamonds to cover; rim diamonds
ssages are provided to admit water to
outside surface plain or cut with spiral
, 10, 15, 20, 30 and 40 ft in length.
sed in very soft and friable materials.

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Table 31. Dimensions of Diamond Bits and Fittings

Diameters		Bit		Core barrel		Rods		Style
Hole, in	Core, in	Outside diam, in	Inside diam, in	Outside diam, in	Inside diam, in	Outside diam, in	Inside diam, in	
1⅜	1⁵⁄₁₆	1¹ ¹⁄₃₂	3¹⁄₃₂	.....	.....	.....	.....	*B*
1¹ ⁹⁄₃₂	2⁹⁄₃₂	1⁷⁄₁₆	1⁵⁄₁₆	.....	.....	.....	.....	*B* Bullock
1½	⅞	1¹ ⁵⁄₃₂	1⁵⁄₁₆	1⁷⁄₁₆	1	.....	.....	*E* Minnesota St
1¹ ⁷⁄₃₂	1	1½	1¹⁄₃₂	.....	.....	.....	.....	.....
1⁹⁄₁₆	1⁵⁄₁₆	1¹ ⁷⁄₃₂	3¹⁄₃₂	.....	.....	1⁵⁄₁₆	.....	*E*
1⁹⁄₁₆	2⁹⁄₃₂	1¹ ⁷⁄₃₂	3¹⁄₃₂	1½	1	1⁵⁄₁₆	2⁷⁄₃₂	*E* Std
1⁹⁄₁₆	2⁷⁄₃₂	1¹ ⁷⁄₃₂	2⁹⁄₃₂	.....	.....	.....	.....	*E* Medium
1⁹⁄₁₆	2⁵⁄₃₂	1¹ ⁷⁄₃₂	2⁷⁄₃₂	.....	.....	.....	.....	*E* Extra heavy
1¾	1⁹⁄₁₆	1²³⁄₃₂	1⁷⁄₃₂	.....	.....	.....	.....	*C*
1¹³⁄₁₆	1¼	1²⁵⁄₃₂	1⁵⁄₃₂	.....	.....	1⁵⁄₈	.....	*A*
1¹³⁄₁₆	1³⁄₃₂	1²⁵⁄₃₂	1⁵⁄₃₂	1¾	1¼	1⁴⁵⁄₆₄	1⁵⁄₆₄	*A* Std
1¹³⁄₁₆	2¹⁄₃₂	1²⁵⁄₃₂	1½	.....	.....	.....	.....	*A* Extra heavy
2	1⁷⁄₁₆	1³¹⁄₃₂	1¹³⁄₃₂	.....	.....	.....	.....	*D*
2¹⁄₃₂	1⅜	2	1¹³⁄₃₂	.....	.....	.....	.....	.....
2¹⁄₁₆	1⅜	2¹⁄₃₂	1¹³⁄₃₂	.....	.....	1¹³⁄₁₆	.....	*B*
2¹⁄₁₆	1⅜	2¹⁄₃₂	1⁷⁄₁₆	2	1½	1²⁹⁄₃₂	1¹⁷⁄₃₂	*B* Std (8 or 10 thread)
2¹ ⁷⁄₃₂	1¾	2½	1²⁵⁄₃₂	.....	.....	.....	.....	.....
2⅝	2	2¹ ⁹⁄₃₂	2¹⁄₃₂	.....	.....	.....	.....	*E*
2¹³⁄₁₆	2	2²⁵⁄₃₂	2¹⁄₁₆	.....	.....	2⅜	.....	*N*
2¹³⁄₁₆	2	2²⁵⁄₃₂	2¹⁄₁₆	2¹ ¹⁄₁₆	2⅛	2⅜	2	*N*
3¹⁄₃₂	2¼	3	2⁹⁄₃₂	.....	.....	.....	.....	.....

(*a*) 1, Standard Diamond Drill Co; 2, E. J. Longyear Co; 3, H. R. Ameling D ing Co; 4, Sullivan Mach'y Co.

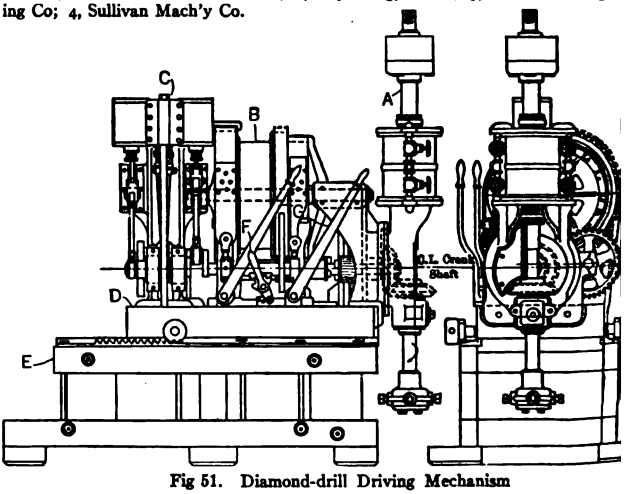

Fig 51. Diamond-drill Driving Mechanism

Inner tube remains stationary, protecting the core from wear and keepin wash water away from it. Cores 1.5 to 2-in diam are better than smaller At Poderosa, Chile, the *E* size was found to be very fragile.

f: feed mechanism *A*, hoisting drum
me *D*. To clear the hole when rods
k on foundation *E*; or else the feed
ide. This
o 6 000 ft.
uch as hand
allow holes
es to 350 ft.
e the rated
otor. But
on, usually
to greater
mall power
n A-frame,
edging into
ths to 2 500
ll mounted
y, gasoline,
wer drills.
any small
REW FEED

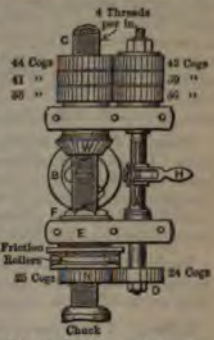

Fig 52. Differential-gear Screw Feed

w screw shaft *C*, and are gripped by the
ging gear *W*, which is keyed to *C* by
ooves. 2 or 3 gears are keyed to the hub
ard through a bearing on main frame.
numbers of teeth, for varying rate of feed.
on upper end of countershaft *A*. Feed
n *A*, but any one of them can be thrown
r *D* is keyed to *A*. Gear *N* is threaded
ub extends upward through a bearing in
vented from moving vertically by nut *F*.
by the gear ratios, the pitch of the thread
peed. Gear combinations are furnished
2 400 rev of *C*. With the lowest gear on
76 times for every 75 rev of *C*, and in
s fed down a distance equal to the pitch
y 75 rev. With a 0.25-in pitch, as illus-
ance per 300 rev. Upward thrust of the
on collar, is measured by a pressure-gage.

almost universally on large drills. In Fig
nder, *B* the piston, *C* a hollow piston rod,
E discharge-water outlet, 1 and 2 inlet
alves. By opening diagonally opposite
aised or lowered. This motion is trans-
ction head *S* to collar *I*, which is rigidly
rod *J*. *J* is rotated by bevel gear *K*,
nion *T* on crankshaft. The drill rods *P*,
pped by chuck *L*, being thus rotated and
-gage measures the thrust on the rods.
e also with 2 cylinders, the piston rods
the drive rod carried in the yoke. Usual
ngle-cylinder feed is 1 ft; with double-
n screw and hydraulic feeds are mounted
admits boring in any angular direction.

ulic feed. Considerable difference of
y given set up, screw feed gives a constant
e bit, thus apprising the runner of slight

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differences in hardness of rock, and of presence of thin seams and small crevices. T mation is often invaluable when the core is much broken and its record cons incomplete. But, in strata of frequently varying hardness, it is necessary to run soft strata at reduced speed, so that the bit may not be damaged on meeting a h tum. With hydraulic feed the pressure is constant, and rate of advance varies less automatically as hardness of the rock varies. Thus, danger to the bit in pass soft to hard rock is lessened. The runner of the hydraulic-feed drill can instan advantage of changes in the formation, varying the rate of advance, within t imposed by the rock, merely by turning a valve. Three different speeds are with screw feed, unless the operator stops to change gears.

Hoisting drum (*B*, Fig 51) in small power drills is driven directly by gearing fro shaft. In the larger machine illustrated, it is direct-driven for light loads; but holes, when the weight of rod may be several tons, a compound gear reduction is The gears are thrown in by lever *F* when hoisting, and thrown out when drilling Hoist should have the greatest speed possible without endangering the diamonds machines have 2 band brakes, operated by levers *G*; small machines, 1 brake.

Engine (*C*, Fig 51) is double-cylinder, vertical, balanced, with quartering Diamond drill engines must be compact, smooth-running, and without dead They are fitted to run by compressed air when necessary.

Boilers. Two types of portable boiler are used, vertical and horizontal tubul horizontal is best adapted to mounting on wheels and is more conveniently tra over good roads. For transport in rugged, wooded, or swampy country, the type is better.

Pump. A duplex, boiler feed pump is generally used to furnish water to r hydraulic cylinder. It must give a steady supply at constant pressure, and be handle gritty water. Pumps wit lined cylinders and brass pistons and fittings are good.

Table 32. Wire Hoisting Ropes for Diamond Drills

Drill capacity		Diam of rope, in	Min length, ft	Advisable min sheave diam, in
Depth, ft	Diam of hole, in			
600	1.75	0.5	75	16
1 000	2.06	0.625	75	21
1 500	2 06	0.75	100	22
2 000	2 0	0.75	125	22
4 000	2.0	0.875	125	37.5

Derrick varies in height a struction according to depth Up to 400 ft a 3-leg derrick Fig 49, giving a 20-ft clear sufficient. Legs should be 3 Southern pine, or its equivalen holes 400 to 1 000 ft in depth ilar derrick giving 30-ft cle with 6 by 6-in legs, should b A satisfactory 3-leg derrick built of 3 or 4-in pipe instead of timber. For deeper holes 4-post steel de 60 to 85 ft high, designed for quick erection and dismantling, are best. Th furnished by drill manufacturers (Table 33). A rod length of 40 ft is abo

Table 33. Weights and List Prices of Structural Steel Derricks
(Standard Diamond Drill Co)

Number	Height, ft	No of legs	Dist betw leg bases, ft	Will hoist in clear, ft	Safe working load, lb	Weight, lb	Pr
1	30	3	10	24	4 000	1 500	
2	38	3	12	32	5 000	1 900	
3	38	3	12	32	10 000	2 300	
4	52	3	16	45	15 000	4 100	
5	63	3	19	55	20 000	5 400	
6	52	4	16	45	18 000	5 500	
7	63	4	19	55	24 000	7 000	
8	77	4	22	67	40 000	20 000	

without excessive bending. For longer crane or crab in the derrick, so that es in Table 32 are plow-steel, 8-strand, s in Table 34 are figured according to

Complete Diamond Drill Outfits

ower	Approx price, $	Approx wt, lb
and	3 700	4 000
team	4 500	11 000
"	5 000	15 000
"	5 000 to 5 700	24 000
"	8 000 to 10 000	32 000
"	8 800	38 000
"	13 000	47 000
"	17 000	57 000
"	10 000 to 15 000	62 000
"	18 500 to 29 500	150 000

rilling Operations

surface set-up with a steam rig. Site source of water, and in a location favor-

should be graded, and sills laid level. The 2 lts of the drill. Outside sills take ends of the 20 ft long. Drill, pump and boiler are set up, derrick is centered over the drive rod. For et the derrick legs on the floor to counteract drilling or hoisting. A shelter of some kind fficient for small machines. In winter a sub- should be built to cover both boiler and drill. cted with bolts will save time. To erect a hanty, will take say 100 man-hr, if grading is nan-hr.

icult to set up, though for shallow holes no roken in 5 or 10-ft lengths, as space requires. which to handle longer rods. Compressed air

on a scow, anchored to bottom, or supported it stable setting, a platform is built on piles, w, securely guyed, and the drill set on a plat- ills are set up by blocking the front wheels, ng derrick over the drive rod. Two men can ntle in about half that time.

e by wash boring (Art 1). On reaching in a few inches, to make a tight joint, pre- ure return of drilling water to the surface.

n. In soft and medium clays and fine gravels, e bouldery gravel and in hardpan, aver speed peed of sinking standpipe will not exceed drill- it may be considerably less than drilling rate

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Drilling. Following is the normal sequence of operations:

Slide or swing feed mechanism away from hole. Set safety clamp in pos standpipe. Connect up bit, core shell (with core lifter), core barrel, and 1 leng and lower through the safety clamp until only a few inches project; then g clamps. Screw hoisting swivel into upper end of a section of rods, swing rods with the hoist and screw them into the joint projecting from the hole. L clamp and lower the string on the hoist brake until only a few inches project Tighten the safety clamp, and unscrew the hoisting swivel. Repeat these operat within a rod length of bottom. Bring feed mechanism back into place. Att swivel and lifting bail to the last length; lower it through the drive rod, and into place. Run the feed to highest position, and tighten the chuck. Loos clamp. Connect water swivel to pump and start the pump. Start drill and r ward until pressure shows that bit is against rock. While drilling, adjust feed sary. When measurement of rod indicates that core barrel is full, or laboring and engine suggest that it is blocked by broken pieces of core, stop the drill. K running until water issuing from hole shows that sludge has been well removed rods by reversing the operations of lowering. Extract core, and place it in co Change bit often enough to prevent diam of hole from falling below gage, and against loss of diamonds from wear of the metal surrounding them.

Casing or cement grout is used to support the walls of a diamond d when they cave so as to prevent progress or vitiate samples. CASING is flu tubing large enough to al bit to pass. Table 35 giv sizes of casing. If the hol tinued of same diam below ing ground, it is reamed ou the casing is introduced. mond drill holes are never s casing is a difficult operatio ing is usually twisted and down. Sometimes a "casi set with a few diamond c placed on lower end of the which is then operated as in drilling. CEMENT GROUT i and more used instead of casing. In badly caving ground the drill is driven as is safe, and the core and sludge saved. Then the grout is pumped do rods or lowered in a bailer. After it has set, the resulting "plug" is through, and a deeper section of hole is sampled and grouted. Quick cements are used, provided they are strong enough to support the hole. A 0.33% soda to ordinary cement shortens the time of setting to 24 to 36 hr.

Table 35. Flush-joint Casing

Size of bit	Casing			Wt, lb per ft
	Nominal inside diam, in	Actual diam		
		Inside, in	Outside, in	
E	1.625	1.69	2.12	3.00
A	2.0	1.93	2.37	5.02
B	2.5	2.31	2.87	7.67
N	3.0	2.89	3.50	10.25

Cementing is usually cheaper than casing, and allows the hole to be started at a diam without considering possible future support. In shattered ground the rods ma to be pulled frequently, on account of blocking of the core barrel. In soft ground if core is desired, frequent withdrawal is necessary or a double core barrel must be In fissured ground, part or all of the water will be lost. If there is standing water in fi strata, the sludge will deposit in crevices close to the drill, and re-enter the hole when current stops. Hence, water must be pumped through the rods during hoisting, an rods must be flushed down in lowering. Bran, sawdust, manure, or cement, are pu down the hole in an attempt to stop the crevices.

Accidents. Fishing. Commonest accident is LOSS OF BREAKAGE OF A DIAMOND (A Usual mode of recovery is to send down an old bit, the end of which has been filled wax. The diamond imbeds itself in this and is brought to surface. When the cannot be cleaned to the bottom, for using the waxed bit, a lost diamond may some be recovered by the bailer. The following device was used in such a case: A scree placed in top of core barrel and a screen clack valve set in an old bit, which was scr

rods were lowered through a stuffing box and the water current g the lost diamond was washed into the core barrel and held

may be due to fracture, stripping of threads, or unscrewing. t is usually a simple matter to screw them together again. reak, a recovery tap (threaded inside or outside according ered on a line of rods and manipulated until it engages the are similarly treated. JAMMING OF RODS may be due to cing with worn bit, or to too little clearance between rods ally gives some warning, but the required casing or grout- for progress. Jamming from a sudden cave can sometimes ing back the feed without stopping the engine, meantime ush will not often occur while pump is running. If the he rods all the way to the bottom, and kept running while danger zone, jamming will not occur. Pulling jammed essary, is destructive. The rods are seriously elongated, the threads may be partly or wholly stripped.

elow the core barrel, is a baffling and serious accident. a recovery tap, or some especially designed tool. Diffi- s from the fact that the bit may lie on its side; further- to turning, and therefore may turn with the tap. Bits he hole to a size allowing use of a bit large enough to bit. If the lost part cannot be recovered, the hole is it and continued (see Art 17, Deflecting boreholes). ten a serious factor in drilling.

ond drill consists of a runner and one or more e a good mechanic, since he must make most of e had long experience in diamond drilling. By ine, pump, and rods, a good runner can learn the ground being drilled. This knowledge is the testimony of core and sludge, and as furnish- ble in the hole. Many operators will not entrust ed machine to a runner who has not had 2 or 3 similar drill, nor give a man work on a deep hole ole experience with shallow ones. In general, runners for screw-feed machines. The helper is unner in handling the rods. Besides the above ually required, who may act as foreman for as for.

vs. A group of drills will require a general foreman. art or all the time, to haul fuel, attend to water supply, runner are usually slightly higher than miners' wages in usual helper's wages of the district. Bit setter is paid runner, minimum wages being $100 to $125 per month. ve lists of experienced runners and bit settlers available raveling and living expenses besides regular wages.

per day depends upon: size of drill and consequent ize of hole and rods; sp gr of the sludge; whether hether the hole leaks or is tight; and whether the of water in the hole is to cool the bit and raise the tter it will suffice for the former. If no water chan- ore barrel, the maximum size of particle that can pass qual to the clearance of the outside diamonds. With rmined by clearance between rods and sides of hole.

per sec is usually satisfactory. To maintain this requires hole, 100 gal in an *A* hole, 60 in a *B*, and 350 in an *N* hole,

using rods of the same letter (see Bits and Fittings, Art 10). With this rising the velocity in the restricted space around the core is considerably higher, re so great loss of core in soft rocks that quantity of water is usually reduced. I much water means rapid progress and low core recovery. Little water increase covery at the expense of progress, and too little water may result in a burned bi hole is tight and all the water pumped down returns to surface, the overflow fro box can be reused. Loss of water is then about 25%. In reusing, the water freed from oil, else the passages will clog.

Coal consumption ranges from 3.5 to 10 lb per working hr per rat hp, according to quality of coal and condition of boiler.

12. Time Distribution in Diamond Drilling

Time efficiency in drilling is the percentage of total working time i actual cutting of rock is done. It rarely exceeds 60%, and depends upo and character of overburden, time consumed in hoisting and lowering r time lost in delays. Time for handling rods depends on depth of hole, an between breaks. Number of lifts necessary depends on nature of grou of bit, and kind and length of core barrel. Uniform ground will so core the full length of the barrel. Cores in broken ground break up and barrel, or are lost. The double core barrel (Art 10) tends to overco difficulty. A solid bit need be raised only to change it.

Table 36. Time Distribution on 8 Shifts Diamond Drilling, Dome Mi Canada (15)

Actual drilling		Pulling rods			Dropping rods			Misc operations connected with handling rods		Misc delays	
					Hours						
Hr	%	Total hr	Aver per pull	%	Total	Aver per time	%	Hr	%	Hr	%
7.87	76.1	1.15	0.57	11.1	0.62	0.31	6.0	0.03	0.3	0.67	6.5
6.98	72.3	1.18	0.59	12.2	1.12	0.37	11.6	0.31	3.2	0.07	0.7
5.97	60.2	2.22	0.45	22.4	1.63	0.33	16.4	0.10	1.0	0.00	0.0
6.23	63.0	1.88	0.47	19.0	1.33	0.33	14.5	0.25	2.5	0.20	2.0
4.58	43.5	1.95	0.48	18.4	2.28	0.46	21.5	0.48	4.5	1.28	12.1
6.13	56.6	2.55	0.63	23.6	1.57	0.39	14.5	0.57	5.3	0.00	0.0
6.48	64.1	1.07	0.53	10.6	0.87	0.43	8.6	0.27	2.7	1.32	13.0
7.50	77.4	1.15	0.57	11.9	0.67	0.33	6.9	0.37	3.8	0.00	0.0
Aver.	64.1	....	0.55	16.2	....	0.39	12.5	....	2.9	....	4.3

Aver hr per time for pulling and lowering rods are found by finding the aver hr p per 100 ft, and multiplying by the aver depth. These figures agree so closely wi individual times for the different depths that the arithmetical averages of drilli waste time can be legitimately taken.

Table 36 shows that, for holes from 550 to 820 ft deep, actual cutting time av but 64% of drilling time, raising occupied 16.2%, lowering 12.5%, incidental delays Table 37 shows for holes 236 to 568 ft deep: 37.7% cutting time, 27.9% handlin and 34.4% delays, some of which should probably be classed as extraordinary. (M tearing down and setting-up are excluded.) Table 38 shows for shallower holes: cutting time, 22.1% raising rods and 14.1% lowering, incidental delays being neg The higher percentage of time here consumed in handling rods is due to the short (10 ft). Obviously, in using these figures, weight must be given to the progress actually cutting and to the nature of the ground, since both factors affect the perc of time consumed in handling rods. But, figures cited are fairly representative of d

les for Coal in Pennsylvania, 1908

otal hr	Depth	Progress per hr, after drill is set up, incl delays, ft	Progress per hr, excl delays, ft	Progress per hr actual boring, ft (c)
9.75	377.5	2.4	4.0	7.0
3.25	358	2.7	5.0	9.0
5.25	279.4	4.0	4.8	7.9
4.50	236.4	3.7	5.0	8.6
6.50	568.7	3.5	4.1	7.5
....	364	2.9	4.5	7.8

able core barrel, 2.5-in core, 6 to 19 ft sur- cores. a. Boiler responsible for most of . c. In shale 7.05, fire clay 7.10, limestone h of moves not given.

on Mariposa Estate, Cal

	3	4	5	Totals	Averages
	79	74.67	136.33	661.00	132.2
	6.5	2.75	3.5	22.25	4.45
	19.0	8.1	4.4	.....	6.1
5	6.89	8.78	18.03	67.07	13.41
	20.2	25.8	22.6	.....	18.3
6	3.68	3.78	4.07	42.55	8.51
	10.8	11.1	5.1	.....	11.6
	.....	.....	15.45	15.45	.....
	.....	.....	19.2	.....	4.2
5	.....	.....	.....	3.25	.....
	.....	.....	.....	.....	0.9
5	.....	.....	.....	23.25	.....
	.....	.....	.....	.....	6.4
9	16.93	18.69	38.95	192.18	38.44
	50.0	55.0	48.7	.....	52.5
	34	34	80	366	.....
	2.3	2.2	1.7	.....	1.8
	4.7 C	4.0 A	3.5 A	.....	3.4

slate. C, slate and serpentine.

9 shows the progressive increase with depth the effect of distance drilled per lift on total

as sunk vertically 3 728 ft through clay, "P" drill, 4 000 ft cap; 25-hp locomotive below 2 000 ft, 2-in bit, 1¾-in core. Core 20-ft to bottom. Crew; foreman, runner, raordinary delays on acct water supply and

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Table 39. Diamond Drilling in Transvaal
(Time Analysis and Progress on One Hole) (16)

Total depth, ft		Depth drilled, ft	No of lifts	Drilled per lift, ft	Aver time handling rods per lift, hr	Raising and lowering, % of total time	Actual cutting, % of total time	Total drilling time, hr	Rate per drilling hr, ft
From	To								
693	899	206	25	8.23	2.0	28.9	71.1	170.9	1.20
899	1 099	200	31	6.46	2.7	33.3	66.7	248.6	0.80
1 099	1 297	198	33	6.00	2.7	40.7	59.3	221.7	0.89
1 297	1 501	204	34	5.95	3.5	52.6	47.4	225.4	0.90
1 501	1 705	204	23	8.85	4.3	45.6	54.4	217.0	0.94
1 705	1 901	196	18	10.91	4.4	39.4	60.6	202.7	0.97
1 901	2 021	120	11	10.91	5.2	41.9	58.1	136.5	0.88

Hole drilled from bottom of a shaft 693 ft deep; rig on surface. **Derrick**, 50
Rods broken in 40-ft lengths. 20-ft core barrel; filled only 3 times. **Rock**, qua

erection of rig, 296 hr (5.8%); delays on acct casing, reaming, choking of hole, r
of lost diamonds, raising loose pieces of core, breakage of rods, repairs to ma
1 329 hr (26.1%); actual drilling, 3 463 hr (68.1%). Time setting 88 ft of 3-in
pipe, telescoped inside 5, 7 and 9-in lengths, 552 hr. Aver rate per hr actual cu
quartzite, 2.6 ft; in badly broken micaceous schist, 2.0 ft; in shale at bottom
2.0 ft. Aver rate combined raising and lowering rods, 15 min per 100 ft of rod
recovery in quartzites and shales, 92.75 to 96%. Table 40 gives times for combin
ing and lowering at various depths in a number of holes.

Table 40. Time for Handling Rods in Holes in Scattered District

Location	Length of sections, ft	Minutes per 100 ft			D
		Raising	Lowering	Raising and lowering	
Bisbee, Ariz	20	.....	.....	8.4	
Leipzig	.....	.....	.....	10.4	
Dornkloof	50	4.2 to 4.8	3.6 to 4.2	7.6 to 9	
Rand	.....	.....	.....	10.8	
New York Aqueduct	30	6	6	12.0	
" " "	30	10	10	20.0	
Lake Superior iron region	10	8.3	5	13.3	
" " " "	10	8.4	5.6	14.0	

Delays due to character of ground, and current repairs and renewals on t
are incidental to all drilling operations. Allowance of 10 to 20% of the tim
mated for drilling (including handling rods) will generally cover incidental d

Table 41 gives delay record on 19 holes from 78 to 2 172 ft deep on the Rand, in
incidental delays ranged from 0 to 34% of drilling time in rock, with an aver inci
delay of 14.3%. Table 42 is an analysis of the delay record on 3 holes at a Mic
copper mine.

There are other delays, due to breakage of tools in the hole; loss of stones fro
bit; dropping objects down the hole; breakage of the rig; lack of fuel, water, or
necessary tool. These cannot be foreseen and must rank as extraordinary. Many
from poor management and careless running. Time allowance to cover these item
matter of individual judgment.

in the Transvaal (16)

ng ce l	% total time: Boring in rock	Incidental delays (b)	Extraordinary delays (a)	Total time, hr
8	81.2	0	0	264
3	15.4	0	13.3	480
2	47.1	9.7	43.0	2 818
4	51.3	0.2	37.1	842
1	36.9	11.5	29.5	651
6	63.3	8.3	13.8	1 858
3	76.5	9.3	4.9	643
8	35.8	.....	46.4	336
3	0	0	68.7	192
3	52.2	3.7	39.8	1 952
0	52.2	3.0	38.8	1 196
	42.6	4.7	52.7	508
2	32.0	4.9	59.9	976
	48.1	16.9	35.0	260
	45.8	15.4	38.8	598
8	68.4	6.7	24.1	4 492
8	48.2	13.3	29.7	4 902
3	43.1	3.1	51.5	1 042
	75.0	7.5	17.5	2 465
..	....	.....	.....	26 475
7	54.2	7.8	31.3	.....
7	78.9	11.4	.....	.....

failure of water supply, want of tools for accident or repair. *b*. All accidents, rebons and broken tools.

d Drilling 3 Holes, at a Michigan (17)

.....	544	1 191	802
.....	1.31	1.36	1.12
.....	2.30	2.23	2.00
.....	58.7	60.6	56.2
ods (a).....	29.7	22.7	21.4
nnections...	3.9	4.6	5.0
.....	0.6	0.4	0.8
.....	1.9	0.7	5.0
.....	3.8	1.2	5.9
.....	.....	4.1	.....
.....	.....	0.9	.....
it.....	.....	.....	3.7
.....	0.7	2.5	0.4
.....	0.7	2.3	1.6
.....	11.6	16.7	22.4
.....	Vert	58° down	Vert

l. No overburden. *a*. Rods handled in 5% solution stood 30 min to etch.

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13. Diamonds, Bit-setting, and Loss of Carbons

Diamonds. Two varieties are used for drill bits: (*a*) Carbon, a n form of pure carbon, granular or compact in structure, without cleavage 3.1 to 3.3, black or grayish black in color. A fresh fracture has appear broken steel; some stones have a greenish or brownish tinge. Nearly al from Brazil. (*b*) Bort is imperfect, or flawed brilliant; occurring in rounded forms, with radiated or confused crystalline structure. Bot and carbon have a hardness of 10, but the crystalline structure of bort it more liable to breakage, and it is not suitable for hard rocks. Cubi roughly spherical carbons are best. Irregular, flat, and pointed forms a to set, and wear unduly on account of breakage of the projections. diamonds has fluctuated greatly in the last 35 years; general trend be ward (Table 43). Small, or poorly shaped stones can generally be pu

Table 43. Range of Prices of Diamonds, per Carat, 1879 to 191

Year	Carbons		Bort	Year	Carbons		A
	Average (*a*), $	Range (*b*), $	Average (*c*), $		Average (*a*), $	Range (*b*), $	
1879–80	5.00	.....	.....	1898	33.00	36 to 40	..
1881–2	6.00	.....	.....	1899	35.00	70	..
1883	15.00	.....	.....	1900–01	40.00	36 to 70	1
1884	8.00	.....	.....	1902	45.00	35 to 65	..
1885–6	6.00	.....	.....	1903	50.00	45 to 60	1
1887	12.00	.....	.....	1904	55.00	40 to 65	..
1888	10.00	.....	.....	1905	65.00	.....	..
1889–90	12.00	.....	.....	1906	75.00	60 to 85	..
1891	15.00	.....	.....	1907	85.00	40 to 100	10
1892	16.00	.....	.....	1908	85.00	75 to 95	1
1893	16.50	15	.....	1909	85.00	95	..
1894	15.00	17	.....	1910	85.00	40 to 65	8
1895	15.50	20	.....	1911–12	85.00	.....	..
1896	20.00	14 to 70	.....	1913	85.00	60 to 90	2
1897	31.00	35 to 36	.....	1914	85.00	.....	..

a. For special grade or extra choice stones the prices, particularly in latter y run $8.00 to $10.00 higher per carat. (Prices from Sullivan Mach'y Co.) *b*. corded in technical journals. Lowest are for carbons of inferior grade, poor small size. *c*. Price of bort is usually 20 to 25% of the price of first-class carbon

below market price, while the larger, well-shaped stones command a p Fragments of less than 0.25 carat are known as scrap; price in 191 $3 per carat, with carbons at $85 to $90 per carat. Selection carbons requires judgment and experience of an expert. Lacking judgment, an engineer should buy through a reliable dealer or diam manufacturer.

Bit setting requires skill and experience, and is an important op Diamonds must have such clearance that the metal of the bit will not contact with the rock, and be arranged to "cover" each other, so that of the area to be cut will fail to be touched by a diamond at each revol the bit. Number and size of stones used vary with size of bit.

The *E* and *A* bits (Art 10) are usually set with 8 stones totaling 12 to 16 cara 4 1-carat stones were used on an *E* bit at Miami, Ariz. Practice in the Lake

16 carats in an E bit. In a series of holes
stones totaling 18 to 30 carats were used.
n is of good quality, and reduced in size
m down to 0.25 carat, but that a fragment
dom buy stones larger than 4, nor smaller
od quality or shape. They have success-
, and seldom more than 6 stones in 2.5 and
r properly the face of the bit. E. J. Long-
ing 6 to 8 carats each, but have used 3 to
rbons of 3 to 6 carats, and use well-rounded

work they are the largest. Clearance
d rock usually 1/64 in. Soft rocks re-
overy and prevent plugging (sticking)
es are cut in the bit, and shaped for
y seated with a backing of Babbitt
is carefully tamped around the stone
Stones are set so that their cutting
of rotation of the bit. Spiral water
bit. To remove the stones follow the
hem out with the metal immediately
lissolved in acid. A skilled bit setter
s are of fairly regular shape) and from
elow 0.5 carat are difficult to set, and
the depreciation in value due to this
n item usually neglected in published

rock, cleavage of minerals composing
manner of setting the bit, size of bit,

dwell mine, the diamonds assumed a glass
and, in the soft sedimentaries forming the
e 1 000 ft without resetting. Broken and
, cause higher diamond consumption than
erosa, Chile, the carbon wear for 2 450 ft
broken chert and jasper in Lake Superior
times for 4 or 5 ft advance. In rocks in
r less than where cleavage is difficult or
est Australia, taking the carbon wear in
itic greenstone was 1.6, in granite 1.9, in
ined amphibolite 4.9, and in very dense

hole the greater will be the diamond
re on the part of the runner increases
eakage and loss than through wear.
ical for hard rock. Inferior carbons
oney in danger of total loss in the hole,
in soft rock perhaps the only advan-
k.

rra Burra and London mines, Ducktown,
rocks, so that the comparison is not a fair
borts in a hard quartz. Table 45 gives
0013 carat in Pennsylvania coal measures

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Table 44. Carbon and Bort Consumption at Burra Burra and London Ducktown, Tenn (18)

Feet drilled:						T
Vein (*a*).....	411	65	226	90	228	
Country rock(*b*)	1 026	481	572	77	612	
Total.....	1 437	546	798	167	840	
Carats consumed	25.859	7.203 9.063	14.703	6.656 8.469	5.031	7
Kind of stone....	Bort	Bort	Bort	Bort Carbon	Carbon	..
Cost per carat....	$10	$10 $15	$15	$15 $85	$85	..
Carats consumed per ft.....	0.01799	0.02037	0.01843	0.05071	0.005984	..

a. Hard, silicious rock. *b*. Schists.

Table 45. Carbon Consumption

Locality	Rock	Carats
Witwatersrand ..	Quartzite, part broken, part excellent drilling.	0.0692
Alaska.....	Diorite, quartz, greenstone, and slate.....	0.0071
Pennsylvania...	Shale and sandstone.....	0.0018 to
New York.....	Limestone, shale, and sandstone.....	0.012, 0.
"	" " " "	0.016, 0
"	Limestone and shale.....	0.0016
"	Shale and sandstone.....	0.0024
"	Limestone and quartzite.....	0.015
"	Quartzite.....	0.0115
California.....	Hard felsite.....	0.04 to
Witwatersrand ..	Hard uniform rock.....	0.02
Michigan.....	Jasper, iron slate, hematite, diorite, and schist	0.012 to
Ontario.....	Limestone and granite, bands of hornblende and quartz.....	0.0108
"	Exceptionally hard granite.....	0.08
Michigan.....	Jasper, hematite, mixed ore, and diorite schist	0.0182
"	Trap and conglomerate.....	0.0066 to
Mexico.....	Soft, even rock.....	0.002 to
Lake Superior iron region	Slates and quartz.....	0.0028
"	Slates.....	0.0028
"	Slates.....	0.006
"	Hard ore and quartz, badly broken.....	0.014
"	" " " " "	0.0054
"	Exceptionally hard broken jasper and quartz	0.0025
"	Black slates, hard ore, quartz, and jasper, badly broken.....	0.0032
"	Mostly soft slates.....	0.0019

14. Recovery of Core

Core recovery is highest in hard uniform rocks, in which it may reac
It is lowest in loose, soft, cleavable and broken rocks, where, unless th
run almost dry, and rods are raised every few inches, recovery may not
to more than 10%. Vibration of rods causes low recovery. For good
in hard rock the drill should be run with low speed and heavy pressure;
rock, the reverse. Recovery is invariably less in upper than in lower po
holes, due to decomposed character of rock near surface.

Porcupine increased core recovery. Drill ed down until the core was packed in the ome typical rocks on the Rand. Table 51 g progressive decrease with size of bit. E.

ond Drilling in the Transvaal (16)

	Percent of core recovered in			
nale	Sand-stone	Quartz-ite	Dolo-mite	Ig-neous
....	.....	.....	.....	87.7
....	.....	.....	.....	70.7
5	.....	53.4	.....	.....
....	.....	52.6	.....	.....
0(*a*)	.....	95.0	.....	87.0
3	.....	66.7	51.0	84.5
6	.....	81.2	69.2	85.1
....	66.0	.....	.....	97.0
....	.....	.....	.....	.....
6	81.0	91.6	.....	44.0
7(*b*)	71.0	97.3	.....	88 7
0	75.0	.....	.....	78.5
1	.....	68.5	.....	.....
8	.....	23.2	.....	.....
....	.....	59.3	.....	.....
4	.....	94.0	.....	.....
....	.....	.....	.....	84.7
....	.....	83.47	.....	.....
....	.....	91.2	.....	.....

Unusually good, not representative.

Core recov-ered, %
33.5
48.4
54.1
94.8
66.4
18.8
36.0
16.1
28.6
42.9
19.9
83.5
69.1
60.5
23.7
80.3

Size of hole is determined by the depth to be bored, and purpose of hole. To obtain a core in soft, friable minerals, as coal and salt, the inside diam of bit should not be less than 2 in. For hard rock or ore, a 1 3/16-in to 1 1/4-in core is large enough. If the stratum or ore-body to be sampled lies deeply, the hole must be begun larger than this for larger and stronger rods, and to give room for casing without excessive reaming, should casing be necessary.

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15. Speed and Cost of Diamond Drilling

Speed is affected by: kind of rock and surface covering; depth, direction, a ation of hole; quantity of water in hole; kind of drill, fittings, and accessor core requirements; labor, climate, and continuity of work; percentage of

Speed of actual cutting is faster in soft rock than in hard. Rate of advance in uniform rock than in alternating hard and soft strata; it is low in shattered an rocks. Hard uniform rocks that core well drill faster than soft rocks which gri the core barrel. Caving decreases speed. A shallow surface covering will increase speed in the hole. Tables 48 to 54 give rates of advance in differe Table 48 shows a fairly consistent decrease in speed and increase in delays, with in percentage of surface covering. Bouldery surface covering decreases spee hole (Table 52). Rate of advance in rock is generally greatest in the second a hundred ft (Table 53). Decomposed top rock decreases speed for depths less ft, and the longer time spent in handling rods decreases it below 300 ft.

Table 48. Diamond Drilling on the Deep Waterways Survey (23)

No of hole	% total depth bored		Total depth, ft	Kind of rock	% total time				Total hr	Ft p
	Earth	Rock			Sinking standpipe	Drilling rock	Delays	Moving		Sinking standpipe
1	67.9	32.1	49.1	*A*	14.3	25.7	42.8	17.2	35	6.7
2	21.3	78.7	114.0	*B*	26.7	40.0	25.3	8.0	75	1.2
3	0.9	99.1	200.9	*C*	2.3	69.2	23.1	5.4	130	0.6
4	10.2	89.8	58.6	*A*	8.6	45.7	34.3	11.4	35	2.0
5	19.8	80.2	58.0	*C*	20.0	42.8	20.0	17.2	35	1.6
6	0.7	99.3	194.3	*D*	4.2	73.2	18.3	4.3	120	0.3
7	13.7	86.2	99.9	*D*	16.3	45.0	32.5	6.2	80	1.1
8	7.5	92.5	180.3	*D*	10.0	60.0	13.3	16.7	60	2.2
9	18.4	81.6	67.8	*D*	16.7	36.6	13.3	33.4	30	2.5
10	46.7	53.3	49.3	*A*	6.7	16.0	6.7	70.6	75	4.6
11	18.4	81.6	68.7	*A*	13.3	51.0	24.4	11.1	45	2.1
12	4.4	95.6	219.0	*C*	7.2	58.5	25.7	8.6	140	1.0
13	44.5	55.5	105.0	*E*	35.5	28.7	28.7	6.7	90	1.5
14	6.9	93.1	132.4	*D*	10.0	60.0	20.0	10.0	40	0.9
15	12.2	87.8	105.1	*F*	5.3	44.2	23.1	27.4	95	2.6
16	77.9	22.1	99.3	*F*	57.2	7.8	24.3	10.7	140	1.0
17	39.6	60.4	31.3	*F*	20.0	32.0	28.0	20.0	25	2.5
18	21.3	78.7	86.6	*F*	16.0	46.0	26.0	12.0	50	2.3
19	45.5	54.5	52.8	*D*	15.0	35.0	20.0	30.0	20	8.0
20	49.3	50.7	75.0	*G*	30.0	15.0	29.0	26.0	100	1.2
21	83.5	16.5	71.5	*A*	72.5	6.3	11.2	10.0	80	1.0
22	49.3	50.7	71.4	*H*	3.8	27.5	12.5	56.2	80	11.7
23	8.9	91.1	85.1	*H*	5.7	57.1	8.6	28.6	70	1.9
24	0.0	100.0	108.0	*H*	0.0	58.1	19.1	22.8	110	.....
25	64.0	36.0	78.0	*H*	6.3	22.4	27.5	43.8	80	9.6
								(*c*)		(*b*)
Aver	22.4	77.6	98.4	....	17.6	40.9	22.1	19.4	....	1.7

A. Limestone. *B*. Lime conglomerate. *C*. Limestone, shale, and sandsto Shale. *E*. Shale and sandstone. *F*. Sandstone. *G*. Limestone and slate. *H*. Q

a. Aver speed per hr in limestone, 2.5 ft; in sandstone, 2.4 ft; in shale, 3.6 ft; sedimentaries, 2.5 ft; in quartzite, 1.7 ft. *b*. Low aver due to charging setting u sinking standpipe. *c*. 33 moves, total 110 miles; 75 miles over bad roads wit and 5 moves covering 500 miles, by rail. *d*. Sullivan "S" drill, mounted on whee portable boiler.

Dec...	3	189	127	316	144	46	192	1.0	1.2	33.0	"	(g) 40.5	2.5	1.05	15.5	608.09	1.48	Limestone & porph
Jan....	6	246	165	411	159	57	216	1.9	2.6	43.5	"	(g) 40.5	2.5	1.05	15.5	608.09	1.48	Limestone & porph
Feb...	2	0	378	378	137	(e) 79	216	1.7	2.7	51.1	Elec....	14.3	1.1	1.78	33.5	479.44	1.27	Ore and limestone
Mar...	5	200	340	540	180	28	208	2.6	3.0	46.0	Elec....	9.2	4.5	2.58	40.3	577.34	1.07	" " "
Apr...	1	278	186	464	181	27	208	2.2	2.5	44.0	"	8.1	1.2	3.28	46.7	631.63	1.36	" " "
May...	3	67	433	500	163	53	216	2.3	3.1	38.9	"	6.9	0.7	5.06	53.5	852.90	1.70	Hard garnetite
June...	6	189	288	477	187	39	226	2.1	2.5	36.3	"	(h) 16.0	4.4	5.25	43.3	1092.97	2.29	Very hard garnetite
July...	6	96	304	400	203	23	226	1.8	2.0	29.1	"	(h) 13.2	11.9	7.05	45.8	1383.87	3.46	Same and diorite
Aug...	8	497	0	497	213	(f) 129	342	1.4	2.3	38.0	"	(h) 17.3	8.1	4.69	36.6	1154.83	2.32	" " "
Total .	78	2 706	3 500	6 206	2 499	852	3351	....	....	....	.....	.....	....	..	....	11 646.35	..	
Aver..	....	....	.....	79.6	....	....	....	....	....	38.7	.....	13.7	5.8	..	41.8	.....	..	

* Diorites, compact garnetites, & hard, silicious eruptives & limestones, classified by drillers as hard rock. All ores, greenstone, classified as medium. *a*. 8-hr shift underground. 9-hr shift on surface. Speed in flat holes about 15% greater than in steep. *b*. Contract work tried May 1906, but found unsatisfactory. No work June & July either year. *c*. Crew normally runner & helper, with additional labor on moving, water supply, teaming, etc. Drill runner at $175 per month set & was responsible for diamonds. Helpers, $3.50. *d*. Electric power (air from elec-driven compressors) is estimated on assumption that a diamond drill requires same quantity of air as a 3.25-in piston drill, or about 20 hp. *e*. Incl 15-mile move to another mine. *f*. Caving ground caused considerable lost time. *g*. Steam direct from 24-hp boiler; required engineer & team to haul wood. *h*. Incr in power cost due to partial closing of mine; hence, a greater propor of power-house expense was borne by the drills. *i*. Incl tools & miscel supplies. *j*. 1 9/16-in hole; 1 3/16-in core. Sullivan Beauty drill.

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Table 50. Diamond Drilling in the Transvaal (16)

Depth, ft			Ft advance per hr		Ft drilled per lift, rock (a)	Character of rock and % of			
Surface	Rock	Total	Surface soil (f)	Rock (d)		Shale	Sandstone	Quartzite	Dolomite
152	456	608	3.04	4.69 (b)	5.36	0	0	0	0
200	100	300	(e) 0.58	1.35	.....	0	0	0	0
7	348	355	0.87	0.26	2.80		Shale and quartzite		
95	255	350	0.99	0.59	.....	0	0	6.8	0
190	140	330	1.32	0.58	7.37	43.3	0	9.9	0
203	625	828	0.75	0.53	4.05	5.8	0	17.8	69.3
16	415	431	0.27	0.84	3.70	32.3	0	1.9	59.3
78	82	160	1.30	0.68	.....	0	17.0	0	0
78	0	78	1.30	.....	.....	0	0	0	0
73	569	642	0.87	0.56	.....	16.5	3.9	40.4	0
79	517	596	1.08	0.83	9.8	11.5	3.2	50.9	0
0	289	289	.....	1.34	.....	27.3	0	24.5	0
18	438	456	0.58	1.40	.....	66.1	0	33.9	0
0	180	180	.....	1.44	6.0	50.0	0	50.0	0
0	289	289	.....	1.05	5.16	0	0	100.0	0
33	2 139	2 172	0.92	0.70	8.87	24.8	0	75.2	0
20	1 140	1 160	0.05	0.48 (c)	4.87	0	0	0	0
28	204	232	1.17	0.45	3.22	0	0	100.0	0
0	1 328	1 328	.....	0.47	7.59	0	0	100.0	0

a. All by contract except last hole. *b*. Diabase, rate exceptionally high. *c*. ground badly broken, rate exceptionally low. *d*. Includes raising and lowering *e*. Boulders of chert imbedded in soft material. Difficulty in keeping hole Cased with 3-in pipe. Includes all material down to solid rock.

Table 51. Diamond Drilling under Hudson River (21)

Hole No.	1	2	3
Depth, ft	1 834	2 051.6	1 651.4
Working time, hr	4 008	4 632	2 040
Actual drilling time, hr (incl handling rods)	2 648	3 432	1 712
Time efficiency, %	63.6	74.0	84.0
Ft per actual drilling hr	0.69	0.60	0 97
Ft per working hr	0.46	0.44	0.81
Ft of 4-in hole	7.2	8.0	8.5
% of 3-in core recovered from 4-in hole	47.0	75.0	78 0
Ft of 2.75-in hole	633.8	711.3	715.5
% of 2-in core recovered from 2.75-in hole	79	91	84
Ft of 2.06-in hole	593.0	754.5	927 4
% of 1.375-in core recovered from 2.06-in hole	41	76	48
Ft of 1.43-in hole	600.0	572.8	.....
% of 0.81-in core recovered from 1.43-in hole	27	22	.....
Total core recovery, %	55	66	64
Dip of hole at top	43°	38°	22° 53′
Dip of hole at bottom	38° 55′	44° 47′	23° 35′
Casing used, 2.5-in, ft	735	0	724
Casing used, 2-in, ft	499	0	0

Sullivan *B* drill, capacity 3 000 ft; 8 diamonds to a bit; 0.0156-in clearanc of rod immediately following core barrel was same size as bit. Rock, clos gneiss.

14 Holes on the Croton Aqueduct

Hard gneiss		% total time			Total ft	Total days
Pt	Ft per day	Drilling	Moving (*b*)	Accidents		
...	...	100	....	....	191	5
...	...	100	....	....	195	7
01	14.4	75	6.25	18.7	174	16
...	...	75	12.5	12.5	160	8
77	12.8	75	16.7	8.3	137	12
36	12.0	53.2	13.3	33.3	135	15
10	10.0	36.3	4.5	59.0	99	22(*a*)
...	...	100.0	....	....	154	19
1	1	48.4	9.7	42.0	170	31
6	6	57.9	....	42.1	143	19
...	...	88.9	11.1	....	172	9
52	10.4	72.2	5.6	22.2	149	18
53	13.2	71.4	14.3	14.3	139	14
1	11	44.5	11.1	44.4	66	9
7	12.0	66.1	7.4	26.5	2 084	204
6.7	...	....	....	....	....	....

of moves not given. *c*. Drilled in 1 day in

t Isle Royale Mine, Mich (22)

hs in ft (*b*)			Total depth in rock, ft	Total hr drilling in rock	Aver adv per hr, ft
00 to 00	800 to 900	900 to 1 000			
..	....	....	700	694	1.01
..	....	....	699 (*c*)	768	0.91
..	....	....	619	715	0.87
..	....	....	605	660	0.92
..	....	....	415	445	0.93
..	....	....	557 (*c*)	432	1.29
..	....	....	440 (*c*)	344	1.28
..	....	....	505	624	0.81
..	....	....	470	576	0.82
..	....	....	401	384	1.04
..	....	....	550	640	0.86
.8	0.7	0.4 (*a*)	1 038	1 320	0.79
..	....	....	504	368	1.37
..	....	....	596	510	1.17
.8	0.7	0.4	580	605	0.96

Includes all such delays as handling rods, re-
through some hard but uniform and thick trap
n hardness decreased it. Rock, typical Lake
d amygdaloids, with trap and felsite.

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Table 54. Diamond Drilling in West Africa (19)

Hole No (*c*)	Diam, in	Material, ft			Kind of rock (*e*)	% total time			Total hr (*d*)	Pro pe
		Soil	Rock	Total ft		Erecting and dismantling	Boring, incl handling rods	Delays		Boring, ft
1	6.0	25	145	170	S	25.6	61.6	12.8	390	0.71
2	3.0	36	60	96	S	57.7	42.3	....	130	1.75
3	3.0	4	611	615	(*a*)	13.7	65.2	21.1	1 170	0.79
4	2.5	....	85	85	SS	55.7	30.0	14.3	150	1.89
5	2.5	9	67	76	SS	47.1	41.2	11.7	170	2.54
6	2.5	9	138	147	SS	58.3	36.7	5.0	120	3.47
7	2.5	15	186	201	SS	33.3	38.6	28.1	142	3.65
8	6.0	13	141	154	SS	16.1	25.5	58.4	685	0.58
9	2.0	2	76	78	SS	9.5	85.7	4.8	105	0.87
10	2.0	10	70	80	SS	10.0	90.0	....	100	0.89
11	2.0	5	23	28	S	14.3	85.7	....	35	0.95
12	2.0	12	149	161	SS	6.7	64.4	28.9	225	1.11
13	2.0	0	341	341	SS	.....	100.0	....	350	0.97
14	2.0	0	369	369	(*b*)	6.6	64.5	28.9	2 900	0.20
15	3.0	6	75	81	SS	19.0	66.7	14.3	105	1.16
16	3.0	1	115	116	S	6.1	93.9	....	165	0.78

Aver rates per hr: steam drill in sandstone, 1.07 ft; steam drill in shale, 0.59 drill in sandstone, 0.80 ft; hand drill in shale, 0.77 ft. *a*. 305 ft shale, 146 quartzite, 28 ft sandstone, 132 ft shaly sandstone. *b*. 154 ft sandstone and sha sandstone, 190 ft very hard quartzite, 21 ft shale and coal. *c*. Holes 1 to 8 steam holes 9 to 16 hand-drilled. Steam crew: 1 white and 5 or 6 natives. Hand-dri 1 white and 8 natives. (*d*) Exclusive of moving time. (*e*) S, shale; SS, sands

Between the horizontal and vertical the inclination of the hole has little effect o In the same formation, holes pointing upward drill slower than those pointing do because of the awkwardness of arrangements at mouth of hole. Occurrence water in such a hole adds to inconvenience in drilling, and tends to retard prog

Whether the drill is working underground or on surface makes no appreciab ence in speed, once the drill is set up. Moving is usually more difficult undergro the moves are likely to be shorter and several holes are often drilled from on Holes drilled from scows into subaqueous strata always advance more slowly than similar formations where the drill can be set up firmly. Rate of advance per d increases with size of drill. But, rate per working hr may be greater with a ligh heavy drill, when holes are shallow and moves frequent (Table 54).

In rock which cores well, the longer the core barrel used the faster the adva soft friable rocks, if high core recovery is essential, use of double core barrel will speed. Taking cores 2 in and smaller, size of core has no appreciable effect o provided drill is not working beyond its rated capacity. Larger holes will dr slowly. Holes drilled with a solid bit advance faster than those with an ann unless the hole is of large diam. Best quality of carbons, well set, gives fastest p

An experienced and skilled runner is essential to rapid progress. Character c labor also has considerable influence on the speed. Holes drilled by contract show higher speeds than those on company account, unless great difficulties and contractor can not finance added expense. Bonus system is sometimes ful. The danger in contract and bonus systems is that unless a minimum perce core recovery is stipulated increased speed may be gained at expense of core r Unfavorable climate, cold or tropical, retards drilling. Drilling 24 hr per day gives a greater speed per working hr than one-shift work. In the latter, time is in starting up, and usually also at end of shift, due to fact that it is unsafe to le drill with the rods in the ground.

r working hr; prices of supplies, fuel, power consumption; and accessibility o.) Costs given in Tables 55 to 60 n all cases cost of diamonds has been 90 per carat for carbons and $25 for tion of diamonds due to reduction in been recalculated to a standard, for comprise from 70 to 100% of total p holes and large outfits, the larger

lling for Coal, Pennsylvania

	Totals	% of total
Coal, 1 050 bush.	67	5.0
Oil and waste..	11	0.8
	466	.....
Repairs.....	24	1.8
Deprec, 20%.....	100	7.5
Supt'ce, ½ man...	200	15.0
Total.....	$1 336	100.0
Cost per ft, 1 820 ft	73¢	

on Deep Waterways Survey (24)

Route 3		Route 4		Av % for all routes
Cost	%	Cost	%	
$ 600	30.6	$ 376	23.0	.....
200	10.2	465	28.4	39.2
530	....	351	....	.....
273	....	191	....	.....
802	40.8	542	33.1	40.6
250	12.7	188	11.4	14.6
53	2.7	25	1.5	2.2
34	1.7	25	1.5	1.9
28	1.3	19	1.1	1.5
$1 967	100.0	$1 640	100.0	100.0
.....	.....	.....	....	.....
5	....	7	....	.....
75	....	77	....	.....
30	....	68	....	.....
$2 077	....	$1 792	....	.....

n; route 3, 22 340 lb at $3.055 per ton; l wood at $3.25. *b*. Carbon cost, $36.50 foreman, $100 per mo; teamster, team, no; 2 laborers at $55 per mo; watchman, ter, say $100 per mo. *d*. Supt, $125 per from drill mfrs, at $300 per mo, plus list market price per carat for carbon wear

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Table 57. Costs of Diamond Drilling, Croton Aqueduct

	Total cost, $	% of total cost	Total ft drilled, 2 084	% of total drille
Labor: 1 foreman @ $125, 1 ass't foreman @ $70, 4 laborers @ $65 per mo, teams and moving	3 865	50.0	Clay and gravel	16.8
Supplies	885	11.5	Gravel and boulders	27.4
Carbons	2 200	28.4	Decomposed gneiss	39.1
Repairs	291	3.8	Hard gneiss (charged with total carbon cost)	16.7
Int on plant, $6 000 @ 12% for 8.1 mo	486	6.3		
	7 727	100.0		

Table 58. Comparative Costs of Hand and Power Diamond Drilling in Nat

	% total cost	
	Power	Hand
Labor:		
Wages and rations	62.3	55.9
Moving	12.8	7.4
Supplies:		
Fuel and oil	5.6	.....
Diamonds (*a*)	17.5	35.9
Repairs including material	1.8	0.8
Aver cost per ft excluding moving.	$1.56	$2.01
Aver cost per ft including moving.	$1.79	$2.17

a. Diamonds cos carat, recalculated $90 per carat. drill crew: $4.50 and keep. Steam-d 1 white and 5 or 6 Hand-drill crew; 1 8 natives. Rock, s and shale. Averag power-driven holes hand-driven 124 ft.

Table 59. Diamond Drilling on the Lake Superior Iron Range

No of holes	5	1	3	5	1	1	2	3	2 (*b*)	2 (*c*)	6 (*d*)	2 (
Total depth, ft	1 066	1 293	478	780	216	174	267	410	634	360	1 350	6
Aver depth, ft	213	1 293	159	156	216	174	133	137	317	180	225	3
Rock	*A*	*B*	*C*	*C*	*D*	*B*	*B*	*E*	*F*	*G*	*H*	*I*
Total cost per ft, $	3.16	6.31	8.82	7.15	6.15	12.34	7.38	7.11	6.95	3.43	3.78	5.
	% total cost per ft											
Labor:												
Wages, camp acc't	34.1	25.3	37.1	23.6	14.2	18.1	19.2	31.9	25.6	36.0	47.5	29
Fuel	3.0	4.3	2.7	0.3	....	1.8	1.2	3.0	4.9	12.3	8.7	2
Bits & rep on drill	4.4	1.7	3.3	2.4	2.2	1.2	1.4	2.6	2.4	1.2	2.3	2
Supplies and rep on mach'y	1.1	1.0	0.4	1.0	0.4	0.3	0.1	0.5	1.4	0.6	2.4	1
Carbons (*a*)	49.9	62.6	49.4	72.1	82.2	77.1	76.2	57.7	63.3	39.8	33.2	62
Superintendence	7.5	5.1	7.1	0.6	1.0	1.5	1.9	4.3	2.4	10.1	5.9	2

A. Sandstone and marble. *B*. Slate and jasper. *C*. Hard jasper. *D*. In *E*. Jasper. *F*. Iron slate, diorite and quartz porphyry. *G*. Diorite, schist a *H*. Hard broken jasper. *I*. Hornblende schist, jasper and quartz. *J*. Quart

a. Probably cost about $15 per carat (recalculated at $90 per carat). *b*. Hol from bottom of a 70-ft pit. Aver advance, 5 ft per shift. *c*. Aver advance, 6 ft *d*. Fan-shaped series of holes, from the bottom of a 95-ft pit. Aver advance, shift. *e*. Aver advance, 14 ft per shift. *f*. Fan-shaped series of holes from bo pit. Aver advance, 8 ft per shift. All 1-in and 1.25-in cores. Little differen between surface and underground drilling. Crew: Runner, $2 to $2.50 per 1 labor, $1.75.

Drilling at Tonopah

Month				
Nov	Dec	Jan	Feb	March
114	74	109	190	129
175	249	358	548	677
192	216	200	192	208
0.59	0.34	0.54	0.99	0.62
36.5	22.5	55	78.5	47.0
32.0	30.4	50.5	41.3	36.4
9.57	10.14	9.37	7.49	7.53
7.39	11.48	7.51	3.64	8.11
ribution of monthly cost, % of total				
41.9	44.2	41.1	49.5	35.5
15.4	17.0	13.2	15.9	15.3
0.9	0.4	.....	.....	0.5
2.3	1.4	1.1	1.9	.....
22.7	13.8	27.4	24.5	44.2
12.2	10.9	3.9	1.6	2.3
0.4	0.1	7.6	0.8	2.2
4.2	12.2	5.7	5.8	.....

Top mine. Raise put up to make it possible
" drill used, capacity 1 500 ft. Compressed
ore barrel. *b*. 3.5-carat carbons, 6 to a bit.
emented to 225 ft, when caving became so bad
h 2-in pipe. Beyond this point there was no
ots. *d*. Crew: runner, $6; helper, $4 per 8-hr
xpense of $176.18 before starting drilling.

on are usually covered by a charge of

lerosa, Chile, are typical of work in remote
labor is hard to get and inefficient; native
drilled was broken, caving, and sometimes
necessary in every hole. Vein material was
and, electric, and compressed air-power were
illed, 2 444 ft. All holes were flat. Cheap-
tric drill. Highest cost, $13.65 per ft, with
th electric drill, $9.40 with hand drill, $11.40
r actual cutting, 1.14 ft per hr (25).

or water. In soft, caving ground the
d hole seriously increases caving.

sfully substituted for water in dry formations.
ngs to the surface. The discharged cuttings
ith water. Using an *E* bit, about 50 cu ft of
ed. Cost of drilling when using compressed
$80 per carat. This cost was slightly below
a somewhat higher carbon consumption (26).

t Boring

shot as the abrasive. The description
ies to the shot drill, with exception of:
re, method of collecting sludge, and feed

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mechanism. The bit (Fig 54) is a steel cylinder, with a diagonal slot in lower edge. It is screwed to the core-barrel *A* (Fig 55). Chilled shot are fed through the rods, and pass between core and inner wall of bit to the bottom of the hole where they cut by a milling action.

Fig 54. Bit for Shot Boring

The bit slot allows much of the drilling water to rise without tending to lift out the shot by passing under edge of the bit. It also allows slot which have worked up the walls of the bit to fall back to the bottom. An annular toothed bit, recommended by the makers, for soft rock, has had little success. Most shot drills dispense with a mechanical core lifter; the core is gripped by wedging it into the core barrel with angular pieces of quartz, called grout, dropped down the rods. The sludge collector, or "Calyx," is a hollow cylinder, open at top, and of same outside diam as the core barrel. The rising current passing above upper edge of *C* loses velocity in the larger cross section, and deposits the coarser sludge in the cylinder. To carry this material to the surface would require a high quantity of water. The rods are screwed to lower end of drive rod, which is rotated by bevel gears and a spline, as in the diamond drill, or square drive-rod. On upper end drive rod is a water swivel, to which is attached a yoke. From the yoke depend 2 reins, put in tension by a hand-operated winch or lever attached to the engine frame. After first few hundred ft, the weight of rods gives sufficient pressure; hence the feed mechanism is simple.

Table 61. Sizes and Weights of Shot-drilling Tools
(Ingersoll-Rand Co)

Size of tools, in.....	2.5	3.0	3.5	4.5	5.5	6.5	7.5	8.5
Hole diam, in.....	2.75	3.25	3.75	4.75	5.875	6.875	7.875	8.875
Core diam, in.....	1.5	2.0	2.5	3.375	4.5	5.5	6.375	7.375

Limitations, as compared with diamond drill, are 3 in number: (*a*) In soft rock the shot become imbedded, stopping the milling action; (*b*) Angle at which the drill can bore is from about 45° to vertically downward. With greater deviation the shot collect on lower side of hole, and bit cuts unevenly or ceases to cut at all. Maximum angle of deviation is less with bits of small diam than with large; (*c*) About 4 in is the minimum economic bit diam.

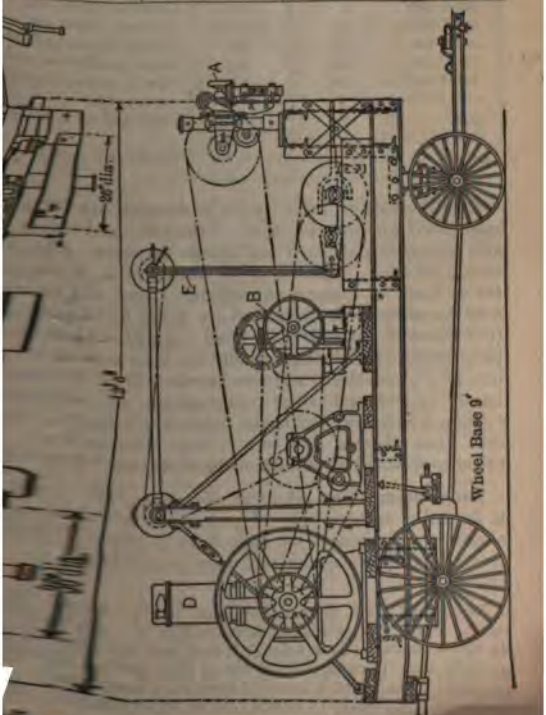
Other limitations, as loss of shot in creviced ground and inability to drill hard rock, have been alleged, but F. W. Sansom attests to its ability to drill the creviced limestones of Joplin district, Mo, and Ingersoll-Rand Co report 16 in advance in 56 min actual drilling time in corundum-bearing rock; 3.25-in hole, 2.375-in core, with consumption of 3.82 lb of shot and 0.32 in of bit.

Combination churn and core-drill rigs have been evolved to meet the first limitation above mentioned. Fig 56 shows a portable rig of this type. *a* is the core-drilling mechanism, *b* pump, *c* hoist or bull-wheel, *d* engine, and *e* the spudding device. A collapsible derrick is mounted on the frame. With this rig a hole can be quickly churned through soft formation that defy the core drill. Another advantage of the combination rig is that a churn drill will bore more quickly to a known horizon, after which the shot drill can be used to take a core.

With obvious changes, the list given for diamond-drill outfits may be used for the shot drill. Table 62 gives approximate weights of outfits, including equipment [illegible] necessary near supply. For remote districts, add about $100 to the $500 to $1,000 for the larger drills, in addition to the 300-ft [illegible] allowed in estimate for steam rigs, [illegible] motors with

e Weights and Prices of Shot Drills (Ingersoll-Rand Co)

	Diam of core, in	Power	Approx price	Approx wt, lb
	1.5	Hand	$ 600	2 600
	"	Horse	720	3 200
	"	Gas or oil engine	975	4 900
	"	Belted	825	3 850
	2	Gas or oil engine	1 150	6 900
	"	Steam	1 115	5 500
	"	Belted	1 050	5 200
	2.5	Steam	1 800	9 600
	"	Horse	1 500	9 100
	"	Gas or oil engine	1 950	11 700
	"	Belted	1 600	9 000
	4.5	Steam	3 200	28 500
	"	Belted	3 120	28 400
	"	Steam	3 700	32 200
	"	Belted	3 640	32 200
	5.5	Steam	4 560	39 100
	"	Belted	4 480	38 000
	"	Steam	8 500	82 600
	"	Belted	8 420	82 000



tion Core and Churn-drill Rig

mechanism. The bit (Fig 54) is a steel cylinder, with a diagonal slot in edge. It is screwed to the core-barrel *A* (Fig 55). Chilled shot are fed t the rods, and pass between core and inner wall of bit to t tom of the hole where they cut by a milling action.

Fig 54. Bit for Shot Boring

The bit slot allows much of the drilling water to rise without to lift out the shot by passing under edge of the bit. It also allo which have worked up the walls of the bit to fall back to the botto annular toothed bit, recommended by the makers, for soft rock, little success. Most shot drills dispense with a mechanical cor the core is gripped by wedging it into the core barrel with angula of quartz, called grout, dropped down the rods. The sludge colle "Calyx," is a hollow cylinder, open at top, and of same outside the core barrel. The rising current passing above upper edge of velocity in the larger cross section, and deposits the coarser sludge cylinder. To carry this material to the surface would require quantity of water. The rods are screwed to lower end of drive rod, which is rota bevel gears and a spline, as in the diamond drill, or square drive-rod. On upper drive rod is a water swivel, to which is attached a yoke. From the yoke depend 2 reins, put in tension by a hand-operated winch or lever attached to the engine frame. After first few hundred ft, the weight of rods gives sufficient pressure; hence the feed mechanism is simple.

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Core diam, in.....	1.5	2.0	2.5	3.375	4.5	5.5	6.375	7.375

Limitations, as compared with diamond drill, are 3 in number: (*a*) In soft rock the shot become imbedded, stopping the milling action; (*b*) Angle at which the drill can bore is from about 45° to vertically downward. With greater deviation the shot collect on lower side of hole, and bit cuts unevenly or ceases to cut at all. Maximum angle of deviation is less with bits of small diam than with large; (*c*) About 4 in is the minimum economic bit diam.

Other limitations, as loss of shot in creviced ground and inability to drill hard rock, have been alleged, but F. W. Sansom attests to its ability to drill the creviced limestones of Joplin district, Mo, and Ingersoll-Rand Co report 16 in advance in 56 min actual drilling time in corundum-bearing rock; 3.25-in hole, 2.375-in core, with consumption of 3.81 lb of shot and 0.31 in of bit.

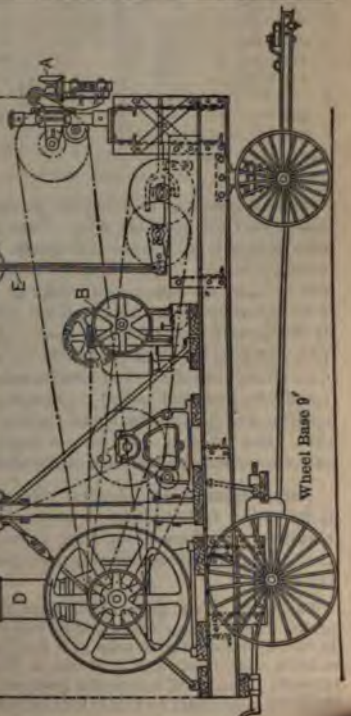
Combination churn and core-drill rigs have been evolved to meet the first limitation above mentioned. Fig 56 shows a portable rig of this type. *a* is the core-drilling mechanism, *b* pump, *c* hoist or bull-wheel, *d* engine, and *e* the spudding device. A collapsible derrick is mounted on the frame. With this rig a hole can be quickly churned through soft formations that defy the core drill. Another advantage of the combination rig is that a churn drill will bore more quickly to a known ore horizon, after which the shot drill can be used to take a core.

Fig 55. Arra ment of S boring Tool

With obvious changes, the list given for diamond-drill outfit (Art 10) can be used for the shot drill. Table 62 gives approx prices and weights of outfits, including equipment necessary near supply centers. For remote districts, add about $300 up to the 500-ft outfit, $500 for the 1 000-ft, $800 to $1 000 for the larger drills, in addition to drivepipe and casing. Boilers are included in estimate for steam rigs, nor motors with power rig.

Prices of Shot Drills (Ingersoll-Rand Co)

Power	Approx price	Approx wt, lb
land	$ 600	2 600
orse	720	3 200
as or oil engine	975	4 900
elted	825	3 850
as or oil engine	1 150	6 900
team	1 115	5 500
elted	1 050	5 200
team	1 800	9 600
orse	1 500	9 100
as or oil engine	1 950	11 700
elted	1 600	9 000
team	3 200	28 500
elted	3 120	28 400
team	3 700	32 200
elted	3 640	32 200
team	4 560	39 100
elted	4 480	38 000
team	8 500	82 600
elted	8 420	82 000



Core and Churn-drill Rig

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hydrofluoric acid, a metal container for the bottles, and a goniometer. The bott
be of constant diam outside and inside, throughout their length. Inside diam
as large as diam of hole permits. By using a phosphor-bronze container, a bo
outside diam can be used in an *E* hole (28). Walls of the bottle should be ab

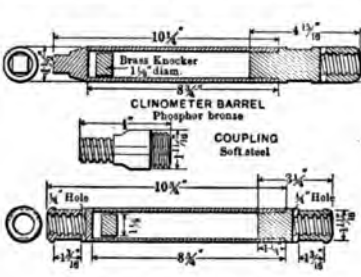

Fig 59. Two Types of Hydrofluoric-acid-bottle Container

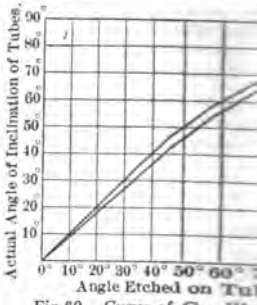

Fig 60. Curve of Capilla Correction

thick. Stoppers should be tight fitting. Containers are of steel, bronze, or l
59), bored to receive the bottle with snug fit. They are made in 2 parts, whi
together with fine threads, to exclude water, and are packed with lead or have
machined surfaces to insure a tight joint. A leak may cause collapse of the bo
to pressure of water in the hole. Containers are lowered into the hole by a wir
attached to the end or in the middle of a string of rods, in which case they are
on one or both ends for drill-rod connections. In operation, the acid is diluted
to depth of hole and method of lowering the container, the aim being to allow the
reach the desired place in the hole before any considerable etching takes place. T

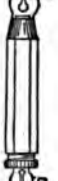

Fig 61. MacGeorge Gelatine Tube

is left at this point long enough to allow a line to be etched. St
acid and time for etching are best determined by experiment on
Container is then withdrawn, bottle washed out, and angle of
measured by a goniometer. The angle thus measured indicates
tion from the vertical that is too small; due to CAPILLARITY, whi
the acid to rise on upper side of bottle, so that the surface of th
rest in the inclined tube, when in the hole, is not horizontal. A
error varies with the angle of inclination, diam of bottle, and str
acid. It is a maximum when the bottle is inclined 45°; is larger i
of small diam, and increases with the strength (and resultant viso
the acid. Fig 60 is a curve for 1½-in bottles and dilute (1 to 1
For the strength of acid and diam of bottle employed a curve sh
plotted, giving the proper correction for any given deviation.

Gelatine method (MacGeorge's). Apparatus consists of a cont
above, and a tube to fit it. The tube, shown in Fig 61, contains in t
bulb *A* a small compass and in the lower bulb *B* a small glass
plummet, both floating in gelatine. The gelatine is heated
in the bottle until it is liquid, then placed in the container,
lowered to the proper point in the hole and allowed to set.
When the tube is withdrawn it is placed in a goniometer with
vert and horiz circles, the compass needle is brought into the
meridian, the plummet made vertical, and the amount of vert
and horiz deviation read.

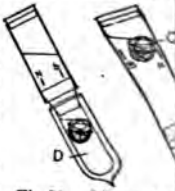

Fig 62. Maas Bor Compass

Maas borehole compass employs a combination of the above methods. It consists of a 1⅛ by 6-in tube *A* (Fig 62), in one end of which is placed dilute hydrofluoric acid; in the other a gelatine solution and a small compass *C*. For holes so deep that the gelatine would set before the compass is in position, a small thermos b contains the gelatine and compass, and a 3-in tube is used for the acid.

nproved and used on the Rand, is described
) is an electric light photographic apparatus,
alves *a*, connected by a coup-
etic needle *b* and a plumb-bob *c*,
over a gimbal *d*. Above the
e, and all are held in position
the center of the coupling, by
side of the tube, a line of small
ends projecting about ¼th in
g the lamps, needle and plumb-
screws acting as guides for the
osition. In the upper part of
a spiral spring *l* attached to it.
be, so that when the halves of
le is held rigidly together and
al springs at top and bottom.
ttached so that the instrument
ses *p*, which carry the compass
mb-bob, are of vulcanite, for
being of brass, the outer ring of
which the ring swings. On the
und the edge is a recessed ring
otographic paper, centered by
ched to a fine silk thread swung
s into a recess in the top of the
b-bob swing above, and almost
has an extra wheel *r*, with a
nnection with a copper spring *t*
an electric circuit. This lights
otographing a sharp shadow of
tographs are developed, the dip
pin pricks coincide. Fig 64
dip is 25°, and the direction N

Fig 63. Oehman's Apparatus

ed at the Cleveland Cliffs mine,
tions, with hydrofluoric tubes.
two long lines on the surface,
wered into the hole. All joints
line. The joints that are to be
rought back to the same point.
ng them in the same line when
owered, the bottle when placed
on the bottle is in line with the
f marks at the surface is deter-
nsit. This method gives accu-
when the rods drop easily in the
need not be twisted in lowering.

shows a clean break on its lower end, it
e to obtain an oriented impression of the
ttom of the hole by means of which the
be oriented. This is done by placing
f the container a piece of lead oriented
o the bottle of acid or gelatine (29)
the apparatus, the rod *R* with brass
d filler *L* replacing the plug *h* in Fig 63.
ation, are described by E. B. Durham

ation of dip and strike of a stratum or
ion of dip from the core, the strike of

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the stratum or orebody being known; (*c*) determination from drill co
true thickness of stratum.

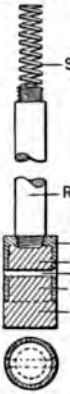

Fig 65. Device for Orienting Stub of Drill Core

Strike and dip. Theodore Simons (31) gives a convenie
of arranging calculations for strike and dip from 3 holes.
reference letters refer to Fig 66: A = lowest point at which
B = point of intermediate elevation; C = point of highest
B_1 and C_1 = projections of B and C on horiz plane through A;
of inclination of line connecting A and C; e = angle of inclination of line connecting B and C; c = horiz angle between above lines; b = angle between line of strike and extension of horiz projection of line connecting C and B; f = angle of dip; $m = \tan d \div \tan e$; $n = 180° - c$. Then $\tan b = \sin n \div (m + \cos n)$, from which the strike is obtained, and $\tan f = \tan e \div \sin b$.

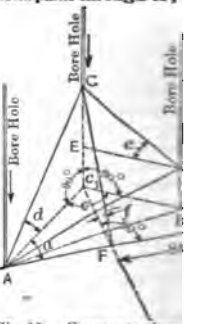

Fig 66. Computation hole Data

Dip of a bed. E. E. White (32) gives a method for determining the dip of a bed from a hole not at right angles to the strike of the bedding, knowing this strike and the direction of the hole. In Fig 67, i = inclination of hole from horiz; d = angle between bedding and axis of hole, as

obtained from core; S = difference in strike between bedding and hole; D = t
strata; p = length of a perpendicular from any point A on the hole to the beddi

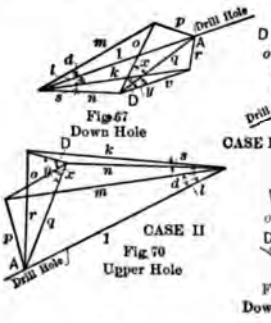

Fig 67 to 74. Various Combinations of Drill Hole and Bed (*E & M Jour*)

r = length of perpendicular from A to the horiz plane through the intersection of t
with the bedding plane; *mon* represents a bedding plane; *kvn* is a horiz plane t
the intersection of the hole with the bedding plane; *qop* and *qvr* are planes thr

to point of intersection with the bedding plane
$= n \div K$; $\sin I = r$; $\sin d = p$; $n = \cos I \cos S$;
$\sqrt{o^2 + p^2} = \sqrt{\cos^2 d - \cos^2 I \cos^2 S + \sin^2 d} =$
$\frac{d}{I \cos^2 S}$; $y = \sin^{-1} \frac{r}{q} = \sin^{-1} \frac{\sin I}{\sqrt{1 - \cos^2 I \cos^2 S}}$,

ise: $D = x + y$, when
me direction as the
steeply (Fig 67, 68).
en beds dip in same
hole, but less steeply
$= 180 - y - x$, when
osite direction to the
steeply than a plane
o the plane through
e strike line of the
). $D = x - y$, when
osite direction to the
teeply than a plane
o the plane through
e strike line of the
4).

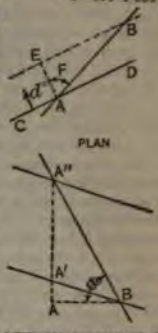

Fig 75. To Determine Thickness of Bed

(19) gives a method
true thickness of a
s dip, the direction
distance penetrated
rough the bed. In
lan of portion of hole in the bed; CAD =
EB = projection of strike line through B;
cular to CAD; d = angle of dip; F = angle
on of dip and direction of holes; P = appar-
ip; T = true thickness; $A''B$ is known in
tion from the core and survey. The derived
tan p = (AE tan d) ÷ AB = cos F tan d;
d; AA'' − AA' = A''B sin a − AB cos F
in a cos d − cos a cos F sin d).

boreholes. Hoffman (29) describes
lecting 2 boreholes; one in order to take
he vein through same hole, the other to
when 3 previous attempts had failed on
essive deviation. Apparatus used in the
ary mines' Vlakfontein borehole (Trans-
l of a pilot wedge, guide wedge, and main
lge. Mode of procedure was as follows:

in diam, the pilot wedge (Fig 76) was made
on, 18 in long, with a 6-in wedge face. The
e wedge face was threaded to fit a piece of
0.75-in hole was drilled in face of wedge and
k *a*, for surveying purposes, was made with
of wedge. The wedge was then screwed into
5-in pipe, jagged at the opposite end (Fig 77).
was filled with water and the pilot wedge and
together, were dropped in with wedge face up.
were then lowered to find out whether the
l pipe were resting on bottom of hole. If not,
aised by means of the tapped hole and dropped
ested on solid bottom, or was wedged tightly
n. Position of the wedge was then surveyed
on an oriented lead block. The guide wedge
wedge, without the tapped hole in beveled end.

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The butt end of guide wedge was screwed into bottom of main deflecting wedge (Main deflecting wedge was made of a 7-ft piece of 2-in round steel, the bottom en drilled and tapped to receive the guide wedge, and the other end was drilled with a bit for 2 in, leaving a ring of metal ⅛ in thick. The solid portion of the 7-ft len then planed, beginning at the under side of the metal ring, down to about 1 ft from forming a wedge with a 1-in radius concave face, a thickness of ⅛ in at top and bottom. The direction of face of pilot wedge was known from the survey, so t guide wedge was screwed up to turn the main deflecting wedge in the desired di when faces of pilot and guide wedges were in contact. One end of a 1⅝-in rod wa in the metal ring at top of main deflecting wedge, and riveted to it with 2 0.25-in rivets; to the other end of the rod was attached 10 ft of 1⅜-in rod, connected to t line of rods by a ball-bearing swivel. On lowering the apparatus into the hole, th wedge engaged the pilot wedge, and turned main deflecting wedge into the prop tion; pressure was applied to shear the copper rivets, and the rods were raised, the deflecting wedge in the hole. When the position of the wedge was found cor survey, drilling was resumed with 1⅝-in tools, the deflecting wedge acting as a g throw the tools into a new line.

In the Witpoort (Transvaal) borehole a means was devised to overcome neces reducing the diam of hole at each successive deflection. After the main deflecting was set, the 1.625-in hole was drilled about 12 ft deeper and the wedges were withdr a recovery tap. A 13-ft piece of 1-in round iron, bent about 0.25-in out of lin fastened to the end of the rods by 2 copper rivets, lowered into the hole and manip until it passed into the branch hole. The rivets were then sheared, the piece left in and the hole cemented to about 1 ft above top of the iron. After the cement ha drilling was resumed with 2.125-in bit. The iron in the hole passed up into the h it over into the branch, and was brought up in the core.

18. Choice of Boring Methods

Chief factors: (*a*) depth of hole; (*b*) speed desired; (*c*) first cost of (*d*) operating cost; (*e*) character of material to be penetrated; (*f*) tra tation facilities; (*g*) information desired.

For shallow work in earthy material or soft rock, a hand rig is ordinarily pref For holes more than 50 ft deep, or in hard rock, a power rig should be used, unless inc first cost, including transportation, outweighs the saving. Speed increases, in ge with size and weight of rig, except that for shallow holes and frequent moves the g time consumed in dismantling, moving and setting-up may offset the greater d speed. Core drills cost more than churn or hydraulic rotary drills; and first cost mond drills, including carbons, exceeds that of shot drills. Churn and rotary d are usually cheaper than core drilling, but the question of operating cost is so close lated to the purpose of the work, character of material penetrated, and labor avai that each case should be considered separately. Character of the material is some the determining factor. Thus, the hydraulic rotary is preëminently suited to d in the soft ground encountered in oil wells in the Gulf-coastal plain, other types o failing completely. Diamond drill is unsuited to very soft or badly shattered hard To secure information as to physical character of or for rock in place, core drill must be

Table 65. Summary of Core-drilling Costs in Shale and Sandstone, Nova Scotia, 1910–1913 (See Tables 66, 67)

Kind of drill	Diam of core, in	Total depth, ft	Av depth per hole, ft	Av speed, ft, per wk'g hr	Av c per $
Hand diamond, belt driven	1 5/16	1 652	331	0.98	0.8
Steam diamond	1 3/16	1 144	286	0.91	1.0
Hand shot	1.75	268	134	0.68	1.5
Horse-power shot	1.75	459	115	1.05	1.3
Steam diamond	2	5 943	594	1.00	1.0
Steam shot	6	6 326	575	1.05	1.6

of 2-in Steam Diamond and 6-in Steam
lstone, Conglomerate, and Coal, Nova
N S Dept of Mines, 1910–13)

Carbon, bits, core shells, lifters shot, grout, etc, $	Casing and drive-pipe, $	Lumber for outfit, $	Freight and haulage, $	Misc repairs, water, etc, $	Total, $	Kind of drill
0.08	0.03	0.05	0.06	.....	0.775	*D*
0.15	0.01	0.01	0.06	.....	0.724	"
0.18	....	0.01	0.04	0.02	1.006	"
0.13	....	0.00	0.01	0.01	0.700	"
0.13	....	....	0.03	.....	1.845	"
0.22	....	....	.....	0.09	1.569	"
0.04	....	0.00	0.01	.....	0.846	"
0.11	....	0.01	0.08	.....	2.012	"
0.12	....	0.00	0.01	0.00	1.040	"
0.20	....	0.01	.....	.....	1.354	"
0.04	0.02	0.08	0.28	.....	1.441	*S*
0.15	0.01	0.01	0.03	.....	1.070	"
0.10	0.01	0.02	0.26	.....	1.433	"
0.15	0.05	0.09	0.26	.....	1.822	"
0.14	0.12	0.06	0.15	.....	1.292	"
0.15	0.00	0.01	0.14	.....	1.031	"
0.12	0.47	0.04	0.15	.....	2.455	"
0.05	0.18	0.10	0.37	.....	2.323	"
0.05	0.73	0.01	0.33	.....	3.231	"
0.12	....	....	.....	.....	1.480	"
0.31	....	....	.....	0.27	3.994	"

nond. *S*, 6-in shot.

land Diamond, Steam, Hand, and Horse-
Sandstone, Nova Scotia, 1910–13

Fuel, $	Abrasives, bits, core shells, lifters, etc, $	Casing, $	Lumber, $	Freight, $	Misc, $	Total, $	Kind of drill
.15	0.05	.....	0.01	....	....	0.684	*A*
.09	....	.....	0.04	0.40	....	1.811	"
.14	0.05	0.006	....	....	....	1.082	"
.04	0.05	0.032	....	0.24	....	1.172	"
.04	0.05	0.002	0.03	0.02	....	0.816	*B*
.03	0.05	0.004	....	0.02	....	0.805	"
.05	0.06	0.004	....	....	0.10	1.105	"
.04	0.06	0.006	....	....	0.05	0.837	"
.05	0.04	0.009	....	0.02	....	0.741	"
.00	0.04	.....	....	0.07	....	1.255	*C*
.00	0.06	.....	....	0.11	....	1.380	"
...	0.05	.....	0.04	0.09	0.33	1.497	*D*
...	0.03	.....	....	0.06	0.25	1.251	"
...	0.01	0.006	0.05	0.43	....	1.920	*E*
.00	0.01	.....	....	0.18	0.05	1.458	"

nd diamond, small engine. *C*, 1¾-in horse-
E, 1¾-in hand shot.

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19. Drill Manufacturing and Supply Companies (Incomplete l

Wash-boring rigs: Keystone Driller Co, Beaver Falls, Pa; Cyclone Drill Co, (Ohio; Loomis Machine Co, Tiffin, Ohio; Sullivan Machy Co, Chicago, Ill

Empire drills: New York Engineering Co, N Y

Oil-well rigs: Oil Well Supply Co, Pittsburg, Pa; Carnegie Steel Co, Pittsbu

Portable churn drills: Keystone Driller Co; Cyclone Drill Co; Star Drilling M Co, Akron, Ohio; Loomis Machine Co; Carnegie Steel Co; Oil Well Supply Co

Diamond drills: Sullivan Machy Co; Standard Diamond Drill Co, Chica E. J. Longyear Co, Minneapolis, Minn; H. R. Ameling Prospecting Co, St Louis,

Shot drills: Ingersoll-Rand Co, N Y; Cyclone Drill Co; McKiernan-Terry Drill C Dobbins Core Drill Co, N Y

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VELOPMENT AND OF MINERAL SITS

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Acknowledgment is due to R. K. Warner for assistance in preparing illustrations tables in Art 59–127.

initions

as quarrying in open cuts and the work-
d work. In a given orebody, mining
:

ebody when found. It is undertaken to gain
alue of the orebody.

to and in a proved orebody, for mining and

racting the ore.

It may be difficult to distinguish between
ploration and development, as the different
her. Confusion also arises when these terms
perty containing several orebodies. In such
rt of exploration. In certain orebodies, ex-
peration by boring; as in the disseminated
lesabi iron ores and gold placers which are

value has been proved by exploration.

ng an orebody are based largely on its geo-
of orebodies is invaluable (Art 5; Sec 2, Art
e determined by the shape of an orebody and
sits are classified as:

g at high angles.

dipping at low angles.

shape standing at any angle. Fig 4 shows

n metal mining. (Letters refer to Fig 1 to 5.)

k above a vein (*h*); called "roof" in bedded

a vein (*f*); called "floor" in bedded deposits.

m the surface (*d*), (*e*); an inclined shaft is

l, and running through country rock or ore
body (*g*).

body, parallel to its strike (*k*).

y is opened up, and from which mining be-
nse as "drift," or to cover all horiz workings
uts in Fig 2 would be called the 1st level, or

t, sunk from an interior point in the mine, as

ze excavated upward (*l*). Terms winze and
enings as at (*n*), Fig 1.

horiz underground passage, coming to surface
lso for such a passage open to daylight at one
), Fig 5, or DRIFT TUNNELS (*r*), Fig 1.

rb "to stope": "To excavate ore in a vein by
ings, one immediately over the other, or vice
The word is used carelessly; as a noun, it

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applies to any underground excavations (s, Fig 1), in which ore is mined, as distin from other excavations in ore, as drifts, raises, winzes.

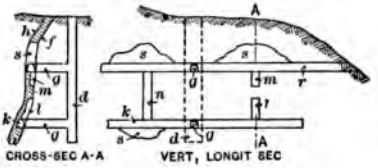


Fig 1

Fig 3

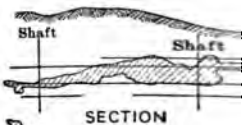
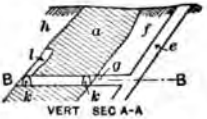


Fig 4



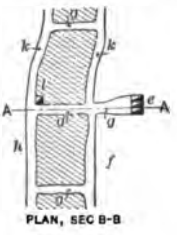


Fig 2

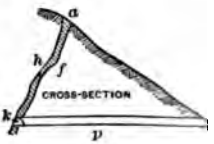


Fig 5

Back. The ore between a level and the surface, or between 2 levels; also desig the roof of a drift or stope.

Adit. A tunnel driven for drainage purposes (also for transport).

PROSPECTING AND EXPLORATION

2. Conditions Warranting Prospecting

Search for mineral is largely guided by knowledge of the geological ass tions and peculiarities of known orebodies. It is done chiefly by miners who absorbed the details of ore occurrence in mines where they have worked. are keen observers, but prospect only for types of orebody covered by experience. This results in many discoveries by inexperienced men who pect without preconceived ideas. Some famous orebodies have been fo accidentally. Crane (2) lists 130 gold and silver mines in the U S, of whic were discovered by chance.

Theoretically, the trained economic geologist should make the best prospector, bu chance of finding commercial orebodies is so small that such men can not afford to pro unless employed by mining corporations; under present conditions, scientific prospec is rare. Engineers generally direct the work of exploration. Economic geology aid

unlikely ones. Most plans for exploration
origin of the orebody concerned (Art 5).

d be kept low on account of the high risk
d and unnecessary expenditure for equip-
e cost of strictly prospecting work; while
pits can be obtained, the required amount
Closer estimates are possible for system-
property generally gives an idea of the
value. The estimated cost of this work,
s a measure of the risk involved (Sec 25).

cting, to determine whether shoots of
em. Since practically all exposed out-
een located, the field for prospecting at
or for those orebodies the outcrops of

often covered with a black or brown stain
ases, if an orebody and the adjacent rocks
t be distinguished unless pieces are broken
of this kind which have not been found.
atient detailed tracing of float (Art 3).

pecks of metal or ore minerals, justifies
nsidered in connection with local geol-
of quartz float in a schist or slate area
small to be of economic value.

Many data have been collected about
of different types of orebody. Recog-
district warrants prospecting. Con-
t prospecting; thus, search for coal is
ssociations and characteristics of ore-
to note that geological conditions at
ose at another and yet not be accom-

or slag exist in different parts of the world,
istorical information about old EUROPEAN
sionally, an old book or record has afforded
eral. Reasons advanced for the abandon-
by savages, (b) pinching out or faulting of
technical knowledge on the part of the
ater in particular prevented mining much
show evidence of skilful mining (3); the
rgy of which was unknown. No develop-
requently, the cost of acquiring property
The slave labor used was the only capital
was abandoned or exhausted. The pur-
sent. In such circumstances the ancients
dies. The existence of old workings is not
kable under present economic conditions.
workings in RHODESIA, where there are
tz veins of which there is no written his-
sionally 100 ft; in most cases mining had
width, about 2 ft. Indications were that
The following conclusions from this work
length of pay shoots at the surface. (b)
gs. (c) Dip of vein is toward the steeper
over an open cut breaks down under long
of the footwall bank of the excavation.
but is usually less than, the width of the

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old workings. Very narrow veins are indicated by narrow workings, with sm containing much waste. (*e*) Grade of pay shoots, vague, but above aver grade if high values be rejected. (*f*) Large and abundant dumps in comparison wi workings, especially dumps containing large pieces, indicate low-grade ore sparse dumps along strong workings indicate that whole vein was worked as pay that its grade was probably good. Roughly, the value of a vein is proportion ratio between extent of old workings and extent of the dumps. (*g*) No light is t old workings on permanence of the vein in depth or on its gold values.

3. Prospecting Methods

Surface methods in common use are: tracing float, tracing by p trenching, and test-pitting. Search for mineral which does not outcrop at considerable depth is done by boring (Art 8), or shaft sinking (Art Sec 7). Methods are varied and combined to suit local conditions or th of the prospector. There are many prospecting devices for use in specia

Tracing float (Cornish, "shoading"). Pieces of ore (float) are brok outcrops by processes of erosion, and gradually work their way downh streams, where they may be carried long distances. The prospector float on a hillside, or in a gulch or stream, tries to follow it back to its sou rough idea of the distance which float has traveled is gained from the s abundance of pieces, and their roughness or water-worn condition.

Many outcrops which are the source of float are concealed by a covering of such cases, at some point below the outcrop the float disappears, or "goes Trenches or test pits are used to follow float farther. The distance between soil- outcrops and the point at which float goes down varies with depth of cover, topo and climate. On flat slopes (6%) in desert regions of southwest U S, float has bee on the surface within a few feet of outcrops covered with 1 to 2 ft of soil. In Ma district, Mont, at one point float went down on a hillside and was followed by pits ft before finding the outcrop; bedrock was 10 ft to 12 ft below surface. In the fa where freezing and thawing have gone on a long time, the "creep" on the hillsides and float may be found far from the outcrop.

In general, tracing float is applicable only in searching for ores tough enough to disintegration by erosion. Gold-quartz, or pyritic ores with siliceous gangue, in p lar, yield float which is resistant and easily recognized. The value of, and result the method depend largely on the personal equation of the prospector. Knowl ore, keen observation, physical strength, and unending patience are essentials. prospectors use the pan or horn spoon to estimate the content and value of float, because of expense and inconvenience of obtaining assays.

Tracing by panning (also called "tracing" in western U S, and "loam in Australia). This, like tracing float, is based on recognition of the resi erosion. Weathering releases small pieces of metal and minerals from out and these migrate downhill as do pieces of float.

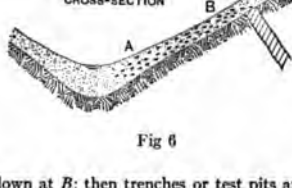

Fig 6

Fig 6 shows an orebody outcropping a soil cover, and the usual position o metal or sulphides derived from it by e In a new district, the prospector along the bottom of hillsides at an ele (*A*) sufficient to avoid the debris brough the gulch from upstream points. Sa of the top 2 or 3 in of soil are panne specks of metal or sulphides are found, ning is continued uphill until the trac down at *B*; then trenches or test pits are dug. The placer miner's pan, or a fryin 4 to 6 in diam, is used for panning. In practiced hands the latter gives accurate re and is better where scanty. Float found in connection with this work sho crushed and panned as an aid in determining the source of "traces." This metho developed by "pocket hunters," and has been used chiefly in prospecting for gold, w

d in the pan even when present in minute
909, to find a concealed outcrop of a very
was found about 700 ft from the outcrop.
r sulphides. As in tracing float, a correct
al knowledge. (See also Art 6.)

es). Their use for prospecting and ex-
f depth for economical work is 6 to 7
a trench with a shovel. Principal
an aver depth of not over 3 or 4 ft;
prospecting and exploring large areas.
or traces under cover. Trenches are
or to the supposed strike of the ore-
s useless to carry trenches to bedrock.

often indefinite, giving no indications of
, several trenches at right angles to one
es of parallel ones, which may parallel the
g on property adjacent to developed ore-
right angles to the strike of the known
rallel orebodies if they exist (Art 5, 6, 7).
l, such prospecting should be done with 2
her; their distance apart is from 50 ft to
the vicinity.

as possible; it depends on their depth and
12 to 14 in wide at bottom and 18 to 21
mum bottom width is about 15 in, width
ne soil. These dimensions are satisfac-
ed examination of bedrock is necessary,
ch cases, minimum bottom width is be-
ng with depth of soil. In trenches on
west point and run uphill, so that they
en ground and tundra of the far north,
, due to thawing on exposure to the air.
nching and examples of cost, see Art 7.)

ft deep is determined more by character
men picking and shoveling in trenching
ver, 4 to 5 cu yd. The rate is lower in
examining float and bedrock. Where
nd 2 to 3 ft into the decomposed surface
ting ground are very variable, no accu-
Wet ground retards progress and makes

vium too deep for trenches. Their
and water, and requiring little or no
to 100 ft; for over 30 to 50 ft, and,
ompete with boring methods (Art 8;
k above the point at which float or
nd, or if panning shows mineral, the
nd a new one started above. This
d or the prospector is discouraged.

driven from the bottom of a test pit, and
ne last stages of such work, especially in
s trenches; they expose only a small area
irregular veins. This objection does not
these, test pits are located with reference
cally, or at random. (For principles, see

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Cross-section of test pits may be circular, elliptical, or rectangular. **Diam o** pits is 30 to 36 in; elliptical pits are 36 to 42 in long by about 28 in wide; **rectang** are rarely smaller than 4 by 4 ft or 4 by 5 ft. In point of cost and speed, **small** pits are preferable to rectangular; the amount of material to be excavated **and** from a 36-in circular pit is about 0.35 of that from a 4 by 5-ft rectangular **pit.** or elliptical pits are excavated with a short-handled pick and shovel; **the** straddles the handle. A windlass is used for hoisting. Galvanized-iron **water p** good buckets for small pits; ordinary windlass buckets are better for **larger rec** pits, 2 buckets or pails being provided for each windlass. A 0.25-in wire rope s used instead of a hemp rope in small pits over 30 ft deep; at greater depths, the t hemp rope across the windlass barrel is apt to throw the bucket against the walls lodge loose material. Objections to circular or elliptical pits: (*a*) they requi trained and willing to work in an awkward position; (*b*) there is no room for dri blasting, if boulders are encountered; (*c*) it is practically impossible to timber th rectangular pits a soft stratum can be close cribbed with plank at slight expen crew on either kind of pit consists of 2 men, one digging and the other on the

Cost. Prospecting for brown hematite in Georgia and Alabama. **Round tes** to 36 in diam; depth, to 50 ft; cost, 35 to 50¢ per ft. In ordinary unconsolidated 2 men sink 10 ft per day (5). Near Birmingham, Ala, a large number of 30-in pi 60 ft deep, have been sunk at 35 to 45¢ per ft; speed of sinking, 18 to 20 ft boulders of ore and rock frequently reduce speed to only 8 ft per day (6). Ecl that, for iron-ore prospecting in U S, the cost for small untimbered test pits, 10 deep, is 30 to 50¢ per ft (7). In Lehigh Valley, Penn, 2 men sink 30 to 36-in pits t of 50 ft at 20 to 30 ft per day (8). In Nevada, the author sank 40 by 29-in ellipti with Chinese labor, in easy gravel, at 20 ft per 12-hr day for first 40 ft. Below 40 f was hard and clayey. Aver speed for depth of 100 ft was about 5 ft per 12-hr sh tract work). Under same conditions, white labor on day's pay averaged 2 to 3 ft p shift in test pits of 4 by 5 ft cross-sec.

E. S. Dickinson gives following data, referring to Table 1: cross-sec of pits, 4 the ground dulled picks rapidly; foreman's wages, $3; laborers, $2. Pits cribb 2-in hardwood plank; cribbing removed and reused. Work done by 3 men; each in a separate pit as long as he could throw dirt out, then another man helped by t dirt back from edge of pit. Pit was deepened by help of a platform, one man sh from bottom, the other shoveling from platform to surface. When depth beca great to allow this, or sooner if many boulders were encountered, a windlass was i and 3 men worked together on one pit. Cost of superintendence is not included i

Table 1. Costs of Test Pits in Clayey Hardpan, Many Large Boulders

Pit	Depth, ft	Hours labor	Total cost	Cost per ft	Pit	Depth, ft	Hours labor	Total cost	
1	17	80	$18.00	$1.06	9	38	255	$64.00	$
2	24	140	29.00	1.21	10	33	240	54.00	
3	7	15	3.50	0.50	11	10	60	12.00	
4	22	75	16.50	0.75	12	9	100	22.00	
5	23	100	24.00	1.04	13	9	20	4.00	
6	26	240	48.00	1.84	14	17	60	12.00	
7	41	270	56.00	1.37	15	12	55	12.50	
8	46	335	72.50	1.58	Totals	334	2 045	$448.00	$1

Aver depth, 22.3 ft; aver speed per hr, 0.114 ft.

The author sank 11 rectangular test pits; aver cross-sec, 4.1 by 4.7 ft; aver 7.5 ft; material, very hard clayey gravel overlaid by 6 in loam; deepest pit, 10 windlasses used; labor, 161 man-hr; aver speed, 0.51 ft per hr. In Lake Superio districts, rectangular test pits cost about $1.25 per ft. For other details see Art 7 Sec 3, Art 11. For use of test pits in systematic surface exploration, see Art 6.

Hydraulic prospecting (booming or hushing). Water, where available great aid in stripping off soil for close prospecting of bedrock. Sometin small stream can be led by a ditch onto a hillside above the area to be prospe

made to flow over any desired section
rock.

. Where water is scanty, small reservoirs
he water is released, and rushing down the
ated, if necessary, until bedrock is clean.
ates. In other places a small quantity of
loosened by picks, is washed away. Large
alt, Ont, shallow surface cover is completely
rt 7).

ee from stones (Sec 9). Pipes are 1 to 2 in
end is filed to a cutting edge, and has a slot
in gripping the soil and
oth and character of soil,
riven with a maul, or a
of pipe is protected by
when used in stony soil;
e deepens. The pipe is
The method is cheap
sed through. If soil is
eral, drivepipes can be
s, limit of depth is usu-

Fig 7. Lower End of Drivepipe

ods are used in search
or under soil. The
fter than surrounding
or.

d to locate quartz veins
st quartz was found in
feel" and sharp clink
om the softer country

f the use of piercing: in France for buhr-
l and clay; on the Isle of Man for shallow
rod; in the Furness district, England, for
lor); in South Carolina, for phosphate nod-
Burma, bamboo rods are similarly used to
gem-bearing gravels.

ethod of prospecting for phosphates on the
irregular patches in the river bed, and is
oats by dragging a bottle filled with water
e neck of the bottle, and a cardboard dia-
the ear, or the sounder holds the end of the
a deposit, floats are set to show its outline

long outcrops of one geological forma-
in working out geological structure, or
to ore occurrence. At Cripple Creek,
ccia; fir trees grow on granite. In the
ash-bearing dikes, there is a noticeably
the southern Appalachians, the vegeta-

dogs, gophers, badgers, and ants, sometimes
out when digging holes. Such material is
r specks of ore minerals.

persons to be efficacious for finding water

eriodically in the mining camps. Some are
dvantage of the difference in electromotive

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force existing between portions of an orebody. Extensive experiments at Virgi and Eureka, Nev, showed that differences in potential are very minute and so err they are useless as a basis for prospecting (15).

Electroscope. Radio-active minerals have the power of discharging troscope; under suitable conditions the rate of discharge is proporti amount of radio-active substance in the mineral. Electroscope is engineers and prospectors to detect radio-activity, and for quantitativ Radium is obtained solely from uranium ores, the amount of radium b constant relation to amount of uranium present; normally 1 unit of ura accompanied by 3.3×10^{-7} units of radium. Thorium compounds and also discharge an electroscope.

Fig 8 shows a simple electroscope suggested by B. B. Boltwood, which can be the field. It consists of a 5-cent piece, a quarter dollar, or an iron washer A; sealing wax or candle grease C; an upright B made fro of tin 0.2 in wide (or a 6d or 8d wire nail flattened at the leaf D. The latter may be a piece of aluminum le 0.1 in wide, obtainable from any dealer in painters su thin tissue paper, both sides of which have been coa graphite by rubbing with a soft pencil. Leaf is attach right with mucilage or flour paste. Bottom of upright be insulated from base A by a layer of sealing wax; must not be touched with the fingers after it is in p cover protects the apparatus from air currents; for th sions shown in Fig 8 an ordinary glass tumbler placed upward will serve. Electroscope is charged by touchin right with a piece of hard rubber (a fountain pen), o wax, which has been electrified by rubbing on one's sleeve. A protractor drawn on cardboard, so that i point is at same height above base A as the point E, is placed behind the tumbler angle read between leaf and upright. Readings are repeated at regular interva termine rate at which electroscope will discharge itself by leakage. The mine tested is placed inside the tumbler; radio-activity is proved by a rate of discha than that due to natural loss of charge.

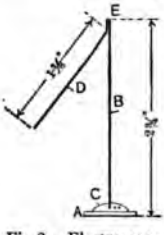

Fig 8. Electroscope

This crude electroscope gives surprisingly accurate quantitative results if hand fully. Table 2 shows results of tests by H. G. Mead under author's direction, o of samples prepared by mixing pitchblende (35.7% U) with inert material in pr portions to give samples containing 2, 5, 10, 15, and 25% U. Samples were through 20 mesh and placed next to base of electroscope in a small pill box. Sa box was used for all samples, each being filled level full, and placed in same posit respect to electroscope. Rate of discharge was determined as above, corrected fo leakage, and net rate of discharge computed in degrees per min per % U. Max e paper and aluminum leaves were 0.7% and 1.9% U respectively. Aluminum leaf more sensitive than paper, and requires greater skill and patience in mani

Table 2. Tests for Radio-activity by Electroscope

Per cent U in sample	Paper leaf				Aluminum leaf			
	Rate of discharge				Rate of discharge			
	Deg per min	Leakage, deg per min	Net rate, deg per min	Deg per min per % U	Deg per min	Leakage, deg per min	Net rate, deg per min	D m
2	0.162	0.0105	0.151	0.0755	0.46	0.007	0.45	
5	0.338	0.0105	0.327	0.0654	1.04	0.007	1.03	
10	0.623	0.0105	0.612	0.0612	2.61	0.007	2.60	
15	0.965	0.0105	0.954	0.0636	3.00	0.007	2.99	
25	1.710	0.0105	1.699	0.0680	5.00	0.007	4.99	
Aver..	.....	.....	.....	0.0667	.....	.....	.....	

the containers used. For quantitative work
d to standardize the electroscope; unknown

portation and microscope for reading angles,
olo; price, $60. Schieffelin & Co, 170 Wil-
radium detector; for rough qualitative tests
space 2 by 3 by 1 in.

Surveying

of locating orebodies consisting wholly or in
Lake Superior districts for locating magnetic
non-magnetic hematites (Art 11), and some-
eliferous pyrrhotite. Magnetic surveys are
magnetometer. Following notes are taken
Smyth, and E. B. Durham (117).

's compass consists of a magnetic needle
in a vertical plane. Needle is counter-
n so that it remains horiz (reads 0) un-
be adjusted for new localities by setting
, moving the counterweight until this
netic material, one end of needle drops;
lip angles are called + or −, depending
ectively is drawn downward. In taking
l in plane of magnetic meridian. Dip
meridian is indicated when compass is
instrument is hung from a bail, so that
sition by its own weight. Compass is
rrangement for determining meridian is
n a staff. The dip compass, hung from a
arallel to the needle of horiz compass.

Fig 8b. (*E & M Jour*)

Observations are taken at regular intervals
OX and $O'X'$ are laid out within sight of each
a multiple of side of square to be used; angle
etc; and range poles are placed at O' and A;
a helper with front end of tape at a; helper
oint O' is reached, range pole there is trans-
s on line $A'A$, and process is repeated.
nethod used on Cuyuna range, Minn, where
nagnetite and attract needle. There are no
with sand and drift. Fig 8b shows idealized

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conditions in a 40-acre tract. Observations are first taken along boundaries a intervals (say 100 paces); max dips occur at *A* and *B*; stakes are set on line *AB* vals of 100 or 200 ft and readings taken on lines at right angles, as on *DE*. Re platted and a line drawn connecting points of max dip, which lies approx over t netic slates. Lines at right angles to *AB* are then staked out 300 ft apart, and exp is conducted by boreholes at regular intervals along them (see Art 11).

Results of dip-compass surveys are usually platted to scale and is LINES (connecting points of equal dip) give information as to form and of underlying orebody. Following are general rules for interpreting such Isoclinic lines are normal to lines of force in the magnetic field produce orebody, and they are adapted to indicating mild but extensive fields dips occur over the magnetic pole of orebody or the enclosing areas around the outcrop. Small orebodies near surface may exert as much at as large orebodies deeply buried, but their effect extends over a small Slow steady increase of dip over a large area indicates a large deposit; dip limited to a small area indicates a small deposit. Lean ore near surfa give same indications as rich ore in depth.

Horizontal-compass surveys. Observations are made at regular i over the area to be tested, by same general methods used with dip c The needle is used to measure local attraction caused by magnetic or A preliminary transit survey may be made and lines established by m which compass is oriented. For rapid and extensive work the dial con preferable; "it consists of a small portable sun-dial, provided with a c needle swinging within a graduated circle, which, when instrument is horiz. On a sunny day if instrument is set up, leveled and turned in until shadow of the gnomon (a thread) falls on the division of hour-circl sponding to apparent time, zeros of graduated circle are in true merid needle gives declination" (117).

For correct results, hour-circle must be graduated for a latitude near point of tion, angle between gnomon and plane of hour-circle must = latitude, zeros of circle must be in vertical plane of gnomon, plane of hour-circle must be level, parent time must be known. Apparent time is obtained from standard time by c for difference in longitude between point of observation and standard meridian, addition or subtraction of equation of time obtained from a solar ephemeris (Sec 13). W. L. Cumings gives following method of determining solar time (117). free from local attraction is selected, dial-compass is used to make an observ Polaris, and a suspended string is lined in between star and compass sights. Aft ian is determined, compass is set on it while sun is shining and the threads' shad true solar time. Observer's watch is set to this time, and used for orienting co subsequent work; compass is set on meridian each morning for correcting watc

Platting. Compass deflections are platted to scale on a map and is LINES (Sec 17, Art 10) drawn connecting points of equal deflection, inter where necessary; surface contours should also be shown.

Interpretation of maps. H. L. Smyth gives following points: Final assumed by needle influenced by a magnetic orebody is in a line which is sultant of the horiz component of force of the earth's magnetism and tha deposit. Important points to be located are those of zero and max de called CRITICAL POINTS. Relation of critical points to deflections of nee their position with regard to disturbing forces, are illustrated by consid of several partly ideal cases:

(*a*) A single magnetic pole, regarded as upper (north) pole of a vertical-bar lying at a depth *h* below a level surface, the lower (south) pole of which is too far affect compass. In this case, angle of deflection is a function of 2 variables, the of point of observation from the pole and its direction from pole. Point of zero is directly over pole; points of max deflection lie on a circle the center of whi

ervations. Under assumed conditions, $h =$
ntric circles, the center of which is over pole.
n decrease with distance from pole.
rial has uniform width and is very long in
a level upper surface over which magnetism
niform depth by alluvium. These are ideal
rmations in Lake Superior region approach
of magnetic rocks has certain general char-
d at 2 poles, but is distributed over or near
cts north-seeking end of compass needle.
When magnetic rocks dip steeply, their
ane of observation not to affect magnetic
ries with quantity of magnetite contained,
of strike and dip of magnetic formations.
ction of earth's force, and is least for rocks

f zero deflection are situated over middle of
gnetic line and gives strike of the strip. A
line when a strip strikes E and W. Points
UM LINES, one on each side of the strip and
stance of a point on MAX line from magnetic
width of strip, then $d^2 = h^2 + a^2$. d is de-
calculated. Numerical value of deflections
its strike is E and W (magnetic), the com-
ip; to north of strip, the deflection depends
arth and deposit. The needle may point S
ance from the strip until a point is passed
eedle assumes any position; farther north,
n pointing of needle and a point of indiffer-
a N and S direction. A strip which strikes
ns on both sides at points equidistant from
oduces greater deflections at stations on its
on its southern side.
if d = half the distance between the points
hen $d^2 = (h^2 + a^2) \div \sin^2 \Delta$. In case of an
halfway between max lines, but is always
dips. Critical points indicate strike, direc-
and also give limits of width. Width can
boundaries must lie inside the max lines;
lower boundary falls inside one max line,
ine and other max line.

l dimensions and depth of magnetic
ially in case of small deposits, but the
may be drawn from survey. Professor
gnetic strip, it is safe to draw following
that they have a uniform surface dis-
will fall outside of and not far from
y has a steep dip; (*b*) inequalities in
icate direction of dip; (*c*) depth is very
g deposit to be a single magnet.

rms. Following description applies to

the needle is mounted between 2 pivots;
d on the E and W line of a graduated circle,
a horiz or dip compass. The zero points of
ints. An arm, about 10 in long, is attached
f box; it is therefore horiz when the instru-
a prolongation of the E and W line of com-
agnet (called deviating magnet) is held by a

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the value of R is found from expression, $R = H\ (\sin A_0 + \sin A)$. Results are plot scale, and R and H drawn in at each station. F is then found graphically by comp the parallelogram of forces at each station; lines F will converge towards magnetic of deposit C and indicate its position.

5. Geological Data for Prospecting and Exploration

(By J. D. IRVING, Professor of Economic Geology, and A. M. BATEMAN, Assist Professor of Economic Geology, Sheffield Scientific School, Yale University)

a. Introductory

All mine exploration rests upon some geological theory as to the probable position, size, extension, mineral character, or value of an ore mass. Correct geological kno frequently enables one or more of the above features to be determined in advance ploration, often with considerable accuracy. In some cases geology furnishes n whatever, and exploration is necessarily blind. Following are some common con under which geology may aid in exploratory work, with methods by which results n secured. (See also Sec 2, Mineral Deposits.)

Sources of information: (*a*) PROCESSES OF ORIGIN OF A DEPOSIT dete its form, size, and geological position, the mineralogy and value of the pr ore, probable occurrence in undeveloped districts, and success or failure ploration for more ore in a partly developed district. Production of ore r is only one of many effects of ore-building processes. If the other effects recognized, they may assist in discovering ores. Favorable indications insure ore occurrence, but, if they are entirely lacking, ores of the type no cated will almost certainly not be found. (*b*) PROCESSES OF SUPERFICIAL AL TION acting on an orebody after its formation. These determine infe from nature of outcrop as to value of undeveloped orebodies, depth to developed ores may be expected to go, and often the presence or absence

Table 3. Classification of Ore-forming Processes

Ore Deposit Group	Process of Deposition
I. Bedrock (or primary) deposits	Due to processes operating on bedr
A. Syngenetic deposits	Due to rock-making processes
1. Igneous	Due to magmatic differentiation proces
2. Sedimentary	Due to processes of sedimentation
B. Epigenetic deposits	Due to processes of mineralization
3. Fillings of cavities	Due to chemical deposition in rock ope
a. Fissure veins	
b. Other rock openings	
4. Replacement deposits	Due to processes of metasomatic replac
5. Contact metamorphic deposits	Due to combined processes of contact amorphism and replacement
II. Disintegration (or secondary) deposits	Due to processes of disintegration a concentration at the surface
6. Mechanical concentrations, placers	Due to sorting action of water or wind
7. Chemical concentrations	Due to chemical deposition at surface
8. Residual concentrations	Due to removal of worthless matrix
III. Metamorphic derivatives	Due to process of dynamic and r metamorphism acting on pre- deposits of all kinds or on roc

This grouping is not intended to supersede other classifications, but to show the of processes of ore formation to the genetic classification on which the practical c ations that follow are based.

orebody at all. (*c*) TIME OF FORMATION OF A DEPOSIT, relative to other g events, determines the time relation between undeveloped ore masses and intrusive rocks, metamorphism, sediments, and all geologic events whic

e of knowledge promises to yield valuable
ISTURBANCES WHICH HAVE AFFECTED AN
s useful when a deposit has been faulted
n. Faulted portions of orebodies may
morphism may be recognized; if not too
ay still be determined and used as under
m, geology furnishes little aid.

igin. Clear recognition of ore-forming
pon which prospecting, exploration, and
It is only when these processes are under-
osition, form, size, continuity, and often
essary preliminary to discussion, is to be
ing processes, rather than of ore deposits
to Kemp's classification (Sec 2, Art 17).

sits. Bedrock deposits occur in solid-
n soil or detrital material deposited in
med by mechanical or chemical decay of
ter surface concentration.

-making processes, and are integral portions of
igh enough in useful mineral to constitute an
sult of solidification from fusion; or they may
sion, geological position, and indications as to
ck types which they represent. Examples:
itaniferous magnetites, and Clinton iron ores.

by processes operating upon rock masses sub-
hey are enclosed; they result from ore-forming
cesses. Nearly all are the result of deposition
orm, size, geological position, and the indica-
ven region are therefore dependent upon the
tricted and intensely localized phenomenon.
(*a*) precipitation in rock openings; (*b*) meta-
(*c*) contact metamorphic action of intrusive
in combination. The nature of the dominant
ps: cavity fillings, replacements, and contact
ification of the group to which a deposit be-
its exploration.

its of any origin may be altered by metamor-
led and the deposits correctly classified. Ex-
a for recognition of origin; in which case none
nowledge of origin is useful in prospecting, ex-
its are fortunately rare in the U S. But, ore
re not necessarily metamorphosed; they have
of the enclosing rock.

Igneous Deposits

te the whole of an igneous intrusion, as in
veden; more commonly the useful mineral
e igneous mass, and occurs in disconnected
its center. RECOGNITION. Igneous ore-
als of which igneous rocks are composed;
no other ore types. They never con-
as calcite, rhodochrosite, barite, siderite,
rossularite garnet, epidote, and chlorite
alteration or metamorphism). They are
ich they are a part or the whole, are never

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surrounded by a halo or zone of sericitic or propylitic alteration, which dec as distance from the deposit is gained, and they constitute a small group c each associated with its own rock type. They are distinguished from c metamorphic ore masses by absence of contact metamorphic gangue mi from replacement deposits by absence of non-igneous gangue minerals a dences of intense hydrothermal alteration common with replacement type

Practical applications. Marginal igneous bodies result from different they are usually of irregular form, and only those of large horiz dimensic likely to extend far below the surface. This is more true for masses nearly dimensional in plan than for narrow and roughly tabular masses along the of the parent mass.

Extensions in depth are unlikely to exceed surface dimensions, though exceptio occur. Tonnage possibilities can not safely be predicated on surface showin igneous deposits there is commonly no abrupt wall between ore and parent rock; merge gradually into each other; hence, exploratory workings in the transitional p can not be expected to succeed. Underground exploration for ore masses not reve outcrop are unlikely to be successful; surface exposures are apt to equal or exc size and number of masses found in depth. Exploration should follow the cont tween igneous mass and country rock. In all bodies of igneous ore, primary mine curring near the surface persist to any depth to which the deposit may go, except for c produced by superficial alteration. Any change in tenor with depth is accidenta

Search for orebodies in undeveloped districts. Igneous orebodi not accompanied by extensive mineralization (sericitization, propylitizatio of the country rocks; for example, the granite rocks outside of the Su nickel ore masses are fresh and unmineralized. The prospector has but o dication of probable or possible presence of an orebody of igneous origin; t the existence of a body of igneous rock of the kind usually associated with ticular kind of ore. Such associations are valuable guides; *per contra*, absence throws suspicion on any reputed occurrence. Search for diamon Arkansas was initiated by discovery of the correct type of rock. Co associations are:

Nature of deposit	Parent igneous rock
Diamond	Peridotite
Nickeliferous sulphides	Hypersthene-gabbro or norite
Non-titaniferous magnetite	Syenite, syenite-porphyry
Titaniferous magnetite	Gabbro, norite, peridotite
Chromite	Peridotite (& its metamorphic derivative, serpen
Corundum	Peridotite, nepheline syenite

The rock mass being discovered, search should be made along its conta orebodies (except in case of diamonds, which are scattered through the entire r

Peripheral igneous orebodies bear a definite relation in size to the parent igneous since large orebodies can not be concentrated from small bodies of rock. If size of i igneous body can be determined (allowance being made for small exposures repres large subjacent masses), it affords a clue to size of possible ore masses. Sulphide ig orebodies sometimes concentrate at points where the igneous body makes an ou embayment; hence, such places should be prospected.

Central concentrations are rare and occur only in certain very basic rocks, b possible existence should be recognized.

Syngenetic igneous orebodies occurring as dikes differ in no respect from igneous rocks, except that one or more of the constituent minerals may be of com value. There are no geologic features which will enable one to predict the occurr igneous dikes in a district. This is true also of dikes of ore; they are where you find Dikes are the upward offshoots of igneous bodies, and must lead to the latter. Th way in which a dike of ore (as at Kirunavaara) can terminate is by eventual tra

, though rare, are particularly reliable bodies

l igneous minerals; if non-igneous minerals Included angular fragments of country rock fused with horses in veins, since they are not ned from solution.

limentary Deposits

rs, are formed by sedimentation processes; presentatives in the U S. Such deposits are nerals of which happen to be of commercial cable.

dimentary layers may be confused with re- d by deposition in an opening made by the for distinction: (*a*) sedimentary layers are eplaced beds are local and found only where ntary orebeds have a narrow vertical range have relatively wide vertical range, but re- beds, in their occurrence, are independent ssociated with intrusive rocks. (*d*) In sedi- extensive with the sedimentary layer; with n the layer. (*e*) Beds adjacent to sedimen- replaced beds are usually highly altered by r other criteria see Replacement Deposits, ns is the fact that they usually jump across djoining rocks.

tary ore layers are the most uniform and ne origin being recognized, their extent over a geographic range (subject only to ne area of any given property. Thick- over the entire area, provided estimates d reach of surface alteration.

: Deposits

stricts containing epigenetic orebodies, equently that their associations with ore ital; their recognition aids materially in nce in any given region. They are: (*a*) s commonly, extrusive) rocks; (*b*) com- ; (*c*) extensive hydrothermal alteration lly increase in intensity as the mineral

Most epigenetic metalliferous deposits, ex- bundant intrusive igneous rocks. Of especial ite, phonolites, trachyte, andesite porphyry, yry; to a lesser degree the basic intrusives, regions having no intrusive rocks probably and few in number.

isturbance are typical of most metal- ions favor the circulation of mineralizing r than mineralization, is, like mineraliza- intrusive activity. Besides large faults, mifying cracks and fissures, often accom- g them especially conspicuous; crushed increasing complexity of structural fea- a mineralized district.

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Mineralization. Ore deposits formed by circulating mineral solution accompanied by widespread alteration of country rock. But, deposition c and gangue is not the only expression of the action of mineralizing solu Examples of notable effects of some common types of alteration are as follc

Contact metamorphic alteration. Mineralizing solutions near contact metamc deposits produce contact minerals, decolorization of red rocks, and recrystallizati limestones, either as marble or lime silicates (see Contact metamorphic minerals, b Contact effects often extend to great distances from actual contacts. Other ty alteration may be superposed on them.

Sericitic and propylitic alteration of igneous rocks. Igneous rocks, poor in silicates, subjected to action of mineralizing solutions, change into an aggregate o flakes of mica, called sericite, often showing a little secondary quartz, and sometimes c which can be detected by feeble effervescence with acids. Sericite at times may b taken for white clay; it gives to the rock a dense white to grayish silky appearance; process has been extensive, the rock is usually soft and crumbles on exposure to air. citization is common in lighter colored igneous rocks; if intense, the rock often cor disseminated pyrite, which on weathering yields stained outcrops. Sulphates form surface alteration of pyrite change sericite into kaolin, the rock losing its silky lust

The first effect of mineralizing solutions on the darker, more basic igneous rocks change the darker ferromagnesian silicates (as hornblende and pyroxene) to a gre chlorite and to destroy sharpness of the component minerals. This is called prop alteration; further alteration results in sericitization.

Localization of ore deposits. In nearly all important mining distric the U S, ore deposits constitute a very small geographic unit, in the heart region of intense mineralization extending over a territory usually greate many miles than the area within which the ores occur.

Leadville, the Comstock Lode, Bingham, Park City, Coeur d'Alene, Butte, Cr Creek, Ely, Chino, Miami, Globe, Clifton, Cananea, and many others, are regions c tense mineralization, which extends beyond the limits of ore deposition. Hence, favo conditions for occurrence in a given locality may often be justly inferred from exte alteration. Where mineralization is slight, or confined to the immediate neighborhoc individual deposits, and has an inappreciable geologic range, orebodies are unlikely t large or numerous. There are few exceptions to this rule. The practical value of t phenomena is illustrated by the fact that the old-fashioned prospector, with years of p tical experience, always recognized them, though unconsciously. Deposits like th and Zn ores of the Mississippi Valley are exceptions. They occur in a region wit intrusive igneous rocks, structural deformation, or heavy thermal alteration. Possib of such occurrences in regions of undisturbed limestones should be borne in mind.

e. Fissure Veins

Types. (a) Simple cracks filled by deposition of mineral from solution; sheeted zones, consisting of parallel cracks, similarly filled with ore. In b types additional ore may be formed either by replacement of the walls or of included plates of country rock, with all intermediate gradations. The ore sulting from replacement of the walls may constitute the bulk of a depo Possibilities of large orebodies are greater in replacement veins than in th formed by simple cavity filling; hence, the prompt recognition of this char teristic is desirable. Fissures are almost always surfaces of complex curvatu The strike is never a straight line for any length; it changes sometimes by ger curves, sometimes abruptly.

Practical considerations. Prospecting, development, and exploration are often ba on assumption that fissures continue considerable distances along the strike observed one point. This leads to errors in locating crosscuts, shafts, or adits to intersect the ve and in locating claims with respect to the vein apex. Knowing the average strike duces risk in predicting extensions from a single exposure, but trenching should alw precede selection of sites for openings. Aver strike may be calculated from seve exposed points (Art 6). Reliable information can also be secured from aver trend of ol

are observable; this is particularly true of elow). For deflection of outcrops by topo-

ir size or continuity.

ns shows that most of them dip at relatively the North Star vein, Grass Valley, Cal (dip, d, that veins of low dip are never wide, be- close up the opening. Dip of a vein may t isolated points are unsafe, unless they are nembers of the same fissure system.

Largest ement Fig 10. Single Filled Fissure

e fissures are not so common as those form a network of cracks, or a series of tes of country rock. A group of par- ned a SHEETED ZONE. The degree to d determines size and value of orebody.

Fig 12. Large Filled Fissure, with Parallel Lodes

fissures are often valuable; they are oftener to the assumption that a well defined vein Fig 13). Cross- l, especially in ration revealed ird mine, Colo. ous; their habit rosscuts.

nation. Since nge in charac- nother. Such st and richest ted for initial t outcrop, the l. No atten- icant outcrops

Fig 13. Two Parallel Lodes Overlooked through Insufficiency of Lateral Exploration

ty at a small angle of incidence may be de- distance and then enter it (Fig 15), or may

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enter directly (Fig 16). The larger the angle of incidence, the smaller the probable c in physical character (Fig 17). A fissure may divide into a number of sprays, enc enough waste to destroy mining profit, and terminate as in Fig 18; it may pinch out (F

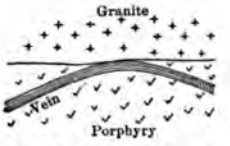

Fig 14. Fissure Deflected by New Formation, Due to Small Angle of Incidence

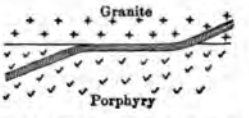

Fig 15. Fissure, First Deflected, then Entering New Formation

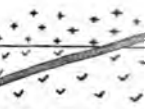

Fig 16. Fissure at Angle of Inci may Enter thus out Change

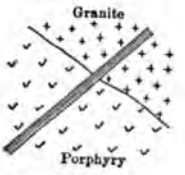

Fig 17. Fissure at High Angle of Incidence often Enters without much Change

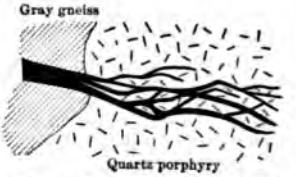

Fig 18. Change in Gottlob Morgengang, Freiberg, on Passing from Gray Gneiss into Quartz-porphyry (after Beck)

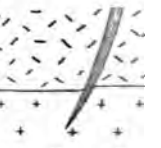

Fig 19. Fissure Pin Out on Entering Formation

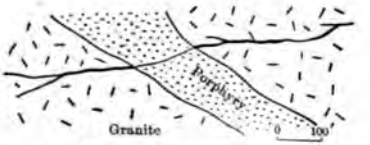

Fig 20. Seven-thirty Vein, Georgetown, Col, Pinching on Passing from Granite into Porphyry (Bull 260, U S G S)

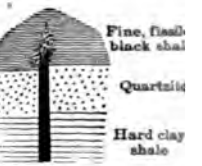

Fig 21. Large Fissure En Abruptly on Passing Easily Distorted Shale, O Col (Bull 260, U S G S)

continue as a mere crack, opening again if the unfavorable formation is small (Fig or fray out or disappear at a contact (Fig 21). A fissure entering fissile rock at a s angle often crosses in a numbe lenticular cracks *en echelon* (Fig 2

Fig 22. Vein Dividing into Lenses *en echelon* on Passing at Small Angle through Cleavable Formation

Practical considerations. plat showing fissures in relat to size and position of differ formations suggests deduction to probable changes due to ab conditions. Character of r also has an effect. Fissures en ing more brittle, from less brit rocks generally divide into stringers; those entering tough rocks or soft sh usually terminate. Outcrops of veins restricted to one of several equally w exposed formations indicate that the latter are unfavorable. Veins can not

fissures, in absence of renewed move-
ions apply also to extensions of fissures
ct often indicates the location of prob-
rence.

within a single homogeneous rock mass,
ning to a mere crack.

ossibilities of pinching are (Fig 23): (a) It
. If previous pinches and swells have been
ein
th,
(c)
f a
ys
he
ral
on

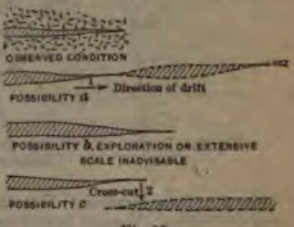

Fig 23

ng
no
ce
ed

gth, though they may not contain ore
h rule; its accuracy depends largely

roots of the original veins; length of vein
Contrast in enrichment may furnish a clue
ps, or middle of veins in any district. The
tated form, the values which existed in its
ater the amount of material stored in these
rease in tenor of primary ores with depth is
ntrast in value of secondary and primary
d the smaller will be the remaining depth of
Much can be learned also from geologic
per than other veins of a district.

mineralization. Amount of displace-
nly a few inches, or at most a few feet,
great displacements occur in fissure-
-mineral and rarely contain ores; this
of great reliability. The search for
fault of large throw is apt to fail; but
r shoots is justified along faults of
ll throw, even when the point of first
ion shows no ore.

e systems. Except in case of a few
s, like the Comstock Lode, fissure veins
ccur alone. Strains which produce
of small displacement generally find
a single one. Fissures having approx
system. Often there are a number of
rict, which may have been formed at
E FISSURES are those which are con-
use; they may belong to the same or
res of the same system are usually of
gnate group are mineralized alike.

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ture of the original rock preserved in ore; (*b*) unsupported nuclei of original rounded by ore which could not have remained suspended in an open cavity; (*c*) of completely developed crystals at the edges of the ore, intersecting the structu original rocks.

Characteristics affecting exploration of and search for repla bodies. Their form and size, which can not often be predicted, are co by: (*a*) homogeneity of rock; (*b*) form of susceptible rock mass; (*c*) num distribution of channels of access; (*d*) length of time and manner in w process operates.

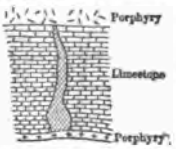

Fig 32. Oreshoot in Limestone, Leadville, Col (Greatest Extension across Bed, because of Dominant Control of Fissured Area). Cross-sec. After A. A. Blow.

Fig 33. Irregular Replacement Orebody stone (Showing Tendency toward Gre tent Parallel to Bedding Planes). Lo After A. A. Blow

Form of replacement masses in homogeneous rocks of simple mineralogy, like lim is usually determined by arrangement and distribution of minute fissures. In ne

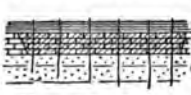

Fig 34. Narrow Bed Between Unfavorable Formations (Replaced Along Fissure Zone so as to give Deposit a Bedded Form). Cross-sec

Fig 35. Massive Replacement of S Clay-slate and Graywacke, Ri Spain (Form Determined by E Zone of Shattering). Plan

geneous formations, like sediments, narrow layers of susceptible rock result in fla conformable to strata above and below, but limited in distribution to the vicinity nels of access, usually fissures. Numerous spaced fissures result in wide, bedded shoots. I replacement of a rock may result in a numbe connected centers, which, if the process co coalesce to form a solid orebody. The bound such deposits fade out gradually where the favorable, though some replacement masses hav boundaries. Fig 32 to 35 show typical forms full discussion, see "Types of Ore Deposits,"

Fig 36. Plan of Flat Shoots in Thin Sedimentary Layers (Arrows Show Direction for Crosscuts and Drifts)

Search in sedimentary rocks. Fissu tems crossing a series of sedimentary bed channels of circulation; if some of the be susceptible to replacement, tabular-shape bodies may be formed in them by rising th waters spreading laterally from fractures the under sides of relatively impervious and uble layers. Search should then be ma crosscutting at right angles to trend of the system constituting the channels of access. When other channels are sected, they should be followed along their strike, to determine whethe

(Fig 36, 37 show conditions in Black
rative purposes.)

Fig 37. Section on *AB* (Fig 36), Showing Position of Favorable Bed, and Occurrence of Ore in it along Fissures; also Position of Drift Beneath Cap Rock

dies in igneous rocks are confined to, lim-
the fractured area in which they occur.

ross-sec of the ore mass below; this is true in
leposit. Replacements in sedimentary rocks
to the bedding planes. Since these planes
ected only at isolated points and inadequately
equently the latter do not outcrop; at Lead-
y where the edges of the including limestone
s, where beds favorable to replacement exist,
rge orebodies and therefore justify search in
e followed down to the favorable bed, where
ch in sedimentary rocks.) When sediments
s are apt to have an adequate surface expres-
to reveal large masses of ore.

amorphic Orebodies

of igneous rock into limestones, or rocks
eculiar group of gangue and ore minerals.
one another, their occurrence in highly
ation with intrusive igneous masses, are
ognition, of contact deposits.

ts (Common minerals marked with asterisk)

LIZATION AND PARTLY ACCESSIONS

ahnite	Staurolite	Paigite
dularia	Quartz	Topaz
lbite	Siderite	Apatite
iotite	*Ilvaite	"Skarn"
alcite	Graphite	Pyroxene
luorite	Ludwigite	Garnet
oisite	Hulsite	Epidote
ndalusite		

S FROM INTRUSIVE ROCK

Iolybdenite	Cassiterite	Tetradymite
rsenopyrite	Willemite	Altaite
alena		

OCK MINERALS

Calcite Dolomite

tallization of the original rock minerals, by
d by combinations of the two. Ore minerals
rals with lime silicates, or of sulphides with

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magnetite and specularite, indicate contact metamorphic origin. Specularite, component flakes set at all angles,, is also characteristic. Following types of contact metamorphic deposits are common; they are listed in order of relative ab

Group	Most abundant ore minerals
Iron group	Magnetite
Copper group	Chalcopyrite, bornite, pyrite, pyrrhotite, zinc blende denite, magnetite, specularite
Lead-zinc group	Galena and zinc blende
Gold type	Arsenopyrite, gold

Practical considerations. Inferences from outcrops as to form and size of metamorphic deposits are absolutely unreliable; they are the most irregular o bodies. Irregularities in tenor are also characterist grade of ore is often low. Hence, much exploratic quired and predictions of extension far beyond expo are unsafe. These are typically marginal deposits (and search should be confined to the borders of the i Contact deposits are not found associated with extrusi as rhyolite, andesite, and basalt.

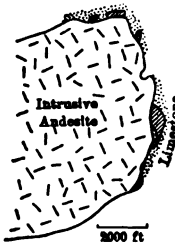

Fig 38. Contact Metamorphic Deposits of Magnetite in Limestone, Surrounding Intrusive Mass of Andesite, Iron Springs, Utah (Shading = Ore; Stippling = Contact Minerals). After Leith

h. Alteration and Secondary Enrichment

Superficial alteration of ore deposits. Ex cold, recently glaciated regions, ore deposits at the surface undergo changes by superficial a which enrich their upper portions, or impoveri upper portions and enrich the lower. The proces SUPERFICIAL ALTERATION and SECONDARY ENRICH both performed by cold, descending waters, ca matter in solution derived from oxidation of the part of the deposit. Outcrops must be interpre the light of these changes; if the ore in shallow s workings is primary, no abrupt change may be exp for primary ores are subject in depth only to fluctu characteristic of their original formation. Seco ore may lead in depth to unworkable primary material, as at Ray, Ariz rich primary ores, as at Bisbee, Ariz. A knowledge of superficial chan criteria for recognition of secondary ores, and of the probable characteris orebodies below oxidized and enriched zones, is an invaluable aid in prospe exploration and development.

Alteration and enrichment processes. When surface water cont CO_2, O, etc, acts on an ore deposit, certain minerals are oxidized and with form solvents which dissolve metals (as Ag, Au and Cu), forming dilute sol which trickle downward. On reaching a point where O is no longer ava as at the ground-water level, their metallic content is, in favorable circumst deposited as secondary sulphide. The metals dissolved may be added to below, thereby producing secondary enrichment. As erosion progresse process is repeated; and both primary and secondary sulphides are diss and reprecipitated. A very considerable enrichment may result. In the mine, Ariz, the unenriched material contains about 0.25% Cu, the enri about 2%. The metallic contents of descending solutions may be depo before reaching the point where O is no longer available; in this case, seco oxidized ores are formed. The part of the deposit in which oxidation and tion goes on is the OXIDIZED ZONE; that where secondary sulphides are depo the SECONDARY SULPHIDE OR SULPHIDE ENRICHMENT ZONE; the unchanged of the deposit below, the PRIMARY ZONE.

ized zone. Deposits especially sus-
containing sulphides, arsenides, anti-
which is generally present, greatly
, which in presence of O attacks most
metals.

$Fe_2(SO_4)_3$; copper sulphides yield $CuSO_4$;
als, Ag_2SO_4. In presence of Mn and alka-
By these processes the upper part of the
ls, and is impoverished in their respective
iron sulphides form limonite, which re-
sty iron-stained cap (gossan).

lie beneath. Most gossans are iron-
sulphides in ores. Other stains may
bonates, silicates, and basic sulphate
annabergite) that results from oxida-
alt bloom" (erythrite), where cobalt
O_2 in manganiferous ores. Discovery

l size of orebody. Gossans comprise:
which metallic minerals make up a con-
sulting from disseminated or "porphyry"-
all part of the deposit. The former give
d. Size of gossan and amount of limonite
osit beneath. The gossan may be much
over of iron oxide from the point where
ide is apt to be more pronounced in rocks
ipitation of limonite, retards migration.

nally the same as the form of mineral body
r simple forms of massive gossans. But
shape produced by the intersection of the
the gossan is not necessarily that of the
zed zone has considerable vertical extent,
e original deposit may also have varied.
y be inferred, for the sheeting of the origi-

osits, as those at Miami, Ray, Bingham,
s are disseminated in small cavities origi-
cracks and seams of the rocks, but do not
The appearance of massive gossan is thus
unding fissures be broken and examined
type of iron stain, together with numer-
acteristic of disseminated orebodies.

an composed of cellular quartz and stained
The cellular spaces give some idea of the
of iron stain be incommensurate with
rt of the total mineral removed was more
, Mn, Co, etc. indicate that these min-
icate that minerals of similar crystal shape
ossans are in most places original minerals,
Fe_2O_3, limonite, complex silicates and light
idation of garnets, may indicate a contact
s overlying disseminated copper deposits,
of the Cu was dissipated near the surface.

e to redeposition from solution, or to
soluble minerals into solution and
deposit contains 0.1 oz Au per ton, and

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half of the ton is removed by solutions which do not dissolve the Au, the ore then contains 0.2 oz Au per ton.

Precipitation of minerals in oxidized zone may take place by; (*a*) Satur evaporation, by which most efflorescences are formed, as chalcanthite, goslarite portant deposits of brochantite. (*b*) Oxidation and hydration. The oxidatio gives rise to deposits of $PbSO_4$, which, being insoluble, remains in the oxidi in certain circumstances, $PbSO_4$ is transformed into $PbCO_3$. Similarly, forms deposits of limonite from iron sulphides; CuO and native Cu from erals; Ag and Au may be released from sulphides and left or precipitated in (*c*) Chloridization of Ag minerals in a deposit has formed important orebodies (*d*) Reaction with wall rock. Some of the metals in sulphate solutions may b tated by contact with limestone or other carbonates; for example, the carbonate ores of Bisbee and Leadville. If the solutions come in contact with SiO_2, metal deposited as silicates; for example, the copper silicates at Globe, Ariz.

Depth of oxidation may vary from a few inches to over 2 000 ft. depends on many variables, a few generalized rules may be of value.

In warm, humid countries of low relief and high water-level, a shallow oxid may be expected. In similar regions of high relief and low water-level, a deep zone may be expected. In widely glaciated regions, or where subjected to glaciation, the oxidized zone will be lacking or only a few feet in depth. In cold it is apt to be shallow; if oxidation took place in pre-glacial time, and the regio been extensively glaciated, the zone of oxidation may be deep, as at Kennecott where it attains a depth of 1 200 ft. In arid regions, with high water-level, it is shallow. In arid regions, with low water-level, it will probably be deep. In ari with no water-level, it may be deep or shallow, dependent upon how quickly the O was used up by oxidation in upper part of the deposit.

General inferences from oxidized zones. Most ore deposits have idized zone; it may be barren capping, or may contain oxidized ores; if the certain inferences may be drawn as to the possibility of ore beneath. If zone contains ores, these probably have been formed by superficial chan a change not only in metallic content, but also in metallurgical char to be expected below. This change is of vital importance in the pro life of the mine, and in the mode of ore treatment.

Precipitation as secondary sulphides. Sulphate solutions which lose their metallic content in the oxide zone trickle downward until a reached, usually at the ground-water level, where no more O is availal this reducing environment, the metals are deposited as secondary su According to L. C. Graton, secondary copper sulphides are deposited invariably by the replacement of other sulphides (16).

Replacement of other sulphides depends upon the relative solubilities; a sul lower solubility will replace one of higher, or, since FeS_2 is relatively more solu CuS, the CuS will replace FeS_2. Relative solubilities of the common sulphides in ing order are: HgS, Bi_2S_3, Ag_2S, CuS, PbS, NiS, CoS, FeS_2, ZnS, and MnS. He would not expect to find zones of secondary MnS or ZnS, because of their relativ solubilities, but would expect zones of secondary CuS and Ag_2S, because they tively less soluble than many common sulphides. Secondary CuS and Ag_2S are commonly encountered; if there is a gossan with Cu stains, an enriched zone of se CuS may be expected, if erosional and climatic conditions are favorable.

Inferences as to completeness of enrichment. This depends up

(a) **Rate of oxidation.** The rate at which material is released from the oxidi depends chiefly upon rate of erosion, length of time involved, climatic conditio meability of the rock, and topography. RATE OF EROSION. Ores are first brough reach of oxidation by erosion of overlying rocks. If erosion removes the top of the faster than it can be oxidized, no oxidation, and consequently no enrichment, res oxidation proceeds faster than erosion, oxidation will be halted by the ground-wat because the latter conforms approx with the topography and is depressed at sa

le water-level, the vertical range of a deposit ex-
on is rapid; hence, the amount of metal released
ss. Condition most favorable for high degree of
d of erosion are about equal; this causes fairly
e deposit above water-level; oxidation and ero-
the enrichment may be repeated several times.
osed to oxidation determines the degree of en-
If it can be determined that the deposit has
recent geologic time, the expectation of a well
NDITIONS affect rate of oxidation (see Depth of
OF COUNTRY ROCK. The more permeable the
nrichment. This is important in disseminated
is rocks, they undergo much more thorough en-
er relatively impervious rock. Ore deposits in
for the limestone reacts with the sulphate solu-
later deposition in the form of sulphides. At
s have found that the ore in porphyry is more
e. In a deposit with impervious enclosing walls,
eposit itself, and greater opportunity is afforded
ides. If the walls are shattered and pervious,
metallic content lost. Faults having impervious
n a disseminated deposit may act as a dam, and
hanging wall and thereby bring about deep and
RAPHY. The zone of enrichment usually con-
ted at the time of enrichment. In many cases,
eys faster than oxidation and enrichment could
od of finding enriched orebodies beneath the
d under the uneroded remnants of older topog-
present topography, a well enriched orebody is
es than under steeper ones; the latter receive
greater opportunity for dissipation of oxidized

olvents. Since secondary sulphide enrichment
is clear that if no minerals are present which,
sulphate, the metals can not be transported to
secondary sulphides. Iron sulphides yield the
e, little enrichment is likely. Presence of FeS_2,
ial to sulphide enrichment, so far as yet known.

pper enrichment, secondary sulphides of Cu are
des. If the solutions formed through oxidation
me in contact with sulphides below the reach of
e enclosing rock must be such as to permit the
her than limestones permit this.

darily enriched deposits. For intelligent
posit, it is necessary to recognize secondary
what part of the zone of enrichment is re-
ore valuable ore may exist above or below
ined whether the enriched ore will be suc-
ore.

n implies presence of a mineral body, which may
mineral, as Cu minerals, be present in the oxide
expected. A very distinctive feature of a sec-
nal character. The oxidized zone is often value-
one; still lower, the less valuable primary mate-
suggests secondary enrichment, but it must be
en found in mineral deposits. Secondary zoning
t topography, and an abrupt mineralogic change
ly has no relation to topography and the miner-
ing is also often nearly horiz or shows evidence

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of tilting from an original horiz position, which is not true of primary zoning enrichment has taken place, the enclosing rock is usually kaolinized by the soluti produced enrichment (17); conversely, this phenomenon may indicate enrichme

Table 4. Common Indicative Ore Minerals

Metal	Minerals usually of primary origin	Minerals usually of secondary (sulphide) enrichment origin	Minerals usually originating in oxidized zone
Copper	Chalcopyrite Bornite *Enargite *Tetrahedrite *Tennantite	Chalcocite Covellite	Native copper *Malachite *Azurite *Brochantite *Atacamite *Chrysocolla *Cuprite *Tenorite
Silver	*Tetrahedrite *Tennantite	Native silver Argentite Pyrargyrite Proustite Stephanite Polybasite Pearceite	*Cerargyrite *Embolite *Bromyrite
Gold	Native gold Gold tellurides	Native gold	Native gold
Zinc	*Sphalerite Willemite	Wurtzite	*Smithsonite *Calamine *Hydrozincite
Lead	Galena		*Cerussite *Anglesite *Pyromorphite Leadhillite
Iron	Pyrite Marcasite *Pyrrhotite Arsenopyrite Magnetite Hematite Specularite Siderite	Marcasite	*Limonite *Iron sulphates

* Invariably primary, secondary, or oxidized, according to column in which name of mineral appears.

Mineralogic Secondary sulp posited in the enrichment are s distinctive and of secondary en (16). Kaolinite gibbsite and gyp accompany secon phides. In Cu a sure indicatio ondary enric when chalcocite lite form a soft, bluish black c "sooty chalcoc other sulphides may be a thin may in large place the origi phide. Upon such a specimen of original sul revealed. The sulphides may seamed by veinl secondary sulphi ting indiscri across two or mo original sulphides

Other criteri riched Cu depos be recognized shape of curves by plotting the from drill-hole against depths (t curves of enrich show a peak on t part; the mo nounced the en the more peaked curve. Primar yield curves of fairly even profile, and may thus be distinguished from second (Fig 39).

Character of primary minerals. If the mine workings are in the enrichment, the nature and value of the ore that will exist beneath may ferred by determining the kind and abundance of the original sulphide r by the secondary ores. For this the microscope is necessary. If dete that the secondary sulphides replace lean FeS_2, then lean pyritic material expected beneath the secondary zone; if they replace galena, bornite, and pyrite, then valuable primary ores may be expected.

6. Systematic Surface Exploration

Value of thorough surface exploration is apt to be underestimate furnishes information which either prohibits or allows intelligent plann

is an essential preliminary to underground
ION prevents overlooking important expo-
ation
nings
.

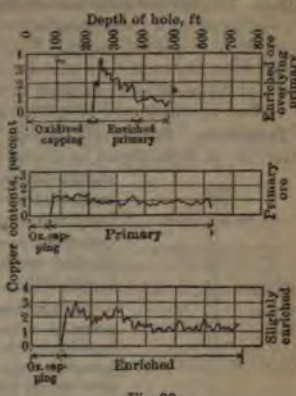

Fig 39

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soil
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hing,
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f the

e data respecting size, shape, and value of large
t are feasible to depths of 30 to 50 ft or more.
enings; when present, exploration must be by
MOUNT OF WEATHERING. Value of results from
d reliability of information afforded by outcrops.
pace with weathering, an outcrop is a reliable
re negative, it is unwise to attempt underground
tirely or partly removed by leaching from the
rface exploration may still determine its size,
lone may decide whether further work is justi-
hile outcrop samples from leached orebodies do
ay determine the location and size of shoots.
haracter of mineralization in depth. Weather-
t it is practically impossible to trace outcrops
ormation from them; underground work is then

dy; judgment determines extent and detail
map showing property lines, topography,
pits, trenches, underground work, assays,
er obtainable details (Sec 19).

than 100 ft to 1 in; for small areas, 40 or 50 ft
l, plane table or transit are used in mapping, de-
ane table is convenient and rapid. Numbered
or 200-ft squares, as a means of locating subse-
probable extensions are plotted on the map and
outcrops). Probable course of faults, contacts,
me way. Such a map shows the relation be-
tcrops, geology, and topography; outlines drift-
s needed; aids in eliminating unfavorable areas

t are labeled and their position noted on
assayed or panned). By showing location
eology, and probable outcrop extensions,
to uncover outcrops is often reduced.

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Tracing by panning (Sec 31, Art 5) may be used in a similar way. panful of dirt should be taken each time, so that results will be roughly tative. The mineral obtained is weighed and kept in numbered bottles, ber and wt of each trace being entered on the map. Blanks should be as carefully as good showings. Having no exposures to guide the work, can be done systematically at corners of squares. Panning along expo crops, or below probable location of drift-covered outcrops, is the cheap of finding oreshoots and determining their length.

Trenching. For a single orebody a series of trenches is usually dug at intervals, at right angles to course of outcrop. Probable extension of should be staked on the ground as a preliminary tance between trenches is determined by type body and depth of soil; they should be close en determine aver width and value of outcrops and missing oreshoots. Trenches over the latter ar together than on barren areas.

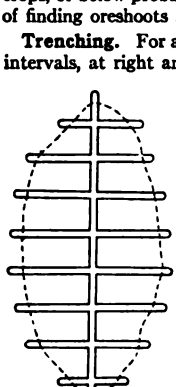

Fig 40

The first trenches should be as far apart as possi generally reduces number, length, and cost of inte trenches. It is more satisfactory to follow narrow out trenching along them. In shallow soil not more than 2 deep soil not more than 4 men, can be employed to adva trenching both ways from a single exposure (19). Fig 4 general arrangement of trenches on outcrops of large orebodies, planned to avoid high cost of complete st Where large drift-covered areas are explored for su mineral, trenches are usually dug along the sides of 50 to 200 ft apart. First work consists of 1 or 2 long t which give information as to geology, depth of soil, e generally divides area into favorable and unfavorable Good showings are explored first. If there are no surface indications, systematic started where the soil is shallowest; deep trenching is usually done last (see Cobalt,

Test pits for systematic surface exploration are spaced on corners of squares. Principles involved in their location are same as for boreholes (Art 8).

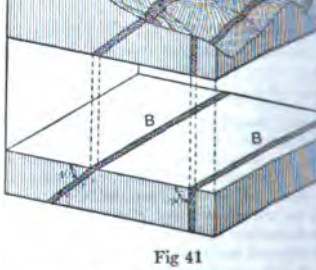

Fig 41

Summary of results. All exposures in exploratory openings are measured and sampled; widths, assays, and any further geological data obtained should be entered on the map, and geological cross-sections made where necessary to interpret structure. Results of shallow exploration may often be summarized by computing from area of orebody exposed the tons of ore per ft of depth. Depth to which the known orebodies must continue to repay estimated cost of development and equipment and yield a profit can then be calculated. This figure, compared local geology and type of orebody, furnishes a measure of the risk involved

Migration of outcrops. With unwarped veins, beds, faults, and cont dipping 90°, the outcrop is in a straight line (the strike), regardless of topogr (Fig 41, *A*). Outcrop of such deposits on flat topography is also a straight

n rough topography, the outcrop of a vein
oked; its deviation from the strike line in-
). Width of outcrop of veins of uniform
phy, as indicated.

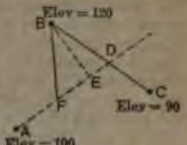

Fig 42

re made by assum-
they are always
ssumption is useful
pprox location of
res can be laid out
h approx locations
l as with bends in
late observed data
uncover outcrops.
iciently accurate for this work, and apply to
.

easured with a compass at some point, or
g 42). Plot the 3 points *A*, *B*, and *C* in
their correct relative position, and note
evations. Connect high and low points
BC. Take point *D* on line *BC* so that
B = (elev *A* − elev *C*) : (elev *B* − elev
ne *AD* is the strike line of plane *ABC*,
bearing is measured with protractor
nown bearing *AC*. Draw *BE* perpen-
to *AD*. Lay off *EF* to scale and equal to
ce in elevation between *B* and *A*. Angle
the dip of plane *ABC*.

ing outcrop on a contour map from a
xposure (Fig 43).

he outcrop of a vein or bed is exposed at point
wn. The underground contours of the vein
ght lines, *CD*, *EF*, etc, parallel to strike *AB*.

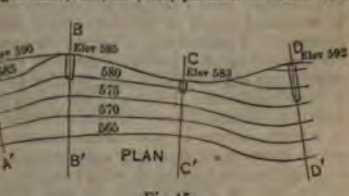

Fig 45

rs of equal elevation mark points on the out-
presents its probable location. Same methods
crops or work at several points. Fig 45 shows
tructure) contour map of a vein, aver dip 45°,
at points *A*, *B*, *C*, *D*. Assume that dip and
Draw lines *AA'*, *BB'*, *CC'*, *DD'*, perpendicu-
nd on them, by means of dip triangles (Fig 44),
sected by successive structure contours. Then

e their dip and strike gradually; hence,
gentle curves. If abrupt changes in direc-
of equal elevation, look for faults or other
luable in planning underground exploration;

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when superposed on surface topography, they give close indications of
tion of outcrops. For further detail of graphic methods, see **Bib** (21,

Staking probable location of outcrop on the ground is done in different
Plot probable outcrop on a contour map, and lay out this line on ground by orc
veying methods. (*b*) Set up a transit over a known point on outcrop, and pl
at regular intervals on strike line. Set transit on each stake, and cross-section t
crop, as for grade stakes in railroad work. This method involves much calculat
can be reduced by using slope-stake diagrams. (*c*) Use transit with Shattuck sol
ment (23); set the attachment so that when it is revolved the deflected line of c
will generate a plane at right angles to the telescope. Transit is set up over
outcrop, main telescope oriented in a line perpendicular to strike and the telesco
down until line of sight is perpendicular to the dip (vert angle = complement of
Deflected line of collimation then revolves in a plane parallel to and slightly to c
plane of vein. If the ground is sighted, each stake will be slightly off the co
Error is slight if the outcrop is traced from one set-up; where many stations are
error is cumulative. (*d*) Transit and top telescope. Loosen capstan screw, so
top telescope will revolve in a plane parallel to main telescope. Set transit over
outcrop, orient main telescope in a line at right angles to strike, and turn main
down until its line of sight is parallel to the dip (vert angle = dip angle). The
scope then revolves in a plane parallel to plane of vein. A rod will give true
outcrop, if sighted on at an elevation = height of instrument + ($c \div \cos$ dip
vein); c, a constant for any instrument = distance between center lines of top
telescopes. (*e*) Brunton compass, if set on a light tripod with a ball-and-socket m
is accurate enough for tracing outcrops short distances. The spindle of the in
is tilted to a position at right angles to dip; then sights will rotate in plane of

7. Surface Prospecting and Exploration. Examples of Practi

Nova Scotia (24). Systematic prospecting is done here for gold
veins in regions covered by deep glacial drift. Prospecting is confined
tively small areas, in which rich float occurs. Surface is fairly level and o
conditions causing float to move downhill are absent. The general direc
glacial movement was a little east of south.

Prospecting methods are based on recognition of fact that while glacial drift c
rocks brought from a distance and entirely different from the underlying bedrock,
be composed largely of material of local origin. Sequence of operations is as
MAPPING. A contour map is made of the area on a scale of at least 40 ft to 1 in.
are recorded details of geological structure, drainage lines, course of glacial transpo
distribution of gold-bearing float, position and depth of existing pits. A compass
is usually sufficient. PANNING. About 200 pans are taken from old dumps, surfa
beds of brooks and sides of old trenches or pits. An exact record is kept, showing l
of samples, number of pans taken, character and composition of drift (which ma
clue to its source), and number and size of colors in each pan. Results are recorded
map and summarized by drawing two lines, one enclosing the area in which floa
occurs, and the other limiting the area containing shotty gold and large colors.
inner line incloses the center of gold distribution; within it and near its northern ext
most of the prospecting work will be done. Panning lessens the amount of heav
liminary work. It usually confirms inferences drawn from the distribution of flo
prevents work in barren areas. Two men, panner and helper, usually do this wor
or 5 days. TEST PITS are then sunk. A notebook is kept showing each pit in cr
on a large scale. Following details are noted: (*a*) depth, thickness, slope, mode
mation and composition of each layer in the drift. Each layer is given a numb
future reference; (*b*) results from panning each layer; (*c*) absence or presence of
quantity and estimated value per ton, number of pieces, size, degree of wear, and cha
of each variety and its associated minerals, as an aid to identification; (*d*) details of
rock, dip, strike, cleavage, results and direction of glaciation; (*e*) depth and amou
water encountered; (*f*) size of pit and number of buckets hoisted; (*g*) labor, time,
material for sinking and timbering. Items *e*, *f*, *g* are useful for estimating future
All pieces of gold quartz are kept for comparison with float from other shafts; they
numbered and put in boxes. CROSS-SECTIONS show no orderly arrangement of st

ber; 2 different layers may contain float from to different periods of erosion; often transporta- rections at different times. But such correlation mparison of their contents, give much informa- t gold. Location of pits. A test pit should limit of area of richest pannings or float. Pits ost movement. If all the gold is in upper layers pits can be placed at some distance apart; if all rift, the vein is probably not far away. If gold ey can be saved by sinking only a few pits below ght end of a row show no gold, the next row is gold is distributed from an oreshoot in a more to be prospected decreases as the source of the

anged drift, rendering above methods fruitless s can be interpreted. Fig 46 shows such a case pit were: (*a*) coarse mixed debris, much water- ng through it from west, evidently the bed of a t; (*b*) un- (*c*) strat- ttle gold- stratified clayey in ining pine tum being ied drift; . Under have con- e to char- prospect- ons were: er one to he site of ly to the

Fig 46

interpreted as follows: A brook ran across an ed (*a*) and undermining hill to the south. The ave partly dammed stream, which then formed . Another period of erosion resulted in another This completely stopped the stream, making a in which silt and humus of bed (*f*) collected. t have crumpled layer (*f*) and formed bed (*g*). n hills on both sides. (For equipment used for

of native Ag, associated with Co and Ni onglomerate, and greenstone; some profit- ess. First discoveries were made by exami- oxidize to pink erythrite or "cobalt bloom." athering, but is a good indicator when the nly broken. All small crevices are carefully e yields a soft black mud, CoO, containing ed arsenates of Co and Ni are found a few eins also considered a favorable indication.

by shallow glacial drift, in which prospecting is ate lines are laid out on the surface at intervals hese; in deep soil, trenches may be farther apart iminary trenches are dug before thorough pros- p are recorded geology, rock outcrops, known ep drift, etc. Trenches are just wide enough for ning the bedrock, they must be 2 to 3 ft wide at

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bottom; width at the top depends on depth and character of soil. When dept 6 ft, trenches are timbered by an occasional stull with uprights or head boards dry soil, bedrock in trenches is carefully cleaned with brushes; in clayey soil, by Use of water is avoided where possible, as it conceals crevices by filling them w Veins are difficult to recognize, especially in diabase; every likely place is bla Each vein found is stripped as far as it can be followed, and shot out at close inte

Details and costs of work. R. B. Watson, Gen Mgr, Nipissing Mining Co. following data on systematic prospecting by that company. In 1912 an HYDRAUL was installed, to wash all the soi surface, for complete inspection of Water is pumped from Cobalt Lake. of installation: one-stage centrifug (Jens Orten Boring Co), capacity per min against 415 ft head; guarant 78%; Siemens motor, 650 hp, 2 2 1 180 r p m; spiral-riveted, 16-in No 10 gage, with bolted joints to jacent lengths of pipe to be turned angles of 10° to 15° without leakage. Some lengths of pipe have flanges on one bolted joints on other, giving a little stiffer pipe line. Size of nozzle is from 3 to pending on whether a very strong stream is desired, or less force with larger volun good results, pressure at nozzle must be at least 100 lb per sq in. When all groun reach of the nozzle is washed off, usually taking about a day, the pipe line is exte the next set-up, which has been prepared in advance. After an area has been clea examined, surveyed, and then serves as a dump for spoil from adjacent areas. D is used to break rock and scatter heavy boulders. In 1914, the plant ran 79.08 time; 153 set-ups were made; aver time lost in setting up, 1 hr 32 min. Aver nozzle, 138 lb per sq in. 68 309 ft of pipe were shifted, at cost of 17.35¢ per ft for h 18 730 ft of roads were built, costing 7.76¢ per ft. Power cost, 1.3¢ per kw-hr. acres were washed, at $499.47 per acre. Aver depth of soil, 3.4 ft; 529 415 cu moved at aver cost of 9.015¢ per cu yd for labor, power, and general supplies. To per cu yd: labor, 4.99¢; power, 3.2¢; supplies, 0.82¢; explosives, 0.22¢; handlin 2.24¢; roads, 0.27¢; total, 11.74¢.

Table 5. Trenching at Nipissing Mine

Year	No men	Miles of trench	Aver depth, ft	Cost per ft
1909	87	33.1	3.4	$0.152
1910	61	31.7	2.7	0.140
1911	25	13.7	2.7	0.122

In 1914, on La Rose property, the soil was shoveled off an area of several acres, i 20 ft wide; distance between strips was made as small as possible, determined height to which men could throw the dirt. With an aver depth of soil of 1.9 ft, th cost 37.2¢ per cu yd.

Witwatersrand, South Africa. Silicified conglomerates carry Au, and as members of a series of sedimentary rocks. The auriferous "banket" dip at 16 to 90°. Faults complicate the work of locating outcrop extensi places. "The surface is first examined to locate one or more beds of sand with which the reefs are conformable" (25). Topography is closely rela the geology; hard beds form ridges and, in some cases, streams follow fault In shallow soil, indications of the character of underlying rocks are obtaine ant hills; over banket these have generally a yellowish gray color, over dikes they consist of red loam.

Trenches are dug at right angles to the strike of the formation at regular int One trench near middle of property is continuous; the others are made only wh central trench shows that a reef is likely to cross. At each point where reef is fo small shaft is sunk deep enough to provide samples from points unaffected by alteration. In deep soil a small shaft is sunk into solid formation, and a crosscut to give a section of the rocks; or 2 or 3 shafts are sunk and connected by cro Values are remarkably persistent and fairly uniform over large areas of reef; hence, exploration gives reliable indications of conditions at depth.

Clinton hematites occur as stratified beds in the Clinton formation (Si age), all along the Appalachian Range, and also in New York, Kentucky Wisconsin. The beds vary in thickness from a few inches to 10 or 12 ft; the subject to all the irregularities of sedimentary rock beds (Art 5; also Sec 2, Ar

y of float or outcrops, or by recognition of
ith which these ores are associated. Above
y some characteristic stratum which out-
an approx location of the ore-bed outcrop
The outcrop is then uncovered by trenches
soil. In the south, weathering extends to
rosion has kept pace with it and unaltered
).

e, dip, strike, etc, exploratory openings (inclined
ular intervals into unaltered ore. Since these
re apt to be uniform in size and tenor over large
ced far apart and still give good basis for ton-
e should be carefully worked out. Diamond
t outcrop evidence, locate faults, etc (Art 8).

Appalachian valley. The two important
acement deposits.

and boulders of limonite, imbedded in clay and
or dolomite. Orebodies are very irregular and
nuous outcrops occur, especially in gullies, or
carried on in open cuts, and exploration must
ality of ore, and depth and character of over-
down to determine whether surface indications
; data on concentration and quality of ore is
samples. Different methods are used for ex-
able. In Birmingham district, Ala, these ores
e surface is rolling and free from stones. A large
am, are sunk 200 to 300 ft apart, but rarely sys-
urn drills are used (26). Similar methods are
f, Tenn and Texas brown ores.

replacement of some particular bed (Art 5).
l nature of rock replaced; methods of prospect-
al topography and geology (7).

ation by Boring. (See also Sec 9)

tion with prospecting and exploration are:
, rock, swamp, or water; search for exten-
odies; search for parallel orebodies; location
ies, and water-bearing strata; detailed ex-
tonnage and value. LIMITATIONS. BORE-
is not always the cheapest method of pros-
urnish completely or accurately the desired
rk is done by test pits, shafts, or drifts.

ar case is based on consideration of follow-
oring is not well adapted to exploring nar-
d irregular orebodies or high-grade, spotty
small size, may miss such orebodies entirely,
rren, or rich spots. Boring is best suited
m grade, as masses or beds dipping less
r iron ores, the disseminated lead and zinc
per" deposits, Clinton iron ores, and coal
ing is done by augers or churn drills; test
n its limited field (Sec 9, Art 2) an earth
e for less than 0.2 of the cost of a test pit
30 ft in surface drift, free from water and
ts usually cost less per ft than churn-drill

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holes. Pits are better under such conditions for both prospect exploration, because of the more accurate information they afford. rock, small shafts often cost less than either churn-drill or core-drill shallow depths (20 to 50 ft). This is true only where no timberi quired and water is absent. In soft porphyry at Ajo, Ariz, 4 by 6-ft shafts, 50 ft deep, cost less than diamond-drill holes to same depth (2 cost of moving and setting up drills is largely responsible for the high co of shallow boreholes. For hard rock in general, and for greater depth rock or alluvium, the cost of boring per ft is 0.2 to 0.5 that of smal SPEED. Boring has little advantage over test pits for shallow work (20 in dry surface drift requiring no timbering. In rock, and for greater d alluvium, speed of boring may be 3 to 10 times that of shafts or other unde openings. Presence of WATER increases cost and decreases speed of unde openings, but does not interfere with boring. Boring is therefore pe applicable to prospecting and exploration in water-bearing formations POSE OF WORK. Many boreholes are made merely to locate strata or o preliminary to sinking or driving to or through them. Thus, in the Clin ores of Alabama, diamond-drill holes are bored at distances of 1 000 to from the outcrops, to locate the ore bed and anticipate the effect of f location of shafts and other mine workings. Similar cases occur in ma mines. On the Rand, also, the reefs under deep-level properties are loc diamond-drill holes before working shafts are started. SPEED VS COST. last-named cases, speed and cost are paramount factors, samples of t body are secondary objects, and boring has an advantage over shaft which increases with depth of the deposit. At the other extreme is th of detailed exploration to determine tonnage and aver values, in which c speed are subordinated to the necessity for accurate samples. In this c type of orebody determines the choice.

Underground openings always give more accurate information and samples, vantage can be taken of the speed and cheapness of boring in orebodies like the iron ores and the "porphyry coppers," where values are uniform over large areas, gradually from point to point. So much work has been done in some districts, Missouri and in certain placer deposits, that empirical factors have been determ correct inaccuracies due to boring practice and to irregularities of the orebodies

Boring from underground points by diamond drills is done in many in searching for parallel orebodies, or to locate the walls of wide orebodi

At Alaska Treadwell mine in 1913, 4 318 ft of holes were thus drilled to outline of the deposit (28). Rocks are hard diorite, quartz, and greenstone, and soft sl taining quartz stringers. A high core recovery was obtained; 90% in one horiz ho ft long. The information was therefore accurate. Drifting and tunneling wou taken 4 times as long, at a cost of $8 to $12 per ft. Cost of drilling was $1.62 to $ ft. Miami Copper Co did 2 714 ft of vertical diamond drilling from underground in 1912 and 1913 (29). Rock was silicified schist, broken into small pieces by fissu individual pieces being very hard. The holes caved frequently and were difficult and sample; less than 10% of the core was recovered. Cost, $5.44 per ft; speed 5.5 ft per 8-hr shift. These examples show that the advantages of boring are in very difficult formations. Fig 47 shows radiating diamond-drill holes for pros at the old Silver Islet mine, Mich. Such holes explore large areas much more and quickly than is possible with underground workings.

Boring has a special field in prospecting for oil, gas, salines, and sulphur, where th hole is used subsequently to conduct the mineral to the surface. For prospecting swamps, lakes, rivers, or deposits of quicksand, boreholes have no competitors. holes are sunk from solid ground, or vertical holes from boats or through the ice in

In remote districts the cost of transporting boring apparatus is generally prohi except for shallow prospecting where hand drills can be used. It should be note borehole results are susceptible of more accurate interpretation in districts where the

stood; shafts or other openings are therefore
n new districts. Steep or rough topography
f moving drills, and may compel the use of

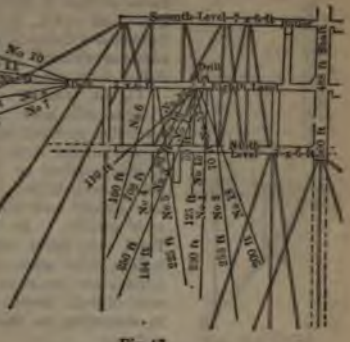
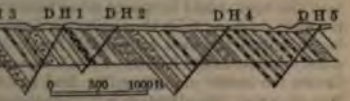


Fig 47

cting for flat massive orebodies. Coordinate
o ft, depending on the type of orebody
there is no adjacent work to indicate the
cation or course of ore, the first hole is sunk
convenient point, say at *A*. In the illustra-
hole would be barren. Assume that the
third holes are drilled at *B* and *C*. They
at the trend of the orebody is not along
or *BC*, and subsequent drilling would be
directions *EF* or *FG*. Instead of drilling
r ore is found at *B*, holes 3 to 10 might be
infinite number of variations occur; as a
close drilling is attempted, it is cheaper to
he general trend of an orebody. In drift-
eas, where orebodies occur as interbedded
f a stratigraphic series, boreholes give a
ss the whole ore-bearing formation. Fig 49
nes perpendicular to the strike of the forma-
pends on the dip of the rock; an attempt is
nished by successive holes will overlap. The



nd-drill Holes at Victoria Mine, Mich (after
che, *Ann Rep* 1909, Mich Geol Surv)

ly obtained if holes in successive rows are
rtical and inclined holes (*CD*, Fig 50) de-
are cheaper where depth of surface drift is
where nothing is known of the dip of rocks
formation is less than 20° to 30°. Inclined
e shorter, there is less danger of misinterpret-
, and they generally cut the short dimension
give trouble in both drilling and sampling.

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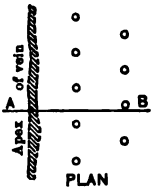
For detailed exploration of flat massive deposits, drill holes are located on c squares. A systematic arrangement is essential, to secure impartial samples a intervals and simplify subsequent calculations of tonn aver values. Interval between holes is a matter of j based on type of orebody. Questions of cost urge that spaced far apart, but accuracy demands that the int limited to a distance over which it is safe to expect the or continue without marked irregularities in form or ten Art 11 for spacing used in different localities.)



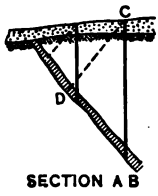


Fig 50

Fig 51, from R. V. Norris, shows one method of diamond-drill holes used in the Penn anthracite fields termining position and thickness of folded coal seams in which holes were drilled is shown by their number 14 was required to interpret data from 12 and 13; s hole 17 was drilled to correlate results from 15 and 1 face geology and outcrops are valuable aids in con such sections. In the Wyoming and Lackawanna Penn, coal seams often outcrop under present or valleys, filled with wash to depths of 50 to 175 ft. borings are made on corners of squares to determi ness of rock cover; size of squares varies from 100 ft, depending on this thickness and on irregularity of surface.

Prospecting for oil. First holes are generally loca reference to geological structure. Underground reserv occur in connection with anticlines, monoclines, synclines, and faulted zones in fields. "The area that can be drained by one well is variously estimated at f

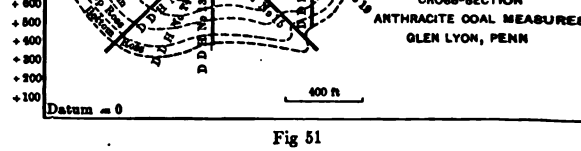


Fig 51

to six acres" (31). L. G. Huntley (32) says: "In spacing wells the utilizatio movement of the oil in a pool and the thorough draining of the maximum an territory with the minimum number of wells are the main considerations. As re first, the wells should be spaced closer together across the dip of the formation th the dip. In many fields the territory is divided among a number of operators, ea ing to extract the most oil possible before his neighbor has a chance to take usually results in the holders of small leases drilling many wells close to bounda forcing the lessees of surrounding land to drill well for well in order to prot own property. No attention is paid to the proper spacing of wells, nor to the effect of one well on another. When large operators control sufficient acreage, the practice in the older fields is to space wells from 400 to 500 ft apart."

Theoretically, if oil wells are spaced to give the same drainage area, they shoul ranged as in Fig 52. Table 6 gives spacings for use with this diagram.

Drill roads are a serious item of expense in swampy districts, and where the top is steep or rough. Standard rigs and diamond drills (except Missouri type) are m wagons requiring roads about 7 ft wide. For traction churn drills (Keystone N Star No 23), used in porphyry copper districts, hillside roads are made 9 ft wid solid. The fill is relied on only in case the machine skids. Max grade advisable which is also about the limit for teams hauling fuel and water. Traction drills c

cks are used when necessary; drills c... sludge;
g ahead. In the Miami district, Ariz., ... current
of 300 ft
Miami,
near ft of
about 75¢
9-hr shift
11) gives

Table 6

Acres drained per well	D Distance between wells, ft	R Radius of drainage per well, ft
1	224.3	129.5
2	317.2	183.1
3	388.5	224.3
4	448.5	259.0
5	501.5	289.5
6	549.4	317.2
7	593.4	342.6
8	634.3	366.2
9	672.8	388.5
10	709.2	409.5

Well
Well
Oil Wells

secured by minimizing delays and labor
for roustabout work so that high-priced
al drilling. Scale of operations limits ex-
profitable. A map is essential to proper
holes are to be drilled must be marked on

ing at hand extra bits and small repair parts.
enough should be provided at each machine so
settling or drying. Supply service for fuel and
should be constructed before drill is ready to
drill are necessary to prevent waste and give
ld be cut to proper length before delivery.

es a driller, helper, sampler, and sometimes
al fuel is used or when wood is delivered in
r each other, one sampler can look after both;
hey should be worked in pairs. Under favorable
within a radius of 0.5 or 0.75 mile. Keeping
vision and use of fishing or other tools in com-
ave his wages; competent foremen receive $150
an extra driller or helper for every 2 or 3 drills.
, casing, and upkeep more than offsets the in-
, casing, and pulling casing, 4 men (usually the
eason, and in general, 2-shift work (either 8, 10
hifts (34).

keeping a number of drills near together has
llest possible crew for 1 drill consists of a drill
(or a driller and helper for motor-driven drills),
n day shift. Foreman sets bits for both shifts
runner does sampling on night shift. In some
used on each shift. In hard rocks giving good
ved, no special samplers are required; the fore-
es. Where accurate samples depend on saving
ry. C. E. van Barneveld cites an illustration
Mesabi range (35). For 30 drills working 1
helpers, 5 pumpmen, 1 blacksmith, 3 diamond
nt, 1 clerk, 2 foremen, and 3 samplers.

drills and makes a considerable saving on
ice is open to objection that the contractor
n drilling speed. This objection is met by

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For detail squares

419

men as samplers on company account, and by payir
N for reaming and casing (Sec 9, Art 6). Where
, it generally pays to buy the machines and import
actice varies. On the Lake Superior iron ranges mos
d, due to complete organization and equipment of loc
tracting firms and their rep
for careful work.

Fig 53

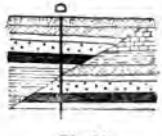


Fig 54

Borehole results are misinterpreted. Fig 53 s
case where 2 holes on the fl
a folded coal seam indi
thicker horiz seam AB. C
use of data from the hole in
would indicate 2 seams ins
the true conditions. Results should be interpreted in the light of co
geological evidence.

Single holes, while valuable for determining position of an orebody, give no info
as to dip or strike, and often furnish erroneous values for thickness. The use of
ing wedges (Sec 9, Art 17) in deep diamond-drill holes, for getting a second sectio
formation from the same hole is a cheap means of securing additional data. Fo
of deflected diamond-drill holes see Sec 9, Art 17. A case of error from single h
occurred in districts such as the Missouri and Wisconsin zinc fields, where ch
holes following rich vertical stringers were taken to prove the existence of thick
bodies.

9. Sampling Boreholes

Accuracy of sampling depends largely upon the character of the r
drilled. Diamond drills give accurate samples in hard rocks where core r
is complete. Accurate samples are also obtainable by churn drills
grained, unconsolidated material, where casing pipe can be driven ahead
drill bit or sand pump.

Soft, broken formations which cave in the hole are most difficult to sample.
the efficiency of the diamond drill decreases both for drilling and sampling, an
drills are commonly used. Vugs, crevices, and brittle sulphides make sampling
Cores are more satisfactory than churn-drill sludge for determining geological det
careful panning of churn-drill cuttings generally makes it possible to locate ch
formation or mineralization within 2 or 3 ft. In coal, cores are necessary to g
rate widths, to locate partings and to secure correct analyses.

Length of samples taken from boreholes varies from 6 in to 10 ft
length, 5 ft. In soft, caving ground, or where mineralization is irregula
lengths are advisable for accurate work. In firmer ground with uniform
the longer lengths give adequate information and save expense for sampli
assaying. Calculations incident to large-scale boring are simplified by
samples in 5 or 10-ft lengths.

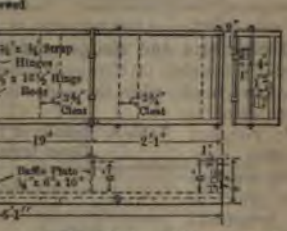
Diamond-drill samples consist of core, or sludge, or both. Where co
cores are obtained they are assayed, or the values estimated. In soft,
rock, where core recovery is low, the sludge must be saved and its assay co
with that of the core. Fine sludge is often difficult to catch, but al
be saved when drilling in ore which is near the merchantable limit of va

Barrels are used for settling sludge in some districts (Art 11); in others, SLUDGI
are made on the ground. Fig 55 shows a standard sludge box (36). Fig 56 sho
cent design (37). The casing pipe projects through the bottom of the box, whic
it to be placed inside the drill shanty. Box is lined with No 26 galvanized iro
baffles are of 0.25-in iron, hinged at top to $\frac{3}{8}$-in rods. For difficult cases, better

es; the first, a small box to settle coarse sludge; ide by 5 ft long, arranged like a whole-current

55. Sludge Box, Cleveland-Cliffs Iron Co, Mich (*Trans* A I M E, 1913)

imes, to force cuttings out of the hole and so Fig 57 shows method then used for catching



Rollin (*E & M Jour*, Mch 28, 1914)

arough one branch of a cross placed in top of forced into opposite branch. The sack used ollecting sludge allowed no dust or slime to pass, indicated the air pressure in the hole (29).

rrors in diamond-drill samples are sed by caving ground. Remedy is to ent the hole, or to ream and case. Sludge lso lost in crevices when all water does return to surface. Putting bran or man- into the hole will sometimes stop this loss. L. Botsford describes another source of r (38).

hole at a Lake Superior iron mine passed ugh following strata: soft slate and pyrite, to 215 ft; banded ore and chert; 215 to 470 oft iron ore, 470 to 545 ft; black slate, 545 to ft. Cores from soft ore assayed 5 to 8% more showed a trace of sulphur; the sludge ran lt ran 5.2% Fe; sludge, 40 to 50% Fe. By them at normal speed for time necessary to much sludge was produced as when drilling ound not caving so badly as to interfere with aterial from walls of hole to salt the sludge.

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Persistence of values in sludge below ore horizon, or "drag" of values (see E samples, below) may be caused also by a low velocity of water current rising the rods.

G. W. Thomson (39), in drilling 4 holes in the Porcupine district, Ontario, fo drag of values as shown in Table 7. Orebody consisted of auriferous quartz strin lenses in schist, the latter carrying no Often the only core recovered from a consisted of short pieces of quartz of a c length of not over 8 in. Values in the sl not correspond with the richer sections of t but appeared 5 to 10 ft below. Drag o was roughly proportionate to the depth at w was cut, and little trouble was exp from caving. Water supply was small, of rising current being from 6 to 8 in

Table 7

Hole	Depth, ft	Drag, ft	Angle of hole
1	320	15	76°
2	705	25	76°
3	655	15	90°
4	1 155	35	90°

Theoretically, it takes about 75 min for a current of 8 in per sec to raise a quar 0.08 in diam from a depth of 700 ft. These conditions would cause the larger and particles of sludge to hang back and appear later than the corresponding core. I tion as to amount of caving, contamination of sludge from walls, and loss of sl crevices can be obtained by weighing or measuring sludge from a sample run, and ing result with calculated wt or volume. This gives an index of reliability of the and shows where casing is necessary. Volume of sludge: cu in = $0.7854\,(LD^2 -$ where L = in drilled, C = core recovered, in; D = diam hole, in; D_1 = diam of

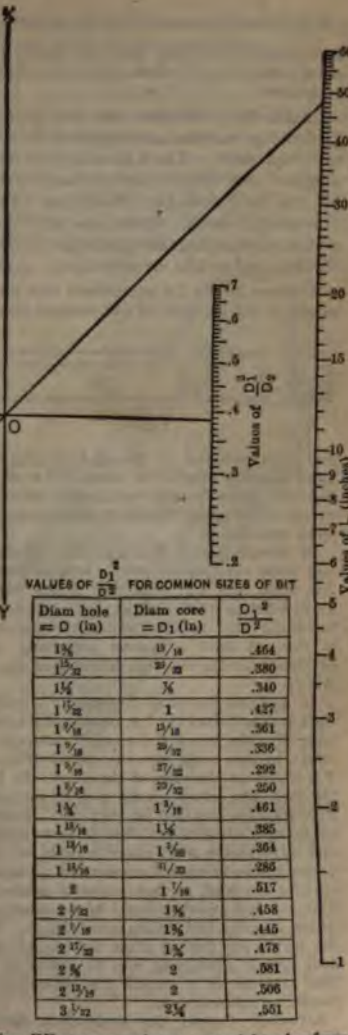
Combining core and sludge analyses. Following methods are u diamond-drill samples: (*a*) Core and sludge from a run are combined and a giving correct results when the sludge is all from the sample run in qu (*b*) Core alone is assayed; correct only when complete core recovery is made, which is rare. Sometimes value of core is estimated from its appearance (Art 11, S E Missouri). On the Rand, cores are split longitudinally; one half is assayed and the other filed for reference. (*c*) Sludge only is assayed; correct only when none or all of the core enters slud being ground up in core barrel. (*d*) Core and sludge are assayed sepa and an arithmetical average taken. This is incorrect, because volumes and sludge are unequal. (*e*) Core and sludge are assayed separately and values combined in proportion to their volumes. This gives correct r also the cores may be split and one half kept for geological study. (Sam Table 8, were taken in 5-ft lengths.)

Table 8. Results of Two Methods of Assaying Diamon Cores from Iron Ores (W. J. Mead) (40)

		Core, in	% iron	Aver value (% F	
				Method (*d*)	Meth
Soft ore	Sludge....	....	51.55	36.37	48
	Core.....	24	21.20		
	Sludge....	....	44.15	37.10	43
	Core.....	9	30.05		
	Sludge....	....	48.95	41.47	46
	Core.....	37	34.00		
	Sludge....	....	45.45	37.55	44
	Core.....	11	29.65		
Hard ore	Sludge....	....	28.20	35.63	30
	Core.....	31	42.45		

Calculations for combining core and sludge analyses. Let D = diam hole, diam of bit outside carbons + ⅛ in; D_1 = diam core, in, = diam bit inside carbo ⅛ in; L = in drilled; C = core recovered, in; V_1 = cu in of core; V_2 = cu in of $V = V_1 + V_2$; A = aver assay of core and sludge; A_1 = assay of core; A_2 = sludge. Then, $A = \frac{A_1V_1 + A_2V_2}{V} = A_1\frac{V_1}{V} + A_2\frac{V_2}{V}$;

$\frac{V_1}{V} = \frac{C}{L} \times \frac{D_1^2}{D^2}$ and $\frac{V_2}{V} = 1 - \frac{C}{L} \times \frac{D_1^2}{D^2}$; whence, $A = \frac{C}{L} \times \frac{D_1^2}{D^2}(A_1 - A_2) + A_2$.



VALUES OF $\frac{D_1^2}{D^2}$ FOR COMMON SIZES OF BIT

Diam hole = D (in)	Diam core = D_1 (in)	$\frac{D_1^2}{D^2}$
1⅜	19/16	.464
1 15/32	39/32	.380
1¼	¼	.340
1 17/32	1	.427
1 9/16	19/16	.361
1 19/32	39/32	.336
1 5/16	37/32	.292
1 9/32	29/32	.250
1⅜	1 3/16	.461
1 15/16	1⅛	.385
1 13/16	1 5/32	.364
1 15/16	31/32	.285
2	1 7/16	.517
2 1/32	1⅜	.458
2 1/16	1⅜	.445
2 17/32	1⅜	.478
2⅜	2	.581
2 13/16	2	.506
3 1/32	2¾	.551

ine *XY* at same point as a straight-edge from of core (*C*) were recovered in drilling 48 in (*L*) th straight-edge, connect 21 on scale *C* with *Y*. Hold straight-edge on *O* and 0.385 (scale $t \div LD^2$. For solving this problem the 2 posi-es passing through *O*.

bining Core and Sludge Assays

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H. L. Seward has constructed the diagram (Fig 58), from which values of $\frac{C}{L}$ be obtained for all cases where the sample run is 5 ft or less. When ratio b and L is less than 0.1, effect of core assay may be neglected, and assay of sludge as the aver.

Churn-drill samples (see also Sec 9). With hollow-rod drills, usi for flushing, samples are caught as for diamond-drill sludge. A sand pum with rope drills. The 6 to 10-in bits employed with these drills produ samples. At Miami, in a 5-ft run, a 10-in bit cut about 450 lb of rock; bit, 260 lb; 6.25-in bit, 180 lb (41). Practice in handling these large of material varies. Water and thin sludge are sometimes wasted, a and heavier cuttings dumped into a bucket for assay. This results in a of slime and serious errors in assay, especially when drilling in brittle s

Keystone Driller Co recommends that the sand pump be discharged into shown in Fig 59, built of 1-in surfaced lumber and set on low horses in from

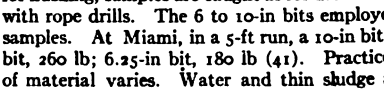

Fig 59

The partitions are set lo tween strips nailed to sid for removal when clea Sludge is dumped into the partment. After settling, drained off through plug far end of box. Remain and water are swept out through large plug hole into a tub or pails, and dried Good sampling practice for churn-drill work has been developed in the porphy districts, where the numerous samples taken and necessity for saving chalcocite slime make this work difficult.

At Nevada Consol Co, Ely, Nev, all sludge and water were poured directly into galvanized-iron wash tubs, and dried over a slow fire to avoid roasting the sulphides. A 5-ft sample required 4 tubs. In other districts, the sand pump is dumped into a launder containing split dividers, built like the Jones riffle sampler (Sec 30, Art 1).

L. S. Cates furnishes drawings of a splitter used by Ray Consol Co. It consists of a launder (Fig 60), into which the pump is dumped and which in turn spills into the splitter (Fig 61). Fig 62, 63 are drawings of the riffles. At Ray, the last split (about 40 lb) is dried for assay. Five tubs are provided for each drill. Samples are taken in 5-ft sections. Sampler keeps a record and pans sludge from upper portions of hole. When specks of mineral appear, the 4 previous samples are dried, so that assays of ground 20 ft above first visible mineral are obtained and useless assaying and drying avoided (42). Similar devices are used in other districts, giving a final sample containing 0.25 to 0.0625 of the total cuttings in a sample run, depending on number of splitters used.

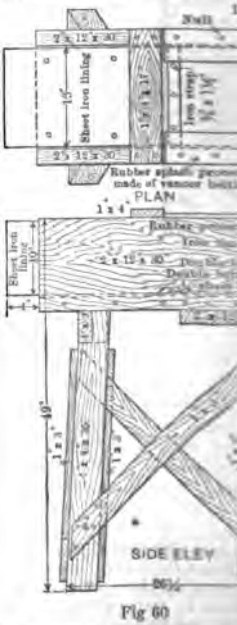

Fig 60

Savannah Copper Co, Burro Mtn district, N M, dried samples in an assay office at a distance from drills. If sludge showed visible mineral, it was reduced in splitter to about 4 gal; if not, it was cut to 1 gal. Final samples were put into 1-gal milk cans, with tight-fitting covers, in which they could be transported in wagons without loss (43). One sampler did for 2 drills, kept notes and checked drillers' measurements of depth.

splitters, and tubs must be scrupulously id salting succeeding samples.

Errors in churn-drill samples, besides hose due to careless handling of sludge on urface, may occur as follows: (*a*) Caving of alls of hole at some point above sample. his results in high or low assays, depend- g on whether cave takes place in rich or arren material. (*b*) Caving of walls of hole t point where sample is being taken. If ave is all from a rich streak, sample is false.

Fig 62

ch caves on accuracy of sample is small. head. (*c*) Inrushes of soft material from t for many feet, entirely destroy accuracy dge by barren or rich material, rubbed off action of tools and rope, may cause serious taining brittle sulphides or where a differ- s between commercial ore and waste. (*e*) ludge lost in this way is apt to consist of or depends on character of mineralization. nc fields where entire sludge has been lost barren (44).

hrough wide variations in assay of adjacent en actual and calculated wts of sludge. Bad lf. Remedy lies in proper use and handling of ept close to bottom of hole and sometimes be h holes a safe rule is to drive casing deeper be- ll sources of error would be eliminated if the drill nd casing driven to bottom of hole before drill-

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ing ahead for next sample. This procedure is costly and offsets advantage of chu in both cheapness and speed. Also, for reaming and handling casing in deep hole wheel is required, which is not usually on portable churn drills. An approx to above procedure is obtained more by starting hole with a large bit, ca soon as sludge shows mineral, and c ing hole with a smaller bit. When ca serious, another casing pipe is inserte the first and drilling is resumed wi smaller tools (Art 11, Porphyry co In any given district, practice genera resents a compromise between cost an lute accuracy. Pains are taken samples accurate enough for the p beyond that, cost of reaming and ca avoided.

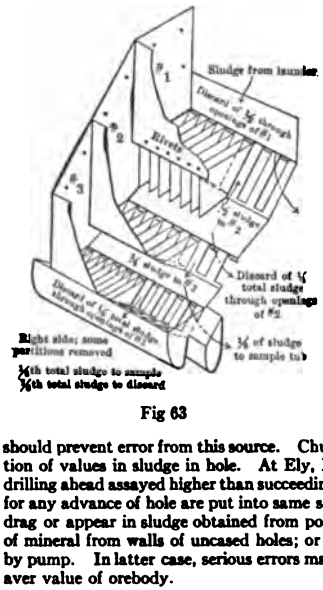
Sludge from launder
Discard of ½ through openings of #1
Rivets
#1
#2
#3
½ sludge to #2
½ sludge to #3
Discard of ½ total sludge through openings of #2
Discard of ½ total sludge through openings of #3
Right side; some partitions removed
½ of sludge to sample tub
½th total sludge to sample
½th total sludge to discard

Fig 63

Another error in churn-drill sampl curs through failure of sludger to pick cuttings. Experiments have been m determine loss from this source in plac ing. A piston (vacuum) pump in goo should be used. I. J. Stauber, while in Burro Mtn district, N M, made it a bail not less than 15 times after each run (43). Fixed rules based on expe should prevent error from this source. Churning action of tools may produce a con tion of values in sludge in hole. At Ely, Nev, it was found that the first bailerfu drilling ahead assayed higher than succeeding ones (45). This causes no error if all b for any advance of hole are put into same sample. In churn-drill holes, values som drag or appear in sludge obtained from points below ore horizon; caused by dislod of mineral from walls of uncased holes; or it may be due to incomplete recovery of by pump. In latter case, serious errors may result in interpretation of both thickn aver value of orebody.

Borehole results should be checked by means of shafts or raises at v intervals, especially in new districts and where they are to be used as a ba calculating tonnage and aver value.

10. Boring Records

Forms are needed for keeping costs, and for preserving sampling an logical data. Fig 64 shows typical cost-keeping form for diamond drillin 65, a similar form for churn-drill work. Fig 66 is an admirable daily record Fig 67, 68 are forms for summarizing drilling, geological and sampling d connection with Fig 64 (36). Fig 69, 70 are combined forms for underg diamond drilling; the smaller is for the foreman on each machine; head makes up the larger form, which is filed daily in chief engineer's office (4

Printed forms are preferable to notebooks, as they secure uniform data. In smal churn-drilling, the sampler's field notes are often recorded in an ordinary surveyor's sit book. The hole is drawn to scale on one page, and rock formation, changes in gr depths, water-level, character of mineralization, and sample numbers are entered in p position as they are obtained; the opposite page is used for explanations and other These notes are transcribed into a similar loose-leaf book for office use, in which are also recorded. Complete detail is essential in interpreting results. Besides the records should include: date of starting and finishing hole; total depth; location o elevation of collar; time for casing; names of drillers, samplers, and helpers (see also S

Platting boreholes. Drawings of drill holes, showing formations and po and assay of ore, are often necessary for intelligent computation of aver va

IFFS IRON COMPANY
, Michigan
MOND DRILLING

.....19t....

.....
..... Hole No.....

	Feet	Core saved	Hours
- - - -	.....		.....
- - - -	.....		.....
- - - -	.....		.....
- - - -	.....	.....	.....
.....ft	.....		.....
.....ft	.....		.....
- - - -	.....		
.....	.....		

	Feet	Size	Hours
- - - -	.....	.....	.....
- - - -	.....	.....	.....
- - - -	.....	.....	.....

....
.....

.....Runner
.....Helper
.....Setter
.....Foreman

in) (*Trans* A I M E, Vol 44, p 82)

RECORD

DATED COPPER CO

..Hole.....
.....191...
.....
.....
.....
.....
.....
.....
.....
.....
.....
.....
.....
.....
.....
.....
.....Driller
.....Foreman
r.....

size 3¾ by 8¼ in)

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GIROUX CONSOLIDATED MINES CO

CHURN DRILL REPORT

Hole Number———

A. Chalcocite
B. Pyrite
C. Carbonates
D. Cuprite
E. Chalcopyrite
F. Native Copper

G. Iron
H. .....
1. Porphy
2. Limest
3. Rhyoli
4. .....

.....Claim,.....Drill No....

.....Shift.....1914

		HOURS						
	Occupation	Drilling	Moving	Repairs	Casing	Removing Casing	Fishing	
	Driller Helper	(2 *lines*) (2 *lines*)						

Size....Length. Depth of Hole beginning Shift.....ft. in. Adv..

of..... Depth of Hole at end of Shift.....ft.....

k	Sludge, Color	Hard-ness	Size of Cut	Caving No Yes Slightly	No Ball-ers 5 ft Sample	Exact Length of Sample Ft In	Exact Depth Ft In	Is Sam-ple Re-liable?	
)									

REMARKS

hort account of condition of hole, rock, and depth and thickness of c
seams or fault gouge)

.....

(4 *lines*)

Vater Level.....ft Supplies Needed.....

r Sand Line Pulley to Platform.....ft.....in ..... D

t in Tubs, No..... .....Sa

cked, No..... Seal No.....

Fig 66. (Original size, 8.5 by 11 in)

THE CLEVELAND-CLIFFS IRON CO

RECORD OF DIAMOND DRILLING

..
..
} *State....T....R....Sec....Hole No....Location:*.....

.....*Course:*.....*Drilled by whom*.....*with*.....

o ur	Core Recovered Ft	In	Stand pipe	Drilled	Total Depth	Kind of ma-terial	Begins at	Dip of strata	R m

Loose-leaf Record (Orig size, 14 by 16 in) (*Trans* A I M E, Vol 44, p 88)

THE CLEVELAND-CLIFFS IRON CO

ANALYSIS OF CUTTINGS AND CORE

...
...
...
} *State.....T.....R.....Sec.....Hole No.....Location:* ....

Determined by..... 191.... *Elevation of Collar*.....

	Iron	Phos									

Loose-leaf Record (Orig size, 14 by 16 in) (*Trans* A I M E, Vol 44, p 88)

T, DIAMOND DRILL

.....

.....191...

eginning of shift.....Feet
nd of shift.....Feet
.....Feet
.....Feet
g put in.....Feet
pipe put in.....Feet

	Feet	Material

.....

.....Foreman
.....Mine
rs, accidents, etc

ur, Dec 10, 1910)

T, DIAMOND DRILL

....*Level*.....*Mine*
.....191....

	Carbons on Hand		Carbons in Bits		Carbons Received		Fragments on Hand	Loss in Drilling
	Number	Carat	Number	Carat	Number	Carat	Carat	Carat

ERIAL	ASSAYS			
	Cu	Ag	Au	Pe

.....

breaking of carbons, etc Head Driller

tte (*E & M Jour*, Dec 10, 1910)

oles on a loose-leaf form of tracing cloth, size 14
Fig 71 shows a scale plat of part of a diamond-
classification of strata drilled; col 2 gives con-
col 3 shows thickness of each stratum. These
n = 20 ft; smaller scales will not show details of
ed (Sec 9, Art 17) deviations are shown on plats.

mond-drill cores are usually open trays,
e enough to take 5 or 6 rows of core; longi-
ooves 1/16 or 1/8 in wider than diam of core.
th 0.5-in slats for partitions.

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If much drilling is to be done, cases should be built into which core boxes slid drawers. Fig 72 shows core box for this purpose; corners are mortised, slats and b are nailed in place. It is advisable to mark by an arrow the direction in which core is laid in box; also depths at end of each row. A more intelligible record is obtained if sticks of proper length are inserted to mark

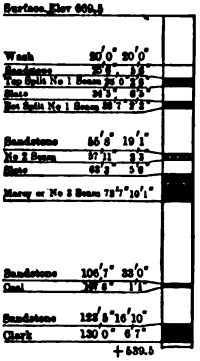

Fig 71

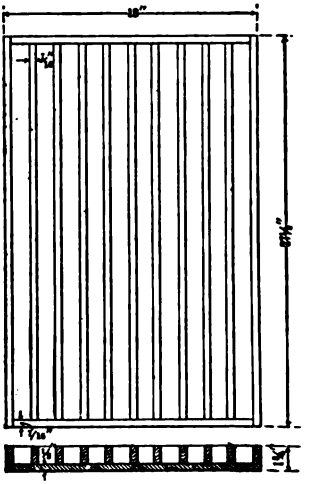

Fig 72. Diamond-drill Core Box

lost core; character of lost core as determined from sludge may be noted on the stick For storing large amounts of core so that they will be easily accessible for observ narrow shelves may be supported at intervals of 1.5 in on the sides of A-shaped frame

11. Examples of Boring and Sampling Practice

Porphyry coppers. These orebodies are large, low-grade deposits of c minerals (usually chalcocite), disseminated in altered and shattered mon porphyry, granite, or schist. They have large horiz dimensions and are ov by leached zones (capping) varying in thickness from 0 to 600 or 700 ft. Atte was first drawn to some of these deposits by favorable surface indications; c were discovered while working higher-grade deposits in their vicinity. range from 1.5% to 3% Cu; lower limit is a commercial one.

Boring practice on these deposits in different districts varies only in minor d Holes are bored on corners of 100 to 400-ft squares; most often, 200 ft. Dri results are checked at intervals by test pits, raises or winzes, and by drifts run on c nate lines to show variation in copper content between holes. Table 9 shows t examples of this type of orebody, and relative amounts of boring and underground done by different companies, before actual production began. An indeterminate p the footages given for underground work at Miami, Ray, and Inspiration represen velopment work preparatory to mining; figures for Ajo cover exploration work Most boring has been done by steam-driven churn drills. Traction drills, Keystone Star No 23, or Cyclone drills of corresponding size, have proved satisfactory. Expe shows that economic limit of depth for this size of drill in these formations is 600 to At greater depths, cost per ft of hole increases rapidly. For holes up to 350 ft depth corresponding in size to Keystone No 3 are cheaper, lighter, and large enough. F tails of churn-drilling practice on the porphyry coppers, see Sec 9, Art 3, Tables 9, 13, 14, and 15; for churn-drill sampling practice, see Art 9. Webber (33) notes var in assay between drill samples and those taken from check raises of as much as 0. ore averaging 3%. H. A. Fuld (49) reports that "where raises have been cut aroun holes, the ore from the raises averages several tenths higher than the results from

l chiefly to loss of slime in crevices. Annual that complete mining of representative areas of 5% of the total estimated copper content from

Mines, Development and Boring

Aver thickness cap-ing, ft	Aver thickness of ore, ft	Tons ore per acre	Total ft of drilling	Depth of drill holes, ft	Date
200	....	418 600	19 200	600 aver	1911
368	167	543 000	45 433	.....	1911
242	113.5	395 200	90 000	200–970	1910
†	50–600	727 000	23 097	275 aver	1913

27, 1911 and "Copper Handbook," 1910–11.
1911.
Edwin Higgins, *E & M Jour*, Apr 23, 1910.
Vol 49, p 593.
00 ft of raising had been done alongside of drill

it is overlaid by low-grade carbonate ores, 20 to

f Boring, Porphyry Copper Mines

hr shifts, ft	Aver cost per ft (a)	Wages			Date
		Drillers, per 12-hr shift	Helpers, per 12-hr shift	Samplers per month	
	$2.90(b)	$6.00	$4.80	$150	1910
(k)	2.75(c)	6.00 5.00(k)	4.80 4.00 (k)	100	
–25	1.19(d)	5.00	4.80 to 4.00	.....	1907
–25	1.87(e)			.....	
61	2.52(f)	6.00	4.50	75	1910
.6	1.68(g) 2.49(h)	6.00	4.00	110	1911
...	168(i)	5.46	4.33	100	1913
15	5.84(j)	6.00	4.50	100–150	1912–14

uilding, teaming, water, fuel, general expense, l expense, 52¢, sampling and collecting technical ls, 27¢ per ft of hole. 9 drills operated simultan- ads cost 44¢ per ft of hole. Water supply, 4¢; samplers, samplers' supplies, and assaying, 21.5¢ machines used (34). (*d*) Topography not very 45). (*e*) Cost of roads, 47¢ per ft of hole; water drills used (63). (*f*) Assaying and sampling 15¢; hole. 2 Star No 23 drills (43). (*g*) Assaying and t, 13¢ per ft of hole. 2 Star No 23 drills (43). er ft; roads, 2.4¢; no charge for general expense. Sampling cost 26¢; general expense, 96¢; estim- . 6 drills operated. (*k*) Aver speed for two 8-hr

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Chile Exploration Co, Chuquicamata, Chile. P. Yeatman and E. S. Berry furni lowing data on drilling from beginning of work, April, 1912, to Oct, 1914: High due to extraordinary expenses and troubles incident to starting work in a remote At first there were numerous small difficulties in getting fuel to drills; water had hauled 2 miles by mule team to an elevation of 1 000 ft, while pumps and pipe line being installed. Delays during first 6 months' operatión, due to late delivery of parts from U S, caused shutting down of some drills for weeks at a time, while wi drillers and helpers continued. 11 holes were over 1 000 ft deep; many were betwe and 1 000 ft. 8 churn drills were purchased, costing, with necessary repair parts, de on property, about $67 000, or $1.47 per ft of hole drilled; this cost is not included in

Miami Copper Co, Ariz. A little underground diamond drilling was done i from 570-ft level; cost, $5.44 per ft; speed, 5.5 ft per 8-hr shift (29).

Ajo Copper Co, Ariz. Diamond drills used exclusively. J. C. Greenway an Joralemon furnish following details: When work was started, it was decided to dril ft of hole on most favorable showings, to see whether property had possibilities of a mine. Diamond drilling could be contracted at $4.75 per ft; no churn-drill cont were available. The limited time required operation of 2 drills. Cost of buying 2 drills was $8 000; cost of churn drilling was estimated as $3 per ft. Assuming sa costs equal for both drills, and allowing nothing for 2 churn-drill outfits at expira work, diamond drilling would save $5 375 on 1 500 ft of hole. These factors, con with risk involved, led to choice of diamond drills. They gave accurate samples, a results warranted further exploration. To secure uniform sampling data, work wa tinued with diamond drills at contract price of $6 per ft. Extremely rough topog which would have prevented construction of drill roads, also influenced choice of

Holes were drilled on corners of 200-ft squares; company men did the sampling. percentage of core was recovered, because of fractured nature of rock. Drill hole sampled in 5-ft sections. All sludge was caught in barrels and settled, water dec and sludge dried and quartered to 3 or 4 lb. Small samples of core and sludge fro 5-ft section were kept for reference. Sludge and core assays were combined fo figures. Shafts, 4 by 6 ft, aver depth 50 ft, were sunk to check drill hole results. ing was done by hand drilling, with Mex and Indian labor. Muck was sampled as were sunk; later, groove samples were cut on all 4 sides of each shaft. A little ing was done on coordinate lines, to show variation in copper content between adj holes, and some raises were put up on drill holes from drifts. Two years were sp doing the following work; 84 diamond-drill holes, 200 to 1 000 ft deep, total, 23 17 1 059 ft of shafts for checking drill holes in carbonate ore; 175 ft shafts for checkin holes in sulphide ores; 2 721 ft shafts in advance of drilling; 1 513 ft drifting in su ores; 142 ft raises. Calculations based on this work showed about 40 000 000 ton Shaft samples in carbonate ore averaged 0.005% lower than corresponding dri samples; shaft and raise samples in sulphide ore averaged 0.05% lower than dri samples, and drifts in sulphide ore averaged 0.26% higher than values indicated by holes at corners of blocks. (This remarkable agreement appears to be due to un character of ore, rather than to any inherent accuracy in diamond-drill samples. Au Table 11 gives cost of drilling, based on 17 310 ft of hole, and cost of sinking 4 000 shaft; superintendence, office and engineering expense are not included. Note tha cluding sampling, a 50-ft shaft at Ajo cost less than a 50-ft drill hole (27).

Table 11. Costs at Ajo Copper Mine, Ariz

Diamond drilling	Cost per ft	Shaft sinking	Cost p
Contract price	$6.00	Labor	$4.96
Supplies	0.0174	Supplies	1.18
Miscellaneous	0.0379	Bl'smith and carpenter shops	0 40
Sampling	0.3422	Miscellaneous	0 07
Assaying	0.1370	Sampling	1 19
Total	$6.5345	Channel sampling	2 28
		Total	$10 11

* Aver depth, 50; max, 125 ft. Last few ft of shafts, 125 ft deep, cost over $30 pe

s are masses of soft or hard hematite, lly called "iron formation." Orebodies centration in troughs caused by folds or s lie horizontally and present ideal con- by boring. Elsewhere, the iron forma- exposed, show their edges only. Region stern Menominee, Gogebic, and parts of hard-rock outcrops, to which early ex- and test pits, was confined.

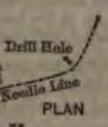
d exploration is now limited to diamond drill- l drilling has been done at random, but geo- the usual hazard of exploratory work is con- with intelligent supervision from 2 to 8% of nding on the district (50). Maps, published w location of areas covered by iron formation gy, stratigraphy, and geological structure are e before drilling is begun. This information ds to eliminate worthless areas. Highly sili- e in which the rocks of the iron formation are ercial orebodies.

Menominee and f structure.

d perpendicular amples (P. B. typical.

ill hole, 600 ft dike and a low iron formation red a favorable e range. "Only outcrop is a ridge of Ajibik ate usually forming the footwall of ore forma- om the quartzite, and hole started far enough . Fig 75, Amasa district. Good orebodies rops; surface drift, 100 ft deep. Magnetic

Fig 73



75

Fig 76

cated a bend in formation about a mile away. ore concentrations, a diamond-drill hole was dip line as mines *A* and *B*. Fig 76, Crystal ich was abandoned in a fine-grained gray dike copic examination revealed its true character t of ore.

nges costs $1.25 to $8 per ft; usually between

llowing instructions for saving diamond-drill posts in its drill shanties (36).

low floor of shanty, in such position that there mple without moving box. Connect a tee to from it to nearer end of sludge box, at such sludge box and just rest upon its top, and of

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such length that it will not project more than 1 in beyond edge of box. The pi not exceed 2 ft long; if longer than 1 ft, it must be split on top for 1 ft nearest slu so that sludge collecting in pipe may be seen. Set box level, so that water will evenly across whole width at far end, and wedge partition firmly in close cont bottom of box. Top of partition should be 1 in below water level. Box is now receive sample. While drilling, care must be taken that no water from drill hol around or over the tee, except through pipe leading to sludge box. There must b from box and the plug at the end must be tight. Sludge samples are taken for e drilled, or less, preferably from even 5-ft intervals; that is, from 460 to 465, 46 etc. Whenever a sample is to be taken, drilling must be stopped and hole wa clean. Pipe leading to sludge box must be cleaned out into the box, and pu then be stopped or tee turned so that water will not discharge into box. Carefull partition in box, so as not to stir up sludge. When sludge is settled, siphon o water, keeping end of siphon near surface of water, and not disturbing or drawin fine sludge from the bottom. Siphon may be allowed to flow until sludge beg off with water. Take out hose and thoroughly mix sludge in box to a mud, whic placed in a pan on a boiler to dry. Pan must be at least 8 by 12 in by 1 in deep, bottom, and is thoroughly cleaned each time before a sample is put into it. I water cannot be drawn off, without disturbing sludge, so that sample can be c in this pan, use a larger pan. When sludge has been cleaned out, remove plug at en wash out with a pail or two of water, then replace plug and partition, and drillin started again. Sludge must be labeled, giving depths between which the sa taken, and must all be saved and turned over to inspector when drilling in iron fo or in other ferruginous or red material. While drilling in material from which should be taken, if water is lost, or if sludge does not come up with water, or if taminated with material caving from upper part of hole, drilling must be stop hole is put in such condition that good sludge can again be obtained, or until permits drilling to proceed. Whenever drill runs into or out of ore, 1 ft or more th ing must be stopped, and sludge box cleaned out, without waiting to complete run. When drill runs out of ore, continue taking and saving sludge samples for a ft, no matter what the material, so that it may be determined whether ore is c

Core. Keep core separate from sludge; each time core is pulled, label it wit between which it was recovered. Each run of core is kept separate, and all c and turned over to inspector. When core is pulled, if it is found that more is sa in proportion of 1 ft of core to 10 ft of drilling, sludge box must be cleaned witho to complete the 5-ft run, and the sludge labeled and saved separately. If slud shorter distance than 5 ft is in box at end of shift's work, and if less than above p of core is saved, sludge may be left in box, provided shanty is locked and box is in from outside. If anybody can get at the box, and if there is no watchman, slu be removed from box, dried, labeled, and placed with other samples.

Mesabi Range, Minn. Ores are massive, flat deposits of hematite, horiz extent as compared with their depth. Orebodies are generally l 200 ft thick, and may extend laterally for a mile or more. They are by soil and glacial drift averaging 65 ft deep. Fig 83 (Art 12) shows a section. There are few surface indications of ore. Early practice was test pits through the drift, and as far into ore as permitted by facili cost for handling water, hoisting, and ventilation. Exploration was c from bottom of these pits by boring. At present, both prospecting and tion are done entirely by boring from surface, which is cheaper, quic avoids dangers from numerous open test pits.

Boring practice. Both churn drills, with perforated bit and hollow rods, and drills are used (35), with water under pressure for flushing cuttings to surface drills are used in surface drift, soft rock, and ore; diamond drills in taconite and o formations. Holes are churn-drilled to hard rock and cased with heavy 3-in. l pipe. Diamond drill is then used until ore is struck or hole abandoned. If more of ore is found below taconite, hole is enlarged by blasting, a 2-in casing put in, an continued with churn drill. Casing is kept close to bottom of hole. Land is h acre tracts, and for preliminary prospecting 5 holes are put down on each tract. of the holes varies; common arrangement is to put 1 hole in center and othe

space their holes about 500 ft apart. Fig 77 ts are being drilled for same parties. If ore is rily abandoned. 5 holes are not final proof; in ontaining 1 000 000 tons was missed. If ore is rdinate lines are exploration done , or 300 ft apart, conite to a depth hole in each 40- hich underlies all entirely by con- a reputation for

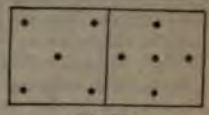

Fig 77

combined diamond and churn drill) consists of ges per shift: drill runners, $5; helpers, $2.25; 90 to $125 per mo. Aver drilling speed for 10 ft per shift, allowing for breakdowns, mov- tractor for churn drilling is $1.75 to $2 per ft; of road building and pumping water is high.

es following data (35). Surface soil is not diamond drills are discharged with flushing tled in barrels, tubs or a series of settling tanks. churn drill, used in all soft formations, allows In passing from sandy drift into ore, casing is e difficult ground is drilled "dry"; pump is water to form a sludge, casing is driven to bot- flushed out. Drill holes are sampled in 3, 5, or supplied with 4 to 7 barrels. A hole bored in fitted with a wooden plug, allows most of the led; remainder stands 2 or 3 hr and is decanted. s for analysis. Core samples are taken in hard er 5 ft drilled in hard ore, slate, and quartzite. drill sludge, if core recovery is more than 30%. With care, very good samples are obtained; equent mine samples check surprisingly closely. rd material and small streaks of soft ore, show Very soft fine ore often stays in suspension, sandy ores of the Western Mesabi, concentra- ns may overrun several points; occasionally the ed carefully, with minimum quantity of water.

a, contain residual brown iron ores, up to he orebodies have enormous lateral area; more than 70 sq miles. There is no over- heavy growth of timber and underbrush. time in boreholes without caving.

D. E. Woodbridge (54) give following data. ade from ordinary 2-in carpenter's augers with first borings were on corners of 100-ft squares. rval between holes was successively increased s were checked by borings 25 and 50 ft apart on ch representing several million tons." Subse- lculated from boreholes. In the Moa district, es, though some drilling was done at greater and rilled in an area of 8 100 hectares (1 hole per 22 906. Work was checked and confirmed in 1910 ependent engineer. Two men on an auger put o ft. Drillers were paid 1.5 to 2¢ per ft of hole pler were provided for each crew. Occasional the ore. Where holes were sampled in sections, as a sample. Where sectional samples were not th, coned and quartered down.

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Southeastern Missouri. The disseminated lead ore occurs in flat m dolomite, at depths of 200 to 600 ft. Thickness of orebodies, up to 70 cover considerable lateral area. Overlying rocks are limestone and sh glomerates, and dolomite, free from chert. There are no outcrops; p ing and exploration are done by diamond-drilling from surface. Churn d used as auxiliaries to put holes down to bed rock through deep surf Geologic features indicate probable orebearing areas in a broad way.

Boring and sampling. H. A. Guess, Gen Mgr, Federal Lead Co, furnishes ing data (Fig 78) (See also Bib 55). "In extending the limits of known orebo only way to do is to feel along with the diamond drill until the full limits of ore defined. For this work, holes are close-spaced, usually 200 ft, sometimes 100 general prospecting for new orebodies, holes are spaced 400 or 600 ft or even furth in checkerboard fashion or in lines transverse with supposed strike of ore run. W is cut, detailed drilling is begun with spacing of 200 ft or less." Procedure va different companies. A solid bit is used until near expected ore horizon; then a with a 5 or 10-ft core barrel, until the hole bottoms in sandstone known to und orebodies. About 25% of total footage is cored. Lead content of cores is estima their appearance; one estimator is employed for 10 or 12 drills. Cores are assay sionally as a check. Throughout the mineral zone, all sludge is caught and ass check estimations and core assays. Sludge serves as a further record when a core part of a soft lead layer is ground up in trying to core it. Combined solid and drilling costs 60¢ to $1 per ft, average 75¢. Aver speed of drilling per 10-hr s solid bit, 60 ft; for core bit, 30 ft. Fig 78 shows outline of workings on one orebo

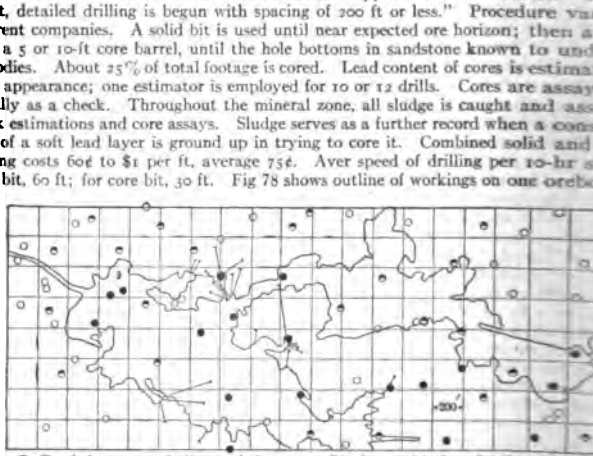

Fig 78 Boring near Flat River, Mo (H. A. Guess)

location of holes for preliminary and final exploration. Short horiz holes are also from different headings, to supplement indications given by holes from surface. results are used for laying out work and estimating tonnage and value (Art 12).

Joplin, Mo. Irregular orebodies of sphalerite in breccia of chert and stone lie at depths of 75 to 250 ft. Ore occurs in large or small pockets narrow stringers, or in horiz sheets 10 to 20 ft thick, of considerable are early days, shafts sunk on outcrops of iron-stained chert led to discovery o low pockets of galena. After discovery of deeper zinc ores, prospecting wa by shafts sunk blindly. At present, churn drills are used exclusively, bein ferred to diamond drills because of fissured ground and brecciated chert. is essentially prospecting; little attempt is made to block out orebodies, count of their irregularity.

Boring practice. O. H. Sheldon and W. T. Inman furnish following data: Ke No 4 and 5, or Star No 23, drills are used. Holes are carried 20 to 30 ft into rock w 8.25-in bit; a 6.25-in bit is then used to 100 ft, or until ore is found; for greater d and for drilling in ore, a 4.25-in bit is used. Holes are cased; diam of casing, 8.75 and 5-in; when in ore, casing is kept as close to bottom of hole as possible. Aver s drilling, 15 to 20 ft per 10 hr. Contract price for drilling, 90¢ per ft. Wages:

f holes is rare. Where ore is found, holes attempts are sometimes made to trace ore-ugles to each other. It is a saying in the dis-nore ore than subsequent mining.

s: The common method is to catch as much y water bucket and let the rest go to waste. Careful sampling has been done by St Paul ter, delivering half to a 20-gal tub. Sample ich was dried for assay. Holes were sampled t 2¢ and 4¢ per ft of hole.

egular orebodies of blende and galena in 100 to 200 ft. Country rocks are lime-thin bedded earthy material to thick, umerous crevices, vugs and inclusions of onfidence is felt in drill-hole results for ue chiefly to irregularity and compara- to drilling and sampling methods.

ystematically. Prospecting is done in areas, as formerly mined from shallow gash veins. k") is struck. Sometimes succeeding holes sit is determined, and are then spaced farther stringer gives impression of a thick orebody. ced around the first strike in a circle of 50 ft am. They are often cased with 6, 5.625 and per shift. Drilling is done by contract, at and about 90¢ per ft for depths of 150 to 250 for any which cannot be pulled. A small rict has cost $3 to $3.50 per ft. Carbon loss ear Linden 60 to 80% of core was recovered. izabethtown, Ill, cost $1.25 to $2 per ft. No Wages, $1.65 to $2.25 per day.

very 3 ft in barren ground. In ore horizon, operators put sludge into a pail to settle, de-tings. Values are frequently estimated from nally a hole is dynamited, to furnish large deposit is disseminated or massive ("sheet loss of the hole.

sphates. Residual deposits of barite, ain by clay and soil from a few feet to ring phosphate deposits, coordinate lines ground and holes sunk at intersections gers. About 1 test pit per acre is sunk to entirely in very irregular deposits (58). ract of about 11 acres containing a deposit of clay, 10 to 75 ft deep. Ore occurs near bed e used for holes at intervals of 50 to 100 ft. Two men can bore 20 to 40 ft per day. This mples in dry clayey material, which will stick caving (59).

es, of Ore Tonnage and Value

t 17; for platting boreholes, see Art 10)

. Let $V_1, V_2, \ldots V_n$ = assays of suc-ir respective lengths; V = aver assay of

$$+ \cdots + V_nL_n \qquad (1)$$
$$+ \cdots + L_n$$

is arithmetical aver of assays of samples.

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Average value and tonnage of an orebody is calculated from bor sults in different ways, depending on ore occurrence and information de

Examples. Iron-ore deposits at Moa and Mayari, Cuba, and placer dredg erties, are examples of surface orebodies with no overburden, and fairly regula For such orebodies, estimates are made thus: Let $V_1, V_2, \ldots V_n$ = aver valu of drill holes 1, 2, 3, . . . n; $A_1, A_2, \ldots A_n$ = respective areas of influence holes, sq ft; $D_1, D_2 \ldots D_n$ = respective depths of holes, ft; A = total area o V = aver value of deposit; D = aver depth; T = total tonnage; C = cu ft p place; n = number of holes. Then,

For holes spaced irregularly:

$$V = \frac{A_1V_1D_1 + A_2V_2D_2 + \cdots + A_nV_nD_n}{A_1D_1 + A_2D_2 + \ldots A_nD_n} \quad (2)$$

$$D = \frac{A_1D_1 + A_2D_2 + \cdots + A_nD_n}{A} \quad (3)$$

$$T = \frac{(A_1 + A_2 + \cdots + A_n)\,D}{C} \quad (4)$$

For holes spaced at regular i

$$V = \frac{V_1D_1 + V_2D_2 + \cdots}{nD}$$

$$D = \frac{D_1 + D_2 + \cdots + D}{n}$$

$$T = \frac{AD}{C}$$

Area of influence of a borehole is the area in which values shown hole are assumed to persist. Theory of averaging samples in general on assumption that values vary at a uniform rate between sample point assumption is met by so taking the area of influence of a hole that eve within it is nearer to that hole than to any other.

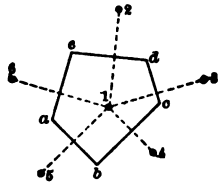

Fig 79

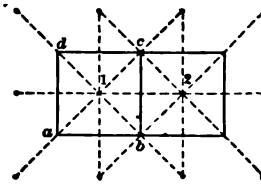

Fig 80

Fig 79 shows general case for holes spaced irregularly. Area of influence for is found by drawing lines 1–2, 1–3, etc, connecting hole No 1 with all those Lines *ab*, *bc*, *cd*, *de*, and *e* pendicular bisectors of lines 1–3, 1–2, and 1–6, res They enclose polygon *abcde* area of influence of hole N 80 shows same construction to holes spaced on corners o In this case, area of influen hole is a square (*abcd* for h the side of which is the between holes.

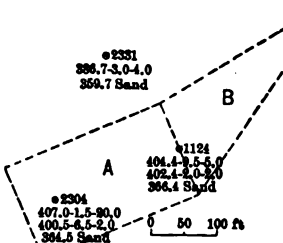

Fig 81 (*Trans* A I M E, Vol 48, p 40)

Practice in S E M

The following practice Federal Lead Co is quot H. A. Guess (55).

Fig 81 shows part of area in which drilling has a number of pay holes and the orebody has been platted. Portion A, area = 2 ft = 37 500 sq ft; portion B, area = 352 × 66 ft = 23 232 sq ft; total = 60 7 In calculating thickness and grade of orebody, each hole is first considered sepa-

	Elev	Thick-ness, ft	Pb, %	Ft × %
	407.0	1.5	20.0	30.0
	400.5	6.5	2.0	13.0
and aver		8.0	5.4	43.0
	404.4	9.5	5.0	47.5
	402.4	2.0	2.0	4.0
and aver		11.5	4.5	51.5
	403.2	5.0	4.5	22.5
and aver		5.0	4.5	22.5

considered poor ground and pillars.

ver %, Pb	Ft × %
5.4	43.2
4.5	51.8
4.5	22.5
4.8	117.5
4.8	

Experience shows that this method gives reliable results in this district.

Mesabi estimates. (Data and drawings from J. F. Wolff, *Mines & Min*, Feb, 1909.) Boreholes, topography, property lines, etc, are first oordinate lines at 100-ft intervals. From ions, at right angles to each other, are con-

l of , cu ft	Vol of ore, cu ft = 50% of vol of body	Tons ore, 11 cu ft to ton	Tons Pb
002	249 001	22 637	1 087

Analyses of samples placed on sections tline the layers of different grades of ore.

al Mesabi Orebody

, on which 5 E-W sections and 7 N-S sec-tion *FF* of this orebody (sample analyses is made, showing superposed contours of

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surface and top and bottom of orebody. Usually, extension of ore bey hole in any section is arbitrarily set at a distance equal to depth of ore hole, and bottom of ore is assumed to be a line joining this point with of hole. The 2 sets of sections are checked against each other, and fro limits of orebody are outlined in plan (Fig 82). Three preliminary e are then made as follows:

(*a*) **Total tonnage** is computed from area of orebody and its aver depth. for wedge shape of edges of orebody, the line marked "limit of area for total ore (Fig 82) is drawn half-way up the slopes on margins of the orebody. Area en this line × aver depth of ore ÷ cu ft per ton gives total tonnage. Areas are measu planimeter, aver depth of ore is computed from borehole data, and cu ft per ton mated from tests and experience.

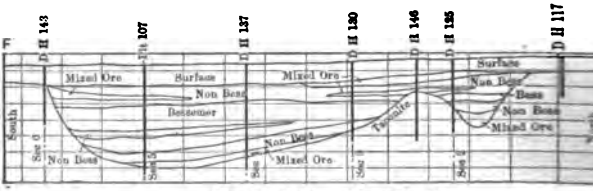

Fig 83. Section *FF* on Fig 82

(*b*) **Stripping estimate.** Proper BERM distance is laid out between edge of and toe of the bank or overburden; slopes of 1 : 1 or 0.75 : 1 are selected (depe character of surface drift), and intersection of this slope with surface contour is Aver area to be stripped (limit of area, stripping estimate, Fig 82) is taken as a way up this slope. This aver area × aver depth of stripping ÷ 27 = cu yd of

(*c*) **Possible steam-shovel tonnage.** With the aid of sections and combine map, a system of tracks is laid out for operating shovels and ore trains. This gi and size of deepest possible shovel excavation. Elevation at which each boreh be cut by this excavation is noted, and aver depth of ore taken out thus determin aver depth × area previously determined (total tonnage estimate) ÷ vol per ton tonnage used for preliminary estimates. Final or detailed estimates are made a

(*d*) **Graded-tonnage estimates** for securing relative tonnages of Bessem Bessemer and mixed ore. Two methods are used. In COMPOSITE-HOLE METH depth of ore of, say, Bessemer grade is first computed as arithmetical aver of Bessemer ore cut in all holes within area of orebody. If a hole shows no Besse it is included in the aver as zero. This aver depth × area (total tonnage estimat per ton gives total tonnage of Bessemer ore. Same calculation is made for othe Second and PREFERABLE METHOD is shown by Fig 82. From cross-sections, the of different grades of ore are worked out and plotted in plan, and planimeter ments are made of the area of each grade at each horizon. From these areas respective aver depths, volume of ore in each grade is computed. Plotting the different grades is intricate work. In the example, the taconite horse, shown i *FF* (Fig 83), cuts out roughly circular areas from 2 ore layers. Such areas are and subtracted from total area of the respective grades in which they occur.

(*e*) **Graded-tonnage estimates and aver analysis.** Tonnage estimates are the second method under (*d*). Aver analysis of ore in a grade is obtained by a drill-hole samples in proportion to their length (Eq 1 above). Products of sample per cent are locally called FOOT UNITS, which are computed for Fe, P, SiO_2, CaO, and S, or for such of these elements as are desired. "If for each ore-bear foot units are computed for each grade and these results added together, the respective total foot units for the grades in the orebody. Such sums are divide spective total feet of samples to give average analysis for each grade of the o TON UNITS is the name given to the products obtained by multiplying tons of a by its aver analysis (in Fe, P, etc). Aver analysis of all grades in the deposit is th ton units for each grade divided by total tonnage of deposit (Eq 2 above).

mmonly spaced on corners of 200-ft squares. tical cross-sections are constructed at right outlined on each. Usually there is some A. J. Sale (60) recommends that sections es (*AG*, *BF*, *DJ*, *EI*, etc, Fig 84), instead of etc), thereby increasing distance apart of ections closer together. Diagonal sections eir points of intersect- make sections more th irregular outlines. le through orebody at lesirable, for detecting ation of borehole data. done, sections do not d so that they outline r orebodies of simple tonnage can then be pths D_1, D_2 . . . , etc,

Fig 84

ly better to make planimeter measurements se the prismoidal formula (Sec 36, Art 11) te sections as middle areas. Aver value of e of ore in each hole is first found by Eq 1. r value of ore in a section is obtained by portion to their depths. Where holes are ould be combined in proportion to the area of the section. For equidistant sections, Σ r values) $\div \Sigma$ (section areas) = aver value e spaced at unequal distances, they should volumes of influence. Estimates are made ould check each other closely. Usually no ension of ore beyond last hole in any section.

holes were on corners of 200-ft squares. Ton- were calculated separately, using data from 4 llowing example of calculation for a block having he area of rectangle formed by these holes being

	× %
	59.30
	41.70
	65.00
	09.60
	75.60
	153.75 ft.
	1 038 169
	% Cu.

Fig 85

gave total l calcula- ntire ton- ock assays eminated gradually In such rely represent limits of profitable ore; minimum methods of mining and milling. Total tonnage

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in orebody increases as lower-grade ore is included in it. A. J. Sale (60) gives a cou method of analyzing these variables. Before averages are made, assays of drill s are corrected for mill recovery, corrected values being called NET PER CENTS. Sets tions are then constructed to include on margins of orebody ore having minim assays of say 1, 1.5, 2, 2.5, and 3% Cu. Estimates of tonnage and aver value (ne grade) are made for each minimum net % used, and results plotted as shown in Curve of net production, in lb of copper, is computed from known points on cu tonnage and net mean grade. Interpolations made on these curves show at once th of including ore of any minimum net grade between 1 and 3%. This device is of limited to the special conditions outlined above.

13. Exploration by Shafts, Tunnels and Drifts

Underground exploration is undertaken where conditions prevent s work, or where surface exploration gives no information as to underlyin bodies. Art 5 contains suggestions as to location of openings with resp geology. General rules : (*a*) Keep workings in the orebody; (*b*) Do firs on best showings, to see whether or not they are superficial; (*c*) Do the as cheaply as possible on account of the high risk involved.

Narrow veins are explored as in Fig 86. On each oreshoot, a small has been sunk, following sinuosities of vein (see cross-sec *AB*), and drifts are each shoot. Depth at which drifting starts in such work is from 50 to 100

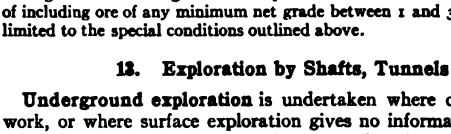

Fig 86. Typical Exploratory Work

depends largely on of surface alteratic level of ground wat is influenced by so local factors that general statements made.

Good showings in th may be explored with *R* or winzes *W*. In lo

shoots, raises may be put up at regular intervals. Workings are sampled and est made of tonnage and value of ore exposed (Sec 25, Art 12 to 15) as work progresse different times these figures, together with geological conditions, form a basis for juc as to whether: (*a*) prospect should be abandoned; (*b*) further exploration is war and, if so, its amount and character; (*c*) net value and amount of ore proved and w probabilities for extensions are sufficient to warrant systematic development and ment for the property. Further exploration would consist of deepening shafts and other drifts at intervals of 60 to 100 ft. WIDE VEINS are explored by same methods, but crosscuts are driven from wall to wall at regular intervals to det width, character, and value of ore.

Veins outcropping across a ravine may be explored with drift tunnel raises (Fig 87). Tunneling in general is cheaper than shaft sinking, an workings drain themselves.

Fig 87. Longitudinal Section in Plane of V

With STEEP TOPOGRAPHY there is great temptation to drive crosscut tunnels to intersect an orebody in depth (Fig 5, Art 1), lateral exploration being done by drifts, raises, and winzes as before. This involves a higher risk than where workings follow the orebody; the latter may be cut in a pinch and not be recognized; faults, and changes in dip or strike may cau tunnel to miss the orebody. Large amounts of water occurring near surface may use of crosscut tunnels for early exploration. Difficulties of such work should be nized; tunnels should not be far apart and there should be money enough to co work, if orebody is not cut at point calculated from its surface dip.

by drift tunnels and crosscuts. Vertical
eposits, and masses which do not outcrop.
are located on corners of squares (Art 3, 6, 7).
ploration usually consists of one or more vertical
crosscuts are driven to outline the orebody at
apping above determine vertical extent of ore.

opment. There is no sharp line between
her begins. Drifts and other openings for
ction. Drilling is done by hand, and wind-
penditure for plant is not made until justi-
l speed of driving small drifts, tunnels, and
afts, see Sec 7; also Ajo, Art 11. Factors
gs are same as given under Development,
l locate exploratory openings with reference
his should be subordinated to the primary
rt 1.

on are to reduce mining risk, lessen cost of devel-
btaining information upon which intelligent plans
quickest methods of obtaining this information
les. But, in many cases, underground explor-
d. (*c*) Underground work at first should be so
at expense of making special excavations of no
d exploration should extend far enough in depth
it below the zone of surface action, or to prove
ltered mineral for mining operations. (*d*) Ex-
ld be pushed far enough to determine the char-
ess and in richness, and to obtain some idea of
s and their relations to areas of barren and un-
gs made in the deposit, as shafts and drifts, will
of irregular and uncertain character, openings
large areas of barren or unworkable ground to
cuts or boreholes should be driven at intervals,
sures, or to prove the deposit itself if wide or
time to time on carefully taken aver samples,
ed by working tests on lots of representative ore.
be pushed far enough to make sure that there is
permanent plant, and far enough to determine
he enterprise. On the other hand, unnecessary
avoided. H. S. Munroe (64).

ply for Prospecting and Exploration

t of tools and supplies for 4 men doing sur-
7) (24):

el	2 gold pans
	1 tub
uger	1 pocket lens [ogist's)
	1 compass (surveyor's or geol-
chisel	100 lb mixed 5½-in and 7-in nails
arrel	25 lb 3½-in nails
forge	1 claw hammer
ill steels	50 ft 1-in hemp rope
ck (hammer)	1 hoisting bucket
spoon	1 windlass
	1 diaphragm pump, 2½-in suction
	1 combined anvil and vise

-in lumber for general purposes; 1½-in lumber
, fuse and caps for, say, 20 shots; ⅜-in round
utfit for 4 men; 1 month's provisions.

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Underground exploration, Montana. R. H. Sales furnished follo of supplies and equipment for a crew consisting of a foreman, 4 min cook, working in Flathead Co, Montana, from Dec 1, 1913, to May In that time they drove 555 ft of tunnels and crosscuts, and sank a 4 by 32 ft, using a windlass for hoisting. Rock was soft porphyry; no wate countered and no timbering required. Wages: foreman, $5; miners o $3.50; on shaft, $4. Two small frame shacks were built, each requiri 1 000 bd ft of lumber. Total cost, including labor, supplies, and eq was about $5 500. The list was made by a man of wide experience in sort of work in the northwest and includes no unnecessary items.

Mining tools and supplies

1 prospecting hammer
2 4-lb striking hammers
1 7-lb " hammer
1 8-lb " "
3 18-in hammer handles
6 36-in " "
3 long-handled round-point shovels
3 5-lb drift picks
4 4½-lb " "
12 pick handles
1 all-steel wheelbarrow
200 lb ⅞-in drill steel
5 lb ⅜-in mild steel
19 lb ⅝-in round iron
600 lb 40% gelatine dynamite
200 lb 60% " "
4200 ft fuse
1100 6 X caps
3 boxes candles
50 ft ⅝-in manila rope
1 5-lb wedge
1 7-lb "

Total cost, $212.82

Mine car and rails

Small supplies

1 keg 10d nails
2 lb 3d "
10 lb 8d "
5 lb 20d "
30 lb 40d "
2 pair 4 by 4 steel butts
2 door locks
4 ¼ by 1¼-in carriage bolts
15 gal kerosene
1 qt Boston coach oil

Total cost, $10.11

Miscellaneous

1 whiskey barrel
4 dinner buckets
1 lantern
1 No 8 Admiral stove
1 No 1 c Columbia heater
1 No 2 c Columbia heater
14 joints 6-in stove pipe
2 joints 6-in stove pipe, with check draft and dampers
1 tent 12 by 14 ft, 12 oz
1 No 1 galvanized-iron tub
1 zinc washboard
1 Keystone clothesline
2 6-in tent flanges
3 pkg ⅝ brass shoe nails
4 " ⅝ C H Hung nails
1 cobbler set
3 pair half soles
3 " heel lifts
1 wash bowl
1 8 by 10-in mirror
1 8-in scrub brush
1 yd 48-in canvas
1 square 2 ply Ruberoid roofing
1 ½-pitch tin roof jack
2 yd 36-in oil cloth
25 bars common soap
10 bars hand soap

Total cost, $58.39

Medicine chest
Blankets

Blacksmith's tools and supp

1 pair No 1 blacksmith tongs
1 pair G & D tongs
1 cold cutter
1 40-lb blacksmith vise
1 No 2 blacksmith hamme
1 70-lb anvil
1 No 400 blower
1 hardy
1 hot cutter
600 lb blacksmith coal
1 bottom swage
1 10-in flat file
3 6-in taper files
3 8-in M B files
2 10-in M B files
1 metal worker's crayon
1 10-in wrench

Total cost, $41.85

Carpenter's tools

1 No 3 hand axe
2 Jennings bits, ½ and 1-in
1 saw set, No 12
1 10-in wood rasp
1 4-ft, 1-man Simonds saw
1 6-ft crosscut saw and extra ha
1 4½-D B axe

3 preserve kettles and covers
2 large saucepans and covers
1 coffee pot
1 tea pot
1 tea kettle
1 pitcher
1 dipper
1 20-in butcher saw
1 10-in butcher knife
1 12-in butcher steel
1 kitchen knife
1 cake turner
1 cast griddle
1 can opener
1 coffee grinder
1 12-qt galvanized iron pail
2 14-qt " " "
12 10-in tin pie plates
12 each, plates, cups, saucers, soup bowls, teaspoons, tablespoons
24 knives and forks
1 1-pt syrup pitcher
Total cost, $20.30

blankets, mine car, and rails, $366.85.

ıs, calculated to have sufficient food value
hy, varies from 3.3 to 4.4 lb per man per
amps is often 6 to 7 lb per man-day, differ-
. E. Woodbridge has found by experience
uitable for remote regions and properly pro-
s are in lb per man per month; wt, 3.3 lb
iful, cut down on ham and bacon, double
nd evaporated vegetables.

b		Lb
	Butter	3
	Canned milk	2
	Cheese	2
2	Tea, coffee, and chocolate	2
	Salt, pepper, celery salt, mustard, two-thirds salt	3
7	Baking powder, if self-raising flour is not used	1
5		
4	Bottle of lime juice	

Commerce publishes following list of sup-
o Chisana; it covers food for 1 man for

	50 lb	Syrup	2 gal
	15 "	Jams and jellies	6 cans
	8 cans	Eagle milk	1 case
	4 "	Cream	1 "
sausage	6 "	Evap prunes	25 lb
	4 "	Cinnamon	¼ lb
e	3 "	Evap peaches	20 lb
	2 bricks	" apples	20 "
	10 lb	" apricots	20 "
	60 "	" figs	15 "
	6 "	Seeded raisins	15 "
	100 "	Coffee	30 "
ar	10 "	Tea	4 "
	1 gal		

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Ground chocolate or cocoa	8 lb	Mince meat	6 pkgs	Sauerkraut	1
Sweet chocolate	10 cakes	Tapioca	3 lb	Cabbage	1
Pepper	½ lb	Sago	1 "	Salt	3
Nutmeg	¼ lb	Corn starch	4 "	Naphtha soap	1
Mustard	¼ lb	Vanilla	4 oz	Tar soap	6
Baking powder	12 lb	Lemon	4 "	Ivory soap	3
Baking soda	2 lb	Evap vinegar	1 bot	Matches	1
Yeast	10 pkgs	Beef extract	8 oz	Candles	1
Macaroni	12 "	Evap eggs	8 cans	Sweet pickles	1
		Tomatoes	1 doz cans	Worcester sauce	2

G. A. Packard gives following list of food consumed per week by 10 me cook doing exploration work on Coronation Island, S E Alaska. brought fresh beef every 2 weeks. Cost per man-day (1904), $0.65.

Fresh beef	25 lb	Parsnips	3 lb	Graham flour	
Dried beef	2 "	Beets	10 "	Cornstarch	
Bacon	4 "	Beans	1 "	Rice	
Mess beef (corned)	8 "	Lettuce	½ crate	Hominy	
Pork	8 "	Pumpkin	1 gal	Gelatine	
Roast mutton	2 cans	Potatoes	125 lb	Baking powder	
Lunch tongue	1 can	Onions	10 "	Crackers	
Ham	25 lb	Peaches	3 qt cans	Cheese	
Pigs' feet	½ keg	Assorted fruits	9 qt	Yeastfoam	
Codfish	2 lb	Pears	1 gal	Molasses	
Oysters	4 cans	Evaporated apples	2 lb	Syrup	
Eggs	19 doz	Cherries	1 gal	Lard	
Tomatoes	1 gal can	Dried figs	5 lb	Butter	
"	1 qt can	Carnation cream	25 cans	Deer (or beef)	
String beans	4 cans	Sugar	30 lb	Pickles	
Lima beans	3 lb	Coffee	7 "	Spices, vinegar, e	
Peas	2 cans	Tea	1 "	tracts, raisins, cu	
Cabbage	3 lb	Flour	100 "	rants	
Corn	3 cans	Rolled oats	10 "		

At Iron Mt, Idaho, 20 men, including cooks, consumed following during 4 winter months. Fresh vegetables gave out and canned goo used. Double the amount of cabbage, turnips, parsnips, and one-ha onions and carrots should have been provided.

Fresh beef	2 383 lb	Salt	105 lb	Molasses	2
Fresh pork	581 "	Dried peaches	30 "	Jelly	3
Fresh mutton	167 "	Dried apples	65 "	Vinegar	6
Fresh fish	100 "	Dried apricots	50 "	Baking powder	8 la
Fresh chickens	112 "	Dried prunes	100 "	Pickles	1
Fresh eggs	69 doz	Raisins	150 "	Lard	2
Case eggs	6 cases	Condensed milk	23 cases	Catsup	2
Ham	472 lb	Canned corn	3 "	Tea	3
Bacon	258 "	Canned tomatoes	5 "	Chocolate	4
Butter	330 "	Canned peas	5 "	Coconnut	6
Flour	29 sacks	Canned peaches	1 case	Soda	2
Graham flour	50 lb	Canned pears	1 "	Yeast foam	2
Corn meal	2½ sacks	Canned pumpkin	1 "	Cornstarch	6
Coffee	322 lb	Canned oysters	1 "	Chowchow	2
Potatoes	40 sacks	Maple syrup	3 cases	Pepper sauce	4
Carrots	200 lb	Crackers	4 "	Currants	5
Turnips	100 "	Macaroni	3 "	Hominy	2
Cabbage	200 "	Cheese	77 lb	Matches	3
Onions	248 "	Sugar	9 sacks	G. S. soap	2
Parsnips	100 "	Oatmeal	3 "	Tar soap	2
Apples	600 "	Beans	1½ sacks	Ivory soap	2

gravel in Guatemala (work very heavy, kly food consumption per man as follows:

.....	0.5 lb	Coffee.....	0.3 lb
.....	5.0 "	Salt.....	1.0 "
.....	2.1 "	Baking powder....	0.17"
.....	1.3 "	Beef, live wt....	10 to 15 "

abor; beef, salted and dried immediately after y. (See also *Trans* A I M E, Vol 29, p 157.)

OPMENT

General

s: (*a*) to provide openings for stoping and rther and more detailed information as

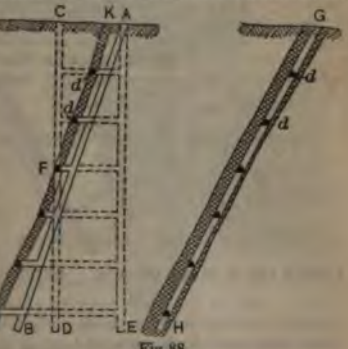

Fig 88

are (Fig 88): a vertical shaft *CD* started aft *AE*; a footwall inclined shaft *AB*; or an inclined shaft *GH*, in the vein. Except in the last case, crosscuts are necessary at intervals to reach the vein (Art 20). A vertical shaft is the correct mode of entry for FLAT OR VERTICAL DEPOSITS, lying under flat topography (Fig 89, 90). In MOUNTAINOUS REGIONS, entry to

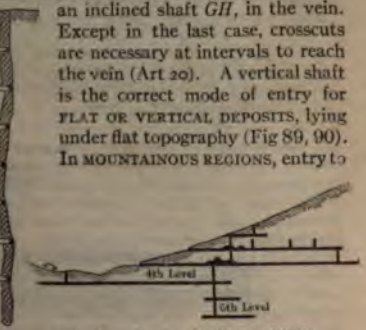

Fig 91. Section in Plane of Vein

be made by crosscut tunnels (*p*, Fig 5) or entry to a massive orebody, by both verti-

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cal and inclined shafts. It is common to find 2 or more openings of
or different kinds on a single orebody (Art 16 to 19).

Fig 92. Chandler Mine Mich (after Leith)

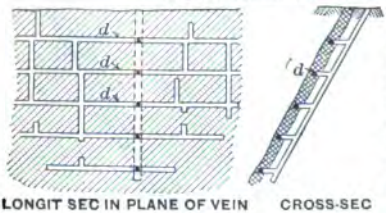

Fig 93

Lateral developme
91 and 93 show typical
VEINS, independent of
entry. Drifts *d*, in the
connected by raises foll
vein from level to level
Development in THIN
similar; the ore is divi
blocks by drifts roughly
angles to one another.
shows development in
DEPOSIT, 10 to 25 ft thick; drifts outline the orebody in plan, raises giv
tical extent, and both are used later in connection with stoping. Fig

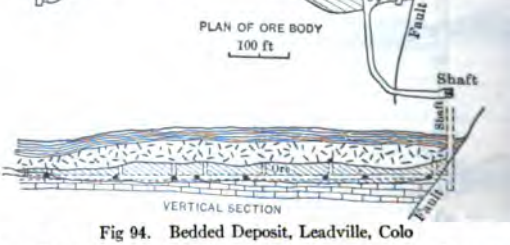

Fig 94. Bedded Deposit, Leadville, Colo

somewhat similar development of an irregular massive deposit (66).
of mining exert a dominant influence on mode of lateral development,

In general, lateral development divides corners of which afford numerous points of gs i-

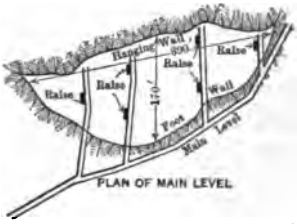

Fig 95. Massive Deposit, Bingham, Utah

numerous or sudden they increase hoisting niting hoisting speed. A straight incline r dip; at some point it must extend into in such orebodies the footwall location large shaft capacity. For small output, l the exploratory value of a shaft in the n fairly regular deposits, large outputs r location; choice is then based on other

3). The native copper occurs in conglomerate of district is 37° to 42°. Operators differ as GES OF FOOTWALL SHAFT: (*a*) Bins for loading and for the most part are cut in ore; this ad- ct from cars, as is done in many mines. (*b*) footwall shaft than for a shaft in the vein; is prevented by substitution of concrete and location obviates necessity for shaft pillars, unk in ore, and which tie up large amounts of may throw great pressures onto shaft pillars, shaft. DISADVANTAGES OF FOOTWALL SHAFT: rosscuts and stations cut in rock; (*c*) it pro- ormation as to character of the deposit. Some those in the vein and *vice versa*, depending on afts have been sunk in the lode. It is an open ft should not be sunk in the vein, and lateral e the orebody. The working shaft could then ls, and placed in vein or footwall in the light

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Tunnels *vs* shafts. Drift and crosscut tunnels can be driven fas cheaper than shafts. Wet ground increases this advantage, as a tunne the overlying ore by gravity, and eliminates pumping. Hoisting plant not required until operations extend below the tunnel level.

Crosscut tunnels prospect the country rock and may disclose para posits (Art 5). Drift tunnel is usually preferable, if a choice exists involved in driving crosscut tunnels is higher than for other method try; it increases with length of tunnel and its depth below known ore. unwise to begin development with a long crosscut before exploration ha an adequate tonnage of ore, and either proved or given strong geologi dence of its extension to the tunnel level.

Several long and deep crosscut tunnels have been driven, to provide drainage nomical transportation for groups of mines. They are justified when the sa known orebodies will amortize cost of tunnel at a profit during life of properties

Drift and crosscut tunnels serve mainly for extraction of ore lying above them must be sunk from tunnel level or from surface to develop deeper portions of a dep C. Hoover (20) points out that internal shafts require large excavations to accommod ing plant; also the latter must be operated by electricity or compressed air. Wh power is generated at the mine, saving is effected by placing the hoist on surface boiler plant. Combined cost of tramming and transferring ore to tunnel cars more than cost of hoisting the extra distance to surface. Questions of conver handling ore on surface may modify choice of method. Tunnels preserve their function after mining above them has ceased; surface water may be intercepte tunnel level, and the head on pumps at lower levels is reduced by difference in between collar of shaft and tunnel level. Saving in pumping may warrant the new and deeper adit, or the driving of a tunnel to tap workings already opened by even though the tunnel is used for drainage purposes only. Length of such a tunn it will pay to drive = present value of annual saving in pumping cost during prol of mine ÷ cost per ft of tunnel; for example, with power @ $60 per hp-yr, cost o ing @ $20 per ft, and a 10-yr life, it will pay to drive 600 ft of tunnel per 100 000 per day per 100 ft head saved.

Vertical *vs* inclined shaft. A typical example requiring a decision the two kinds of shaft arises in case of a pitching vein, as in Fig 88. An shaft is generally sunk in footwall (AB). Vertical shaft may be at CD either location requires more crosscutting than the inclined shaft, and fr standpoint the inclined shaft has an advantage which varies with dip of with depth. A greater length of incline is necessary to reach any level; this compensating advantage of the vertical shaft increases as the di flattens. Shaft AE always requires more crosscutting than CD; for total length of crosscuts is a minimum when $CF = FD$. Depth CD mus sumed in comparing merits of alternative locations. Factors to be conside

First cost. Vertical shafts cost more or less per ft than inclined shafts, acc local conditions and the way in which ground breaks. Sometimes drill holes in o other can be placed to take advantage of planes of weakness, thereby increasing s decreasing cost. Sinking very flat inclines resembles drifting; they usually cost ft than a vertical shaft of same cross-sec (see Sec 7). Cost per ft of crosscutting is same for either mode of entry. A comparison of total costs is made by appl costs to total footage of the different openings involved. For given depth and u of sinking and crosscutting, an angle of dip will be found at which the cost of han shaft CD (Fig 88) with its crosscuts is same as that of footwall shaft AB with its c on flatter dips, an inclined shaft entry is cheaper, and *vice versa*. Critical angle cases is about 70°. For all dips permitting a choice, a footwall vertical shaft A crosscuts cost most.

Cost of equipment. To reach a given level, a vertical shaft requires less hoist piping, wiring and timber, than an incline. Cost of surface plant is related to shaft in a very general way only; it should be determined for each alternative.

cal shafts is generally cheaper than in inclined
ven level is less; hoisting ropes last longer; there
s, and skip wheels, axles and boxes are trouble-
ded. As against this, aver length of tram to
mming may more than compensate extra dis-
Comparisons of operating cost can be made in
utlined above.

the skip limits rope speed in inclined shafts to a
possible only with straight shafts, good rolling
in well constructed vertical shafts has reached
potential capacity for a given cross-sec. Large
ut material increase in cross-sec of shaft, and
oisting speed. Since the track supports a por-
ven output may not be greater than in a vertical
ls at higher speed.

(Fig 88) requires pillars of ore for its support
or over thick orebodies, even if pillars are left,
om consideration and limit choice to shafts *AE*
erations. Usually vertical shafts cost less for
so, it is more difficult to set timber and keep it
hibits the use of inclined shafts.
Aich, a vertical shaft was sunk in the footwall
rebody, 68° to 70°; at depth of 2 400 ft, cross-
erence in maintenance charges in favor of verti-
ed cost of crosscutting and tramming (69).

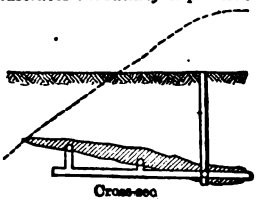
f the foregoing factors (Art 17). Deep vertical
ve same disadvantages as long crosscut tunnels.

vertical shaft may increase the time necessary
is important in planning development in small
deep-level projects (Art 17).

Methods of Development

commonly affecting both mode of entry and
that development methods are a compromise
nditions often predominate in final choice.

inous regions that a tunnel entry secures
t. Fig 96 illustrates the futility of positive
less than
with topog-
an incline


Cross-sec
Fig 96

ence in de-
t openings
dominates
nd mode of
anger and
nes, water-

d, which would increase maintenance cost.
ical staff; by cooperating with the mining
otage of development work required, and
e highest exploratory value and avoid bad

hows a single large oreshoot having a flat
position *AB*, length of drifts through bar-
h. Inclined shafts *CD*, if sunk in or under

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the orebody at an angle to the dip, will reduce amount of dead work; b
of this plan are often disappointing, because of the usual irregularit
shoots. Fr
standpoints, a
the two meth
ilar to that be
tical and incli
(Art 16).

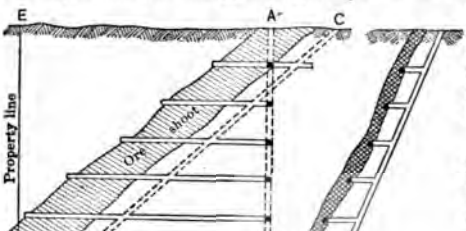


Fig 97

Fig 98 is an
such work. Th
lay in a narr
fissure, pitchin
a series of c
and amygdaloi
passing into gr
the north. D
years after disc
tical shafts w
sively sunk to
level. Incline
was sunk in the vein to shorten lateral haulage to shaft No 4, and to avoid sink
vertical shaft through greenstone where fissure was barren and maintenance c
also reduced surface tramming distance. The incline, which was begun when s
4 were about 500 ft deep, bottomed the shoot at between 700 and 800 ft; it was
doned and the shoot followed with shafts 2 and 4.

In case of a pitching oreshoot passing through the end line of a property, as at *EF* (Fig 97), the problem of entry on the adjoining ground is like that presented by deep-level mines. Position of oreshoots largely determines position of raises, as they are kept in ore to provide points of attack and ventilation for stopes (Art 20).

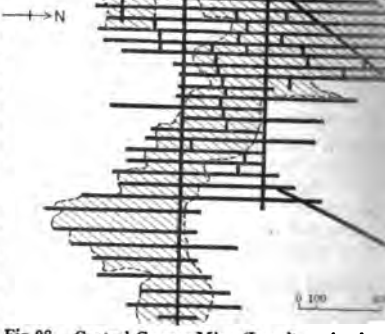
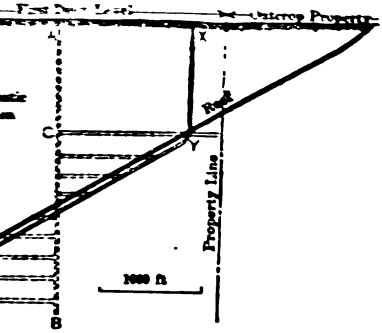


Fig 98. Central Copper Mine (Longit sec in plane

Deep mines (deep-level mines). Peculiar development problems arise where mining rights terminate at vertical planes passed through surface property lines. Fig 99 illustrates conditions on the Rand. Gold-bearing beds, locally called REEFS, extend depths. Dips vary greatly; in central Rand, aver dip at outcrop is 50 flattening to about 30° in depth. Properties worked from the outcrop a OUTCROP MINES; those covering adjacent extensions on the dip are k first row, second row, etc, of DEEP-LEVEL MINES. There are 3 modes of a deep-level property underlain by an inclined deposit: (*a*) A vertical sunk to the reef near the rise boundary, and continued as an incline in wall, the 2 parts of the shaft being joined by a carefully designed curv TURNED-VERTICAL or compound shaft (*XYZ*, Fig 99). (*b*) A CENTRAL V SHAFT *AB* is sunk, and the reef reached by crosscuts. (*c*) An arrangem

inclined parts of shaft are not directly con-
ne independent underground hoist, delivers



tcrop and Deep-level Mines, Witwatersrand

p a deep-level property. Other advantages, as
e: saving in crosscuts and raises, and ultimate
utility of shaft to any depth. Disadvantages:
cost of extra length of inclined section; slower
hoisting, due to necessity for slowing down at
curve; greater wear and tear on incline and on
ropes, especially around the turn. From stand-
point of first cost, choice is based on comparison
of inclined shaft *YZ* (Fig 99) and vertical shaft
CB; result is governed by dip (Art 16); the flat
dips of the Rand obviously favor inclined shafts
(20). Some engineers favor the central vertical
shaft for all cases (for further detail and differ-
ent points of view, see Bib 30, 70, 72, 73).

Factors of reduced speed and increased wear on hoisting rope, etc, become serious as depth increases, and, combined with mechanical difficulties in deep hoisting, have led Rand engineers to favor stage hoisting for depths over 3 000 ft. If proper storage is provided at transfer point, stage of a shaft, since both parts of it can be hoist-
osts of hoisting reported for turned-vertical
ormer; this is to be expected unless depth is
ystem fully utilized. A turned-vertical shaft
s was done in the case shown by Fig 100.

vel Shafts in 1900 (Leggett, 70)

	Number of shafts	Aver distance from outcrop, ft	Ultimate depths, ft		Aver ultimate depth, ft
			from	to	
.	54	1 730	430	2 505	1 189
..	21	4 162	1 920	4 100	2 611
.	3	8 113	4 000	5 000	4 500

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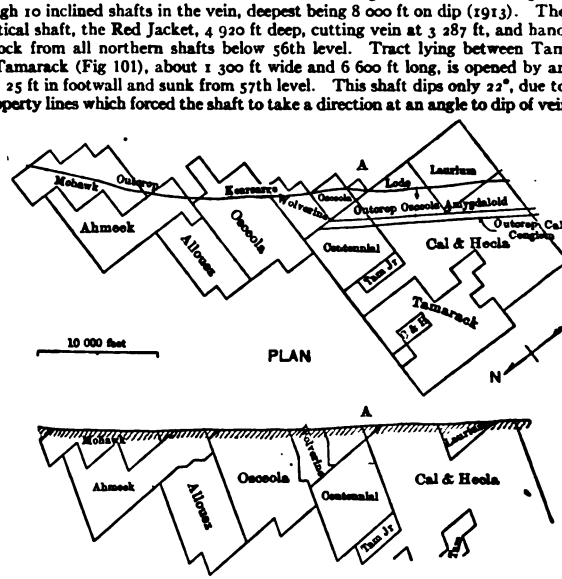
Michigan copper region. Fig 101 shows approx property lines of a group covering outcrops of Calumet conglomerate and Osceola and Kearsarge amygda Dip of beds, 37° to 38°.

Tamarack Co works the underlay of the Calumet conglomerate through 5 vertic ranging in depth from 3 409 to 5 308 ft (1913). No 5, the deepest, cuts the vein ft; bottom of shaft is over 1 600 ft horizontally from the vein. From the bottom of No 3 shaft (depth 5 253 ft) an incline in the vein is operated by a compressed- This was to save deepening the shaft and driving long crosscuts, which have high nance cost. Calumet & Hecla Co has worked this conglomerate bed for length through 10 inclined shafts in the vein, deepest being 8 000 ft on dip (1913). The a vertical shaft, the Red Jacket, 4 920 ft deep, cutting vein at 3 287 ft, and hand per rock from all northern shafts below 56th level. Tract lying between Tam and Tamarack (Fig 101), about 1 300 ft wide and 6 600 ft long, is opened by an shaft, 25 ft in footwall and sunk from 57th level. This shaft dips only 22°, due to of property lines which forced the shaft to take a direction at an angle to dip of vein



PROJECTION ON PLANE OF KEARSARGE LODE

Fig 101

cars are hoisted in this shaft and transported mechanically 1 500 ft on 57th leve Jacket shaft. This plan saved a very deep and costly vertical shaft.

Lower half of Fig 101 is a projection of property lines on plane of Kearsarge ve outcrops as shown and dips 34° to 42°. Ahmeek Co opened its property with 4 of which are sunk in the vein from outcrop. To develop northern part of propert outcrop is owned by Mohawk Co, 2 shafts were sunk, starting on angle of 80°; they curve on a 400-ft radius and enter the Kearsarge vein at 1 275 ft, on angl There are 3 levels, reached by crosscuts, above point of intersection of shaft a These 2 Ahmeek shafts start on surface close together; but diverge to N and W

Allouez Mining Co has also sunk 2 turned shafts to Kearsarge lode, starting fro at angles of about 80°. Depths to change of angle are 1 435 ft and 2 307 ft; tot 3 544 and 3 407 ft (1915). Both can be sunk to 9 400 ft before reaching pro (Fig 101). Tamarack, Ahmeek, and Allouez mines are controlled by Calumet Co. The turned shafts were sunk in the light of experience gained from the dee rack and Red Jacket vertical shafts and their long crosscuts.

Centennial mine has a small tract *A* (Fig 101), covering outcrop of Kearsage a right of way 100 ft wide connecting it with the large underlay property shown. these conditions a novel mode of entry was adopted. Two inclined shafts were in the outcrop, close together; southerly shaft runs straight down the dip; other is

ere it curves 15° to the north in plane of vein. t reverse at each end to keep the rope on idlers, The hoisting rope has shown greater wear than on ment is satisfactory (67).

ine, Cal, has a different purpose. A gold-quarts l shaft. Later a 1 630-ft vertical shaft was sunk ence continued as an incline to 5 850 ft, or 2 200 haul to surface by 2 370 ft, and its first cost is maintenance cost of a long incline through old is kind. Red Jacket shaft, of Calumet & Hecla s an auxiliary opening with similar purpose. It d maintenance cost of a 5 400-ft inclined shaft.

n inclined deposit, the upper parts of which are Choice is between vertical, turned-vertical, and depth to virgin ground and dip of deposit. Mt ne numerous factors to be considered. Orebody containing a concentration or vein of magnetite 24 to 40 ft; aver dip, 60° to 65°; country rock, o depth of 700 to 900 ft vertically. In 1914, it otwall shaft (dip 63°), 25 to 50 ft below deposit. tended ultimately to 1 800 or 2 400 ft. Reasons l ore pockets for a vertical shaft, because of the ng in the lean mineralized footwall zone; free- lue to mining operations; a vertical shaft to de- e 100 ft lower than, and 1 000 ft distant from, shaft would deliver ore direct to mill (74).

To secure economical handling, haulage t or change direction by easy curves, and mportance of these points varies with size ulage. Following are extreme cases:

Gilpin Co, Colo, the tonnage per shift from any id drifts properly follow the ore with little regard Rand, a main haulage level, 14.5 ft wide and ex- les), connects 2 shafts at a vertical depth of 2 200 all. It is planned for electric haulage to handle aulage ways are to be driven as required at inter- enings are the regular drifts on the reef. Similar ew Modderfontein and Modder B mines (75).

at orebodies, influence location of openings. rebody at its lowest point, all grades in the led cars. A location favorable for lateral , and vice versa. Methods of handling in levelopment are also interrelated (Art 20).

ment work can be planned to reduce pump- 6). Drainage also makes it desirable to the lowest point of flat-dipping or basin- en drain toward shaft and auxiliary sumps unction of lateral development is to drain es, like some Michigan iron deposits, this ent work done in advance of mining. New drain before extraction begins. This also ake pumping operations fairly uniform (76).

levelopment openings be run in duplicate. complete control of ventilating currents oints of entry are often required for venti- lation affects chiefly the location of raises lems until great depths are reached.

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18. Number of Openings

Determining factors. VENTILATION AND SAFETY demand at openings to surface; they are required by law in many districts. In meta stopes reaching the outcrop often afford an adequate second opening; lieries, rigid ventilation requirements compel at least 2 openings. Othe tions to be considered: (*a*) Required OUTPUT may be in excess of capac single shaft; this is unusual at ordinary depths, as a shaft can be desi handle large tonnages. In deep shafts, the time required to get men on shift through a single shaft is serious; this, together with time for h timber and supplies, greatly reduces ore-handling capacity. (*b*) SEPARA BODIES may require separate shafts; depending on their size, distance depth below surface, and surface conditions. Separate shafts are sunk t a saving over cost of entry from, and handling through, another shaft away. This is a matter of estimate in each case; size of orebody concern be sufficient to return excess cost of separate shaft out of the saving e Cost of shaft sinking increases with depth, that of drifts and crosscuts de hence, increasing depth should reduce the number of shafts and increa area served by each. Topographic or other conditions affecting surface port may prohibit separate openings; conversely, they may show a saving face over underground transport which alone will justify a separate shaft. statements apply also to veins in which several oreshoots occur, separa wide barren or low-grade areas. H. C. Hoover states that if cost per ft c sinking is 4 times that of drifting, through 1 000 ft of barren vein cost n than 1 shaft 1 000 ft deep (20). (*c*) OD OF UNDERGROUND HAULAGE. F is a diagrammatic section in plane inclined vein in a property of consid length. If ore occurs scattered thro

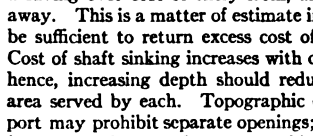

Fig 102

whole vein, question arises whether it is better to sink 1 central shaft *A*, or 2 *B* and *C*, placed at quarter points, or several shafts. This is determined by method of haulage on levels. Cost of hand tramming per ton increases with distance; there is a limiting length of tram beyond which a new sha save its cost; economic limits for animal and mechanical haulage are Tonnage produced per shift on 1 level affects method of haulage (Sec 11). small, it may be possible to install mechanical haulage on every second or third level, ore from levels above being transferred to the haulage level through winzes at distances apart which are within the economic limit of hand tramming.

Table 13. Hand Tramming in Mich Copper

Name of mine	Aver length of tram, ft	T tra per per
Ahmeek	563	1
Allouez	753	1
Cal & Hecla, Osceola	582	1
" " " conglomerate	171	1
Centennial	900	1
Isle Royale	750	1
North Kearsarge	702	1
South Kearsarge	444	1
Osceola	850	1
Superior	480	1
Tamarack	508	1

Michigan copper mines are illustrations of properties covering long distances along the strike of persistent lodes (Fig 101). Hand tramming was formerly universal, and a shaft was sunk about every 1 000 ft along strike. There are instances of 2 shafts 600 ft apart reaching depths of over 3 000 ft. Table 13 shows aver hand ming distance in Calumet & Hecla and allied properties, from Jan 1 to June 30, 191

conditions; they also vary because some loading is
g. Recent shafts are farther apart; aver distance for
cy mine, with electric haulage, aver tram is 1 800 ft.

ere length of property along strike is less than 1 500
vas sunk; on longer properties 2 shafts were deemed
ıd for ventilation (25). Transvaal law requires 2
properties are as close as 1 000 ft, but usually 2 000
and, but labor is cheap. Many deep-level proper-
large areas (400 to 900 claims of 1.4 acres each);
between shafts is 3 000 to 4 000 ft (78). Occasion-
with adjoining property, large areas were opened
concentrated on a few levels has been introduced as
d (Art 17). Brakpan mine, East Rand, illustrates
to a flat orebody of large area; property is 6 000
shafts were sunk 4 400 ft apart in direction of dip;
707 ft. They are connected at bottom by an in-
which levels are opened (79).

ND HAULAGE, ideal location for a single shaft
ning distance. It is obviously impossible to
advance. In massive orebodies like the por-
ape of which are determined by boring in ad-
of the deposit can be found, and the nearest
int gives shortest aver tram.

ıtion of Openings

ing location of openings with respect to dip
andpoints of geology, maintenance, haulage,
ssed in Art 17, 18, 20.

PHY affects location of openings. Sites for
lable; shaft and tunnel locations are often
ermanent mine openings should avoid gulches
sult from cloud-bursts; in mountain regions,
reference to possible slides of rock or snow.
stions of SURFACE TRANSPORT. Lowest loca-
s desirable, because it gives the highest backs
determined by position of a R R for shipping
fecting the position of a spur track to be built
ation must also be planned with reference to
on roads, or other means of transport; this
nine as well as ore from it. PROPERTY LINES
affect location of entry (Art 17, Deep mines).
hey may limit dump room below a proposed

e delivered from a mine opening at a point high
ity transfer to surface transport. Such elevation
of entry planned to secure it in connection with
ible; a compromise is always necessary.

urnished in part by Prof. H. S. Munroe)

ned orebodies varies from 50 to 300 ft; com-
f custom, is 100 ft, but practice tends towards

levels: COST. Drifting is NARROW WORK, cost-
drift requires timbering, track, pipe, and ditches,

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which must be maintained during life of stopes above. Cost of shaft stations, a
or bins, can be reduced by concentrating haulage on alternate or third levels.
standpoint of first cost, a long interval between levels is desirable. TYPE OF O
High-grade, spotted or pockety mines require levels to be close together to avoid
orebodies; a wide interval in such cases also adds to cost of reaching and ex
scattered pockets. SUPPORT OF HANGING WALL. Levels sometimes form one side
left to support roof; allowable area of unsupported hanging wall then limits level
SPEED OF STOPING and character of ground are related factors; level interval sh
such that stopes are completed and abandoned within the time that they can be k
without undue maintenance cost. Thus, at Leonard mine, Butte, Mont, a 200-ft
in vertical, filled, square-set stopes increased cost and decreased speed of mining
upper 100-ft lift to an extent which warranted driving intermediate levels at 100-
vals (80). METHOD OF MINING. A retreating system, where stopes are started at
lines and abandoned as they are carried back toward entry, may allow a larger lev
val than an advancing system. Cost of maintenance of levels themselves gene
creases with the time they are kept open. This is again related to speed of sto
method of mining, and may determine the max interval. In small-scale work, on
high-grade oreshoot, the time required to stope one lift determines time available
ing and drifting through the shoot on next level, and consequently determines lev
val; such conditions arise in leasing. DIP OF OREBODY, if over 40° to 45°, allows
ore to fall by gravity to level below; if less than 10°, cars can generally be run
ing faces. In these two cases, the dip does not limit max level interval. On int
ate dips, if ore is shoveled to drift, the level interval should be small; for economy
of distance with other handling devices in flat stopes, see Art 92. REGULARITY OF
influences handling methods and therefore the level interval; an irregular footwall i
the limiting angle on which ore slides and always requires some shoveling; badly
deposits require levels placed to reach displaced blocks regardless of other factors

Economic level interval in any mine is a matter of experiment. The
levels are usually spaced farther apart than the upper, because early work
mines the mining factors, and reduces the exploratory function of these op

On the Rand the level interval in some properties has been increased to a s
extent. At Modderfontein B mine, ore is developed in blocks 800 by 1 000 ft;
Modderfontein, level interval is 500 to 600 ft, with footwall main haulage drifts
2 000 ft apart; at Brakpan mine, level interval varies from 300 to 1 200 ft. This
due to an attempt to cut down development cost, has not been uniformly successfu
made possible in these mines by: (*a*) very flat dips (7° to 10°), allowing cars to run
faces; (*b*) regularity of vein; (*c*) knowledge of reef characteristics, resulting in co
in the continuance of value and size over large areas (75, 79, 81, 82).

Raises and winzes. For following indications or exposures of ore up o
from a level, the location of raises and winzes depends entirely on ore occu
To determine outline and value of shoots, raises are often made at fairly
intervals. For this purpose the interval should not exceed that over wh
may be expected to persist without material change in thickness, charact
tenor. Closer spacing than the level interval is not justified; max distanc
to 500 ft or more (see Modderfontein B mine, above); it is a matter of exp
with a given orebody. In connection with method of mining, raises and
provide entrance to stopes; serve for handling ore, waste rock for filling, a
plies; are usually the openings from which stoping starts. For these pu
their location varies greatly (see individual mining methods). In veins wh
occurrence is regular, the output required, combined with distance to which
opment has been carried in advance of stoping, sometimes determines th
interval. Each stope has a limited tonnage possibility; the number of
that can be opened simultaneously depends on available points of attac
general extent of development.

West Australian practice. E. D. Cleland states: Where development is consi
in advance of ore requirements, the winzes (or raises) in lowest level may be 500 t
apart, and are sunk merely as a guide to future stoping. In upper levels where

or winze interval must be 125 to 150 ft. This
ccurrence; where ore is irregular, no figures can
that in deep-level Rand mines, where develop-
re 400 to 500 ft apart; throughout the district,
o the limit fixed by loss of interest on money
the number of winzes (that is, the greater the
f development.

levels are essential where natural venti-
o circulate air currents.

fice for ventilation; sometimes ventilation re-
ample, delays while waiting for smoke to clear
t of drifting; with natural ventilation, delays
holed through to level above. On important
interval independent of ventilation; but, with
nit beyond which either increased cost or slow-
. This limit, controlled largely by local factors
ore.

and Crosscutting

s, mountings, and explosives; comparison
of charging and firing; mucking; driving

n tunnels in following respects: (*a*) they are
) less emphasis is laid on speed of driving; (*c*)
be taken of relative softness of orebodies and
ss due to banding, slips, or well-marked walls;
than ordinary drifts and crosscuts, and there-
ound; (*e*) long tunnels usually bear entire cost
illing, ventilation, transport, etc; larger plants
hich are distributed over many openings. For
are much lower than those for long tunnels.

TS is same as that of tunnels (Sec 6). UN-
have an arched back, which tends toward
palling off. Importance
and narrow openings; in
nging wall often forms the
ot broken into (Fig 103).
like Fig 9, Sec 6) varies
nd method of mining (Fig

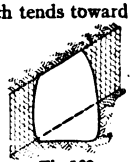

Fig 103

rifts and crosscuts is given in Tables 15, 16,
: (*a*) In EXPLORATION (Art 13), cross-sec
y by reducing amount of muck to be han-
to 6.5 ft high by 3.5 to 4 ft wide. Smaller
e cramped space reduces efficiency of labor.
width of development drifts and crosscuts
for hand tramming, 24 to 28 in wide and
inimum clear width of 4 to 4.5 ft for single
rger cars, cross-sec of opening is designed
ar and posts of drift sets is usually 12 in
ween car and lagging, or room enough for
t. For safety in untimbered openings, at
and wall on one side of drift; clearance on
inimum clearance between cars on double-
s are always desirable, especially for high

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speed and heavy cars. Shovelers need a total width of about 7 ft to both sides of a small car at face of a drift. (*c*) DRAINAGE DITCHES for large amounts of water may increase width of drift or crosscut beyond quired by car (Sec 6, Fig 12, 13). (*d*) Requirements for VENTILATION may determine minimum size of opening. Design of cross-sec must room for ventilating pipes, if used (Sec 6, Fig 11). (*e*) In veins up to 8 wide, drifts in oreshoots are often carried full width of vein, thus reduc per ton of ore obtained. (*f*) HEIGHT OF DRIFTS and crosscuts is a matter room; 6.5 ft clear is about the minimum.

Drifting and crosscutting by hand drilling is done in connecti exploratory work (Art 13) in small mines, where expense for plant is n fied, and in districts where skilled labor is not available for running drills. Drill holes are not spaced systematically, but placed to take ad of all irregularities in the face. Drilling is done double-hand in hard gro single-hand in aver ground; hand auger is cheapest and best for soft containing few hard streaks, and in coal.

In small drifts, 7 to 12 holes, 2 to 2.5 ft deep, are necessary to advance face 1. These figures vary widely with hardness and toughness of rock; in aver ground an of 0.7 to 1 ft per shift in a 1-man drift or crosscut is good work.

Examples. At BODIE, Cal, 1 miner drilling single-hand in 5 by 7-ft drifts and c width of veins 3 to 30 in, vein rock varying from hard banded to soft granula country rock firm, fairly hard andesite, advances 1.304 ft in drifts and 1.187 ft in per 8-hr shift (84). At RHYOLITE, Nev, 1 miner drilling single-hand in drifts 4.5 7 ft, in fairly soft mineralized porphyry, makes 0.5 to 3 ft per 8-hr shift, aver 1 laying, mucking, and tramming was done by other men). Table 14 is record of driven by hand in ARIZ (1913); cross-sec, 5 by 7 ft; length, 374.5 ft; aver tram 875 ft; rock, fairly soft porphyry; no timbering required; 3-shift contract work by labor, 1 miner and 1 helper on each shift doing drilling, blasting, mucking, and tr Tunnel was completed in 500 man-shifts in 91 days. Aver advance per man-shift varying from 0.359 to 0.852 ft; aver progress per day, 4.12 ft; aver wages per man-shi

Table 14. Record of Crosscut Tunnel in Arizona (D. D. Irwin)

Item	Per linear ft		Per cu ft	
	Amount	Cost	Amount	Cost
Labor per contract	.....	$4.09	.....	$0.117
Dynamite (40% Hercules)	2.32 lb	0.383	0.066 lb	0.0109
Fuse	11.7 ft	0.047	0.336 ft	0.0013
Caps (5X)	5.34	0.045	0.152	0.0013
Candles	2.96	0.079	0.085	0.0023
Ventilation (gasolene and oil)	.....	0.088	.....	0.0025
Track (8-lb rail)	1 ft	0.209	.....	0.0060
Totals		$4.941		$0.1413

Table 15 gives data for Mexican labor (1901). Daily advance (3 shifts) was f progress per man-shift low. Ventilation was poor in timbered drifts No 3 and 4. mite consumption was low, due to character of rock and size of cross-sec.

A. L. Oke gives following data on hand work by native labor at mines in Mex Argentine (86): MEXICAN MINE; double-hand drilling; ⅞ and ¾-in steel; long h 8-lb hammers; Mexican dynamite, 40% and 60%, depending on ground; dry, hot, ventilated mine; average ground tough, rather than hard; fairly clean hanging break to; 8-hr shifts. Average progress per man-shift in softer drifts, 0.76 to 0.9 hard ground, 0.31 ft. Maximum advance in drifting in a week of 11 shifts, 3 day and 1 men on night shift, 32 ft. Men drilled about 9 ft of holes per shift in soft 3 to 3.5 ft in hard ground. ARGENTINE MINE, single-hand drilling; ⅞-in steel;

g, Esperanza Mine, El Oro, Mex (85)

	Lb 60% dynamite		Labor per ft		Aver advance	
	Per lin ft	Per cu ft	Miner shifts	Peon shifts	Per man-shift, lin ft	Per man-hr, cu ft
1	1.34	0.038	3.2	0.14	0.3	1.31
7	2.07	0.059	3.3	0.567	0.26	1.14
8	1.09	0.019	....	....	....	....
1	3.08	0.088	3.35	0.62	0.25	1.09
2	3.35	0.096	....	....	....	....
8	0.43	0.012	....	....	....	....

rtz. (*c*) Soft swelling andesite. (*d*) Moder-

te and gelatine; dry, cool mine, poor ventila-
o 10 hours. Aver progress per man-shift in
.22 ft; in hard ground 0.37 to 0.55 ft. Usual
cluding Sundays, was 50 to 60 ft per month in
Men drilled about 10 ft of hole per shift in

Chiksan mines, CHOSEN, KOREA (87): Drifts
oreans) per shift, double-hand drilling. From
Two 10-hr shifts give cheaper work, but less
rd to 70 ft in good ground. Dynamite con-
ear ft. Cost per ft, excluding hoisting, about
Crosscuts, 4 by 6 ft, are untimbered; 2-shift
e consumption was about 2.2 lb per linear ft;
t per linear ft in soft ground, $2.67; in hard
; muckers and trammers, $0.20 per shift.
es, Minn, illustrates cheapness and speed ob-
nt of the softness of the hematite, few mines
ent of drill holes is fairly systematic. C. E.
Work is on contract; two 10-hr shifts; 2 men
, track-laying, and local tramming to distances
ers, 3.5, 4, 6, and 8 ft long; hard streaks are
steel. Main drifts are 9 ft wide by 8 ft, inside
d consists of 5 to 7 6-ft holes. A 6-ft hole is
ded with 7 to 10 sticks 40% dynamite, lifters
mucked first. This reduces powder cost, but
lear; where ventilation is poor, entire round is
to 125 ft; contract price, $3 to $4 per ft, in-
sh all supplies except timber. In wet ground,
e, contract price sometimes reaches $5 to $6.
oo to 225 ft per month on a 4-man contract.
o ft per month; contract prices, $9 to $12 per

n drifts and crosscuts is indicated by fol-
cter of labor, and hardness and toughness

ft, in weathered granite, $3 to $8 per ft; IDAHO,
tzite, $10; UTAH, prospect tunnels in weathered
, contract prices for 4 by 6.5-ft drifts in quartz
ces for drifts in narrow, weathered quartz vein,
ard quartz vein, 2.6 ft wide, dip 45°, wall rock
drifts in narrow replacement vein in andesite,
sericitized rhyolite (like dry kaolin, auger drill-

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Machine drilling, definitions. All types of rock drill are used for drifting cutting. To avoid confusion in this and other articles, terms are used with meaning: PISTON-DRILLS, in which the bit is attached to and reciprocates wi LEYNER DRILL, large hammer-type drill, mounted like a piston-drill. STO (stopers), small drills of the hammer type, with air-feed. MOUNTED STOPE-DR mer drills, mounted by attaching air-feed cylinder to a column or bar. PLUGG small hammer drills (like the Jackhamer), usually having a D-handle, and hel by the driller. (For details of drills, see Sec 15.)

Machine drilling; arrangement of holes. See Sec 6, Art 6.

Location and number of holes depends on several interrelated factors. O small cross-sec afford limited space for drills; length of drill bit then limits a holes in V- and pyramid-cuts and may force use of draw-cut; in some mines wi is increased in hard ground, to allow more efficient arrangement (89). Some drill faster than others in flat holes and uppers, or in certain kinds of ground holes should be planned to suit the drill available. Hard ground requires more soft, especially relievers between cut and squaring-up holes. In soft talcose tings may pack around the bit in wet holes, while dry holes give no trouble; then designed with as few down holes as possible. In general, more economica sults if total footage of hole required for a round is not greater than can be d shift, allowing time for blasting.

Rounds for PISTON and LEYNER drills are shown in Fig 104 to 108 (with Fig 3 to 8, Sec 6). Fig 104 (90) is typical for small arched-back

Fig 104

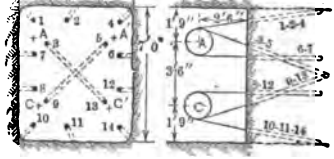

Fig 105. Pyramid Cut

Fig 106.

in soft ground; one more lifter is sometimes necessary. Fig 105 (91) is MID-CUT for small openings in hard ground. The drill is mounted on a column set up in middle of drift; the arm occupies successively positions *A*, *A'*, *C*, and *C'*; in each position, some holes are drilled from top and some from underneath the arm. In soft ground, holes 2 and 11 may sometimes be omitted. Leyner drill is well adapted for this round. Fig 106 (92) shows a DRAW-CUT made by piston drills in aver ground at Cripple Creek, Colo; size of drifts, 3.5 by 6.5 ft to 5 by 8 ft. Fig 107 shows V-CUT in a wider drift, at Homestake mine, So Dak. In drifting along a vein wall, or some other well defined plane of weakness, the SIDE-CUT (Fig 108) is advantageous; it is drilled from a single set-up of a column. Numbers of holes indicate order of firing.

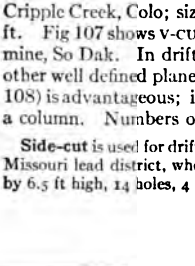

Fig 107. V-cut

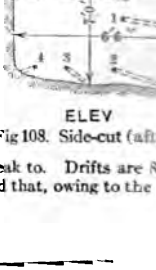

Fig 108. Side-cut (aft

Side-cut is used for drifting in flat-dipping dolomite in S E Missouri lead district, where there is no slip on one side to break to. Drifts are 8 by 6.5 ft high, 14 holes, 4 to 8 ft, making a round. It is stated that, owing to the

d consequently drill and break better (93) (Table
d in drifts and drift-stopes in Mich copper mines.

Fig 109. Inverted Draw-cut

fficient for flat or down holes, unless aided
neral, the cut holes can be placed more effi-
vith piston drills than with stopers, which
ke more holes in tough ground; but this is
greater drilling speed of stopers.

copper mine, Chile. S. B. Gordy furnishes Fig
ing arrangement of holes used for driving tunnels
, 8 by 8 ft cross-sec, through easy tuff and blocky
Holes 1 to 3 are drilled with stopers; the rest with
rs. This round has been found quite satisfactory;
oles 4 to 6 are drilled with stopers, but generally,
the flat holes, the Jackhamer is preferred, being
ace by hand (Ex 18, Table 16). A 4-hole round
same mine in loose easy ground, for driving 5 by
uts; the 2 uppers are drilled with stopers; the 2
h Jackhamers. Rounds similar to Fig 110 have
for driving short tunnel headings in the Penn-
nthracite fields.

stopers for drifting, some device is necessary to
a simple form, consisting of a sprag, wedged across
om floor, a loose
se for drill. An
by dotted lines).
s an inexpensive
ngles, from verti-
s turned down at
a plank; to the
to U shape (94).
. The sprag in
the air-feed of a
re. End of air-
rests in shallow
. Tail piece of
n link) fastened
sity for sprags of
ll may be held in
ft behind it (95).
h plugger drills,
even for drilling
the SHEPHERD'S-
ghtens work of

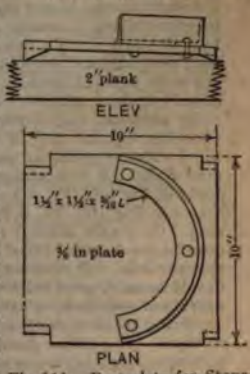

Fig 111. Foot-plate for Stope Drill in Drifting

mers, leasers rested the drill in a wooden trough
hole to be drilled. Drill is held to its work by a
tom of trough. In hard rock, requiring flat back

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Table 16. Dr

Example No	Location	Rock formation	Cross-section Height, ft	Cross-section Width, ft	Drills No	Drills Kind	Mounting	Kind of round
1	S E Missouri	*A*	6.5	8	2	2½ in *P**	*CO*	*SC*
2	Desloge Lead Co, Mo	*A*	6.5	11	2	*P**	*CO*	*VC*
3	St Joseph Lead Co, Mo	*B*	7	12	3	2¾ in *P*	*HB*	*VC*
4	Argonaut mine, Cal	(*e*)	8	7	1	2¾ in *P*	*HB*	*DC*
5	Goldfield Cons, Nev	*C*	7.5	5	1	2¼ in *P**	*HB*	*DC*
6	Cobalt, Ont	(*g*)	7	5.5	1	3⅛ in *P*	.....	.....
7	Leonard mine, Butte, Mont	*D*	8	7	1	3 to 3½ in *P*	*CO*	*DC*
8	Homestake mine, S D	*E*	8	7	1 or 2	3 in *P* or 3⅛ in *P*	*HB* & *CO*	*VC* & *PC*
9	" " "	*E*	8	12	2	3⅛ in *P*	*CO*	*PC*
10	Portland mine	*F*	7	4.5	1	2¼ in *P**	*CO*	*DC*
11	Cripple Creek, Colo	*F*	7.5	5.5	1	3⅛ in *P*	*CO*	*DC*
12	Alaska Treadwell	*G*	7	10	2	3⅝ in *P*	*CO* & *HB*	*PC*
13	Sunnyside mine (*q*)	*H*	7	5	1	*JK* (*p*)	.....	.....
14	Round Mtn, Nev (*r*)	*I*	7	5	1	2 in *P* (*s*)*	.....	.....
15	Melones Mining Co, Melones, Cal	*J*	7	6	1	3 in *P*	.....	.....
16		*J*	7	5	1	2¾ in *P**	.....	.....
17		*J*	7	6	1	3 in *P*	.....	.....
18	Ray Cons, Ariz	*K*	7	5.5	1	*LY**	*CO*	*IDC*
19	Isabella mine, Cripple Cr, Colo	*L*	7	4	1	*S*	.....	*IDC*
20	Braden Copp Co, Chile	*M*	8	8	1	*S* & *JK*	.....	*IDC*
21	Miami, Ariz	*N*	7	5	1	*P** or *S*(*n*)	*CO*	*DC*

* 1-man drill. † Average. *A* = Dolomite. *B* = Hard limestone. *C* = Dac drilling and breaking. *D* = Mineralized shear-zone in granite. *E* = Schist. *F* andesite breccia. *G* = Siliceous diorite and slate. *H* = Medium hard, tight *I* = Vein matter, altered rhyolite, soft and loose. *J* = Amphibolite schist, co irregular stringers of quartz; medium hard, fitchery; breaks well; on footwall is a banded quartz which is very difficult drilling. In Ex 15, two holes are drilled in vein, requiring use of 3-in drill; Ex 16, no holes are drilled in quartz and, as ground and easier to drill, Liteweight Sullivan drill can be used. In drifts, Ex 15 and 16, h drilled parallel to schistosity of rock; in crosscut, Ex 17, holes are at right angles tosity and break much better than in drifts. *K* = Hard, brittle schist. *L* = A phonolite, breccias, etc. *M* = Tuff and blocky andesite; see Fig 110. *N* = Sc tured schist, easy drilling and breaking. *P* = Piston drill. *LY* = Leyner drill Stope drill. *JK* = Jackhamer. *CO* = Vertical column. *HB* = Horizontal bar. Side-cut, Fig 108. *VC* = V-cut, Fig 107. *DC* = Draw-cut, Fig 106. *IDC* = I draw-cut, Fig 110. *PC* = Pyramid-cut. (*a*) 1 shift drilling; 1 mucking; 26 days. (*b*) 1 shift drilling; 1 mucking; 25 working days. (*c*) 25 days; 3-shift done for speed; round drilled in 6 hr; blasting in 1.5 hr; crew 3 drillers, 3 helpers, mers per shift. (*d*) Minimum. (*e*) 4 to 5.5 ft advance in schist and slate; 2.5 ft a in vein material, speed reduced by soft ground and necessity for close timbering; advance in greenstone, so hard that sometimes it took 2 shifts to drill a round. A per month of 26 working days showed similar variation. Crew, 1 driller, 1 he muckers. (*f*) 3 drifts; speed work; two 8-hr shifts; 1 driller and 1 man mucki tramming; day's-pay bonus system; 0.171 to 0.188 lb powder per cu ft. (*g*) McI Darragh-Savage mine. In conglomerate, 11 to 13 holes; average 27.5 ft hole drill shift; 0.212 lb powder per cu ft; average advance per mo (4 weeks of 13.5 shifts), In quartzite 10 to 12 holes; average 32.5 ft hole drilled per shift; 0.209 lb powder ft; average advance per mo, 92 ft. In slate, 9 to 10 holes; average 40 ft hole drill shift; 0.196 lb powder per cu ft; average advance per mo, 120 ft. (*h*) Labor per 1 driller, 1 mucker; timbermen lay track when necessary. (*i*) 2 drill-shifts requir

Advance per round, ft	Dynamite: Lb per cu ft	Dynamite: % Ngl	Advance, ft per month	Date	Bibliography
4 to 4.5	0.385	35	90(*a*)	1912	(93)
5.0	0.252	35	90–100 (*b*)	1912	(93)
6.06	0.134	60	455 (*c*)	1912	(100)
2 to 5.5 *e*	0.208†	40	130–260(*e*)	1910	(101)
3.5 to 4.3	0.182†*f*	40	208–260	1910	(102)
.....	0.206† (*g*)	60	76–120 (*g*)	1909	(103)
5.0	0.143†	40&60	.....	1913	(80)
5.0(*j*)	0.309	40	.....	1904	(104)
5.0(*j*)	0.285	40	.....	1904	(104)
.....	0.294	40	.....	1905	(96)
.....	0.374	40	.....	1905	(96)
5.0	.....	40&70	.....	1904	(105)
4.0	0.154	40	(*h*)	1916	(*y*)
3.5	0.180	40	(*h*)	1916	(*y*)
3.5	0.452	40	} (*i*)	1916	(*w*)
3.5	0.326	40			
4.0	0.396	40			
6 to 6.5	0.105	35&60	.....	1909	(42)
4.0	0.312	40	.....	1916	(*x*)
3.5 to 4	0.094	34&51	.....	1915	(*z*)
4.5 to 5	.....	....	166-214(*o*)	1909	...

s 15 and 17, crew per shift is 1 driller, 1 helper, driller per shift and 1 mucker alternate shifts. nd drilled in 4 hr. (*l*) Round drilled in 6 hr. mounted on a column gave about same results ng and blasting; one 8-hr shift; mucking done ated Jackhamer, ⅞-in hollow steel, water con- hese drills are being replaced (1916) by mounted n, Supt. (*x*) Data from C. G. Mitchell, Gen (*z*) Data from S. B. Gordy. COSTS PER FT. $5.10 to $6.91; 8, $8.23; 9, $11.68; 10, $6.40;

l stull wedged across drift about 4 ft back from l; 6 5-ft holes constituted a round, which was ved each other, so that drilling was continuous, (126).

5), used 15 Jackhamers in 8 by 7-ft drifts, pro- replacement deposit in shale and limestone and red 20 5.5-ft holes; aver output per driller per nuck. 1 machine man, working on a bonus, s (127).

ad crosscutting (see also Sec 6, Art 2). rs 2.5 to 2.75-in, 1-man drills, for aver for very hard rock.

compared performance of 3⅛ and 2¼-in piston ills operated under exactly similar conditions; an on the small. Result showed saving in total

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cost of 27% with small drill. Rate of advance with small drill was 10 to 20% with large drill (96) (Table 16, Examples 10, 11).

Mich copper mines in 1912 introduced 1-man drills and effected marked ec

Alaska-Treadwell mine. Water Leyner 1-man drills have replaced 3-25- drills. The small drill consumes 20% less air per min, but runs more steadily, so air consumption is about same as before. Saving in labor far more than offsets cost for hollow steel, drill sharpening, etc. 4 small drills on a cross-bar replaced 2 l in a crosscut 12 ft wide, with no increase in labor but greatly increased speed of adv For comparison of Leyner and piston-drills for horiz openings, see Sec 6, Art 2.

Mounted stope-drills may be used for drifting, for rounds like those drills. Mounting causes delays in setting up and shifting, but the light advantages (see following data from FRANKLIN FURNACE, N J).

Substitution in drifts of mounted stopers for large piston drills reduced dril from 18.4 to 5.7 man-hr per ft of advance and cost of explosive from $1.84 to $1. latter reduction is due to smaller diam of holes drilled and to flexibility and ease stopers which permit and encourage more efficient placing of holes. Cost for explosives was reduced from $5.33 to $2.70 per ft (98).

Use of stopers and pluggers for drifting and crosscutting was develop West largely by leasers, because of their lower initial and operating costs remarkably low costs have been obtained with them, and they bid fai place other machines in many places, especially in ground which drills an easily. Objection to drifting with stopers is that the muck must be 5 or 6 ft back from face before drilling can begin, also in dry ground t duce much dust unless sprays or water jets are used. For very soft grou ger drills, with automatic rotating device and auger bits, give excellent (See Colby mine, under Cost of drifting and crosscutting.)

Breaking ground in drifts and crosscuts. Data in Tables 16 and in a general way the effect of character of ground and size of cross-sec on and arrangement of holes, kind of drill used, and powder consumption. alizations are difficult, because of interdependent factors; the data indica bilities warranting careful experiment and analysis in planning rounds f openings. With piston and Leyner drills an aver of 1 hole is used per 4 face (= aver burden per hole of 2 to 3 ft). Examples 17 and 18, Table extremes in stope-drill practice in this respect.

Mucking and tramming. Speed of drifting and crosscutting is o termined by time required for mucking, rather than for drilling and If more than one heading is available and speed not essential, it is usually to alternate drilling and mucking crews at any one face, thus giving clean drift in which to set up, and allowing use of full shift for drilling. especially important if drilling is difficult, or if a large footage is requ round. General points in regard to mucking in tunnels (Sec 6, Art 13. apply here also, especially where speed is desired. Table 18, by R. K. gives mucking and tramming duty. A plat of sheet iron or plank, laid close to the face before blasting, materially increases output per mucker-hr on loading and tramming from chutes are included for comparison and f estimating on raises (Art 22; Sec 11, Art 9). Rate of mucking is also aff size of pieces; observations by author in Colo gave following data:

Car, 28 by 28 by 54 in; capacity, 2 500 lb; 1 man mucking small material floor filled car at rate of 3 tons per hr; with good floor, 5 to 6 tons per hr. I muck containing many big pieces, which had to be lifted into the car, 1 man load per hr; 2 men loading mixed material from rough floor and taking their time ha tons per man-hr. 2 men, 1 picking and 1 shoveling, made following rates per m large and small material mixed, 2.9 tons; all big pieces, 3.2 tons; all fine muck, 3.

Crosscutting Data (106)

	4	5	6	7	8
	7	6	5	10	9.5
.5	9	8	7	9	7
	4	1	3 (*a*)	3 (*d*)	4
	8	8	8	8	10
	6	6	5	6	4
'	3 in, *P*	*L*°	*P*°	*P*°	3¼ in, *P*
c)	1(*e*)	1	1 (*e*)	1(*e*)	2
.5	60	43	55	45	33(*i*)
	60	105(*f*)	30 (*g*)	20 to 60 (*h*)	40
	0.328	0.265	0.105	0.106	0.325
c)	4.8	21(*c*)	5.84	10.8	4.5
o	2.46	3.5	4.0	3.3	2.75
2	1911	1912	1912	1913	1911

round. 2. Cananea Consolidated, Mex, hard
rd quartzite. 4. Mammoth Copper Co, Cal,
ite. 6. Erie Consol Co, Cal, slate and quartz.
ed andesite. 8. Alaska-Treadwell, Douglas
ner, 1 shoveler. (*b*) 1 driller, 1 mucker. (*c*)
riller and 2 trammers on each shift; 1 timber-
time to clean out face. (*g*) 1.5 hr required in
takes about 10 min. (*i*) Footage drilled per
ner drill. °1-man drill (2¼ or 2½-in diam).

(55, 67, 106, 107, 108, 109, 110, 111, 122)

of	Distance trammed, ft	Cars per shift	L'gth of shift, hr	Tons per man-hr
	0	18	8	2.25
	2 400	16	8	2.00
	100	14	8	1.75
on	1 000	10	8	1.57
on	1 000	15	8	1.17(*d*)
ft	300	17	8	1.79(*c*)
on	0	....	10	1.60
ft	0	....	....	2.00
n	700	11	8	1.52
ft	600	10	8	0.88(*c*)
ft	275	70	10	0.72(*e*) (*c*)
..	1 000	....	8	0.60
..	.....	....	4.5to5	2.0 to 3.3
;	200	15(*f*)	8	3.09
:	0	....	....	3.00
	0	....	8	2.46
ft	300	20	8	2.11(*c*)
ft	100	16	8	2.05(*c*)
n	1 000	6	8	1.58
ft	600	11	8	0.98(*c*)
..	600	....	9	1.56 to 1.72
ft	150	80	8	10.32(*c*)
on	450	31.5	8	8.46
n	700	45	8	6.19
ft	300	40	8	4.22 (*c*)
ft	400	30	8	4.06(*c*)
on	1 500	20	8	3.12
	90	65	8	4.04

ted on basis that 20 cu ft = 1 ton. (*d*)
enough work to keep 2 men fully occup
r 2 months.

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Timbering in crosscuts is similar to that for tunnels (Sec 6, Art 17) generally lighter on account of smaller cross-sec and shorter life of these op This may be true also of drifts, but timbering in the latter is often desi support filling or ore in the stope (Art 36–38). Usual forms of drift tim

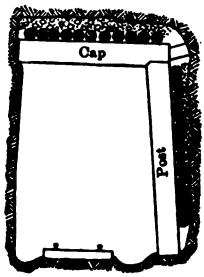

Fig 112. Half Set

(*a*) HALF SET (Fig 112), cap resting in a hitch end and on a post at the other; used where ba one side of drift require support; (*b*) THREE-Q SET (Fig 115, *A*) cap and 2 posts, used where ba both sides require support; is the commonest FULL SET (4-piece), made by adding a sill to th quarter set, is used where the floor is soft (Sec 6, Posts of sets usually have a batter of 1.5 to 2.5 ft of vertical height; increased where lateral p is heavy and sets must be maintained for a consi time (Fig 113). Round or sawed timber is u drift sets (for comparison, see Art 50; also Sec 16). SIZE OF TIMBERS. Round timber is usual 12 in diam; for heavy ground and large openings 24 in diam or more. Ordinarily, lightest sawed used is 6 by 6 in; sizes to 12 by 12 in are common. SPACING OF SETS d on size of timbers, weight of ground, and length to which available laggi cut without waste; usual interval is 4 to 6 ft; in heavier ground, 2 to 3 ft; treme cases, sets are placed skin to skin. LAGGING (*L*, Fig 113), used to prevent falls of ground between sets, is of round 4 to 6-in poles, half-round mill slabs, or 2 to 4-in plank. Plank is used where tight joints are necessary, and in districts where all timber is imported and plank is as cheap as other forms. Length of lagging usually = distance *c* to *c* of sets; it may cover only back of sets, or back and one or both sides; in rare cases sills are lagged also. La on back is usually placed skin to skin; on sides open lagging (Fig 113) is gen sufficient. Space between lagging and is best packed with broken rock. S (stretchers) (Fig 113, *S*) are dis pieces to brace the sets longitudinally; are heavy lagging poles, or 4 by 6 or 6-in timber, cut to fit between sets at between cap and post, and spiked in in shifting ground, feet of posts are b also. Sets are firmly blocked to blocking and lagging often form ade lateral bracing. JOINTS. Simple are desirable; object is to get full str of timbers with as little framing as pos

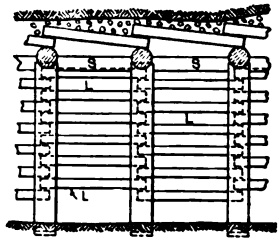

Fig 113

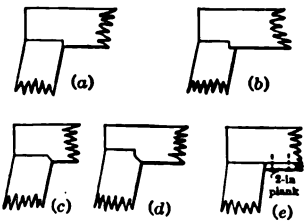

Fig 114. Joints Between Cap and Post

Fig 114 shows typical joints: (*a*) is cheapest to frame, but cap may split u vertical pressure; (*b*) avoids this danger; (*c*) is as good as (*b*) and cheap frame; (*d*) and (*e*) are designed to give both cap and post the strength o

rtical and lateral pressures are heavy. and sill in full sets.

be reinforced by diagonal braces between nless sets are high). In some cases a second

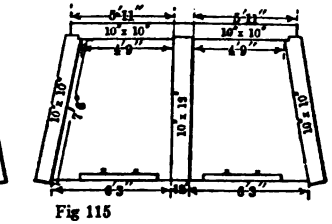

Fig 115

ee also Sec 21). At some mines, these imbering, mucking, and tramming; at g and general expense are added. Fol- tice (see also Drifting and crosscutting

ft, in ore on a contact vein in quartzite with y, timbered; mule-tram, 1 750 ft; hoisting hinemen, pipe and trackmen, $3.25; muck- lies: dynamite, 12 to 13.5¢ per lb; timber, o $23.50 per M. One large piston drill used. hifts (109): machinemen, $2.44; muckers, , $0.56; miscellaneous labor, $0.25; cost of lumber and timber, $1.28; hoisting, $1.54; cost per ft, $11.11.

h hard quartzite, dip 25° in direction of ad- od air; 3 shifts per 24 hr; two 3½-in piston a and 2 muckers per shift; each shift blasts;

	Laguna drift	Combination drift	Jumbo drift
driven, ft....	260	228	208
of 2 shifts....	30	27.5	29
vance per 8-hr	4.33	4.17	3.5
per shift.....	2	2	2
trammed, ft.	800	1 200	1 200
harge (*a*).....	$0.41	$0.37	$ 0.51
r.....	0.64	0.26	0.21
ives.....	1.00	0.93	1.02
.....	4.24	2.91	4.15
ng (*b*).....	0.28	0.63	1.02
st per ft.....	$6.57	$5.10	$6.91

ers compressed air, drill repairs, and lubri- er cost for 1909 = $1.83 per drill shift. (*b*) ck from drift was used as stope filling, thus oisting cost.

4; muckers, $3.75 per 8 hr. Accompanying x 5, for details; also Sec 21, Ex 2).

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Argonaut mine, Jackson, Cal (1909) (Table 16, Example 4). Cost of 741 ft cut, driven from shaft about 330 ft to vein and 50 ft into footwall, was: mac $2.34; carmen, $1.62; dynamite, $1.28; fuse, $0.11; caps, $0.03; candles, $0.08 $0.51; track, $0.18; total cost per ft, $6.15. Power cost, $4.50 per hp per month; dynamite, 11¢ per lb; wages, machinemen, $3; helpers, $2.75; carmen, $2.50. Part of work was contracted at $3.75 per ft for all labor(101).

Utah Copper Co	Drift (a) 7 by 7.5 ft	Dr 5 5 b
Drilling and mucking	$3.50 (c)	$
Powder, caps, and fuse	1.25	
Compressed air	0.30	
Miscellaneous	0.55	
Total	$5.60	$
Timbering where necessary	1.61	
Cost per ft	$7.21	$

Utah Copper Co, Bingham, Utah (1915). Drifts in monzonite-porphyry; breaking and mucking were contracted; company furnished supplies and paid for timbering; 8-hr shifts (122). Accompanying record gives cost per ft for 2 drifts:

(a) Motor-haulage drift; No 9 Water Leyner dri ress, 5 ft per round; cost per cu ft, 10.66¢. (tramming drift; 2.5-in piston drill; progress, 3.8 ft p cost per cu ft, 10.1¢. (c) Contract price, $3.50 to depending on whether dry or wet ground. (d) price, $2.25 to $2.50 per ft, depending on length

Portland Gold Mining Co	Cost pe
Machine-men and helpers	$2.65 $
Trammers	0.74
General tramming cost (b)	0.03
Pipe and track-men	0.06
Operating drills (c)	1.34
Powder, caps, and fuse	2.07
Pipe and track	0.41
Supplies (d)	0.03
Hoisting and gen'l expense (e)	1.46
Cost per ft	$8.79(A) $

Portland Gold Mining Co, Cripple Creek, Colo (1903). Two crosscuts were run to test relative merits of large and small piston drills (Table 16, Examples 10, 11). Machinemen, $4; helpers, $3.50; dynamite, 12.7¢ per lb; fuse, 0.35¢ per ft; caps, 0.7¢ each. Accompanying record covers 20-day run in each crosscut; 800 ft tram to shaft; no timber used; 2-shift work (96):

(A) Crosscut 5.5 by 7.5 ft; 3½-in piston drill; len en, 110.5 ft. (B) Crosscut 4.5 by 7 ft; 2.25-in pis length driven, 97.5 ft. (a) 1-man drill. (b) Repai and tracks; lubricants. (c) Cost per shift, 3½-in dr 2.25-in drill, $1.85; includes compressor cost, repairs air pipes, etc. (d) Picks and shovels. (e) Proportio of foremen and shift bosses, assaying, surveying, lighting, office, and general surface expense.

Homestake mine, Lead, So Dak (1904) (Table 16, Ex 8, 9, and Fig 107). Following record shows cost of work with large piston drills; dynamite, 12¢ per lb; miners, $3.50; helpers and shovelers, $3; 10-hr shift (104):

Homestake Mine	7 by 8-ft drift	8 by 12-ft
1 miner drilling	2 shifts $7.00	3 shifts $
1 helper "	2 " 6.00	3 "
1 miner blasting	1 " 3.50	1 "
1 helper "	1 " 3.00	1 "
1 shoveler	2 " 6.00	3 "
Explosives, caps, and fuse	10.75	
Blacksmith, drill repairs and machine shop	4.85	
Total	$41.10	$5
Cost per ft	8.23	1

Mineville, N Y (123). Drift, 8 by 9 ft; aver face of drift was 0.6 hard gneiss and 0.4 magnetite. Two 2.75-in piston drills on tripods drove a heading 7 ft wide by 8 ft high with 16 4.5-ft holes per round. A third drill, working 2 shifts per week, trimmed side of drift to full width with 8 to 10-ft holes. There were 2 10-hr shifts; round was drilled and each shift; 60% gelatine dynamite used in cut holes; 45% in side holes. 3 muck a shift did all shoveling and track laying, and trammed cars (2.5 ton, 22 cu ft) tances to a side track, where they were picked up by electric locomotive; sho shovels and no mucking plates were used. Drill runners, $2.50; helpers and muck

ıde at following cost per ft: drill runners, $2.40; er, $2.46; caps, $0.04; fuse, $0.12; air (aver for aver for preceding 7 mo), $0.19; ½ foreman's time @ $2.50, $0.39; total cost per ft, $10.68.

. G. O. Argall, mgr, furnishes following cost explosives, $1.091; air, $0.625; steel, $0.213; otal cost per ft, $5.927.

on Co, Ishpeming, Mich (124). Crosscut, 10 1914–15. Formation, slate, greywacke, and Leyners mounted on a horiz bar. Face was was drilled and shot section *A* and upper ke lower part of *B*; was parallel to slips Each shift blasted t squared at end of ers and (for most of ading a car while 3 ıe, 240 working days; 1.5 ft; aver per day, ft; poorest, 12.75 ft. fter which cars were

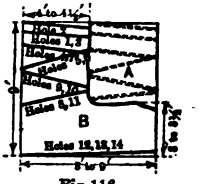

Fig 116

latter part of work a portable loading machine 25 to 30%. Artificial ventilation was afforded r min, operated by a 15-hp motor; 10-in air wing statement covers entire length:

bor	Supplies	Total	Per ft
30.27	$ 1 587.73	$10 018.00	$ 3.384
69.03	.....	8 169.03	2.760
17.92	498.39	1 616.31	0.546
.....	6 889.86	6 889.86	2.325
.....	720.00	720.00	0.244
60.00	230.00	690.00	0.233
35.50	1 196.27	1 331.77	0.451
.....	75.00	75.00	0.025
.....	95.00	95.00	0.032
12.72	$11 292.25	$29 604.97	$10.000
6.18	3.82		

riving 9 by 16-ft crosscuts, with 1-man drills, , $2.70; aver tons broken per drill per 2-shift $0.29; shoveling and tramming, $1.44; sup-

(1914). Drifts and crosscuts in mineralized oles per round; 1-man piston drill; 23 to 30 2.8 ft per 8-hr drill shift; 2-shift work. Most d track men, $3.50; muckers and trammers, hift, affording a clean set-up; timber rarely $1.25; labor, mucking and tramming, $0.90; ith, $0.22; explosive, $0.80; track (rails, ties, ; candles and lubricating oil, $0.09; power and coal and drill steel, $0.06; superintendence,

owing record covers 6 months' work in 8 drifts or details of conditions and supply costs see s):

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Table 19. Raising

Example No	Location	Character of ground	Cross-section			No of holes	Sq ft face pe hole
			Width, ft	Length, ft	Area, sq ft		
1	Bingham, Utah.....	*A*	4	6	24	6 to 9(*c*)	2.7 to
2	Fig 120.....	...	3.5	7	24.5	10(*a*)	2.4
3	Ohio Copper Co, Utah.	*B*	5	5	25	5	5
4	New Jersey Zinc Co (*b*)	*C*	5	5	25	17	1.5
5	Fig 118.....	...	4.5	6.5	29.25	12	2.4
6	Rand, So Africa.....	*D*	4.5	6	27	10	2.7
7	Nowata Lead & Zinc Co, Mo.....	*E*	5	7	35	14	2.5
8	Bingham, Utah.....	*A*	5	7	35	6 to 9	3.9 to 5.
9	Isabella Mine, Crip Cr, Col.....	*F*	4	9	36	16(*a*)	2.2
10	Golden Cross mine, Cal	*G*	6	8	48	6 to 9(*a*)	5.3 to
11	Treadwell mines, Alas.	*H*	6	8	48	18	2.7
12	Ouray, Col.....	...	5	11	55	8 to 9(*a*)	6.1 to 6
13	Negaunee mine, Mich.	*I*	8	8	64	24	2.7
14	Leonard mine, Butte, Mont.....	*J*	6.5	11.5	74.8	15 to 18	4.2 to
15	Granby Consol, B C..	*K*	8	10	80	18	4.4
16	Mother Lode mine, B C	*E*	8 to 10	12 to 20	96 to 200	18(*d*)	5.3 to 11

A = porphyry: (Ex 1) soft; (Ex 8) hard. *B* = quartzite. *C* = tough l *D* = banket. *E* = limestone. *F* = phonolite, andesite, breccia, etc. *G* = schist. *H* = siliceous diorite. *I* = jasper and schist. *J* = granite. *K* = br tuff. (*a*) See examples of practice in raising. (*b*) Fig 119, also Ex 9, Table 3.5 to 4-ft round. (*d*) Holes 6 to 7 ft deep in rows of 3. (*x*) Furnished by C. G.

charging. At beginning of shift, muck is cleaned off platform at top, and back down; planks are laid across chute from next to top stull, to prevent steel or t falling into chute; sometimes canvas is spread over these planks to make them long planks are then laid across top stulls extending over the chute; they are stulls at manway end, and, for safety, their ends should be wired down; an open between planks on manway side, through which miners can climb. One ma does this work in about 1.5 hr. The round is then drilled from top platform, in 2 to 2.5 hr; rest of shift is spent in cutting hitches, putting in stulls, lagging, ladders and air pipe. Before blasting, top stulls over manway are covered wi plank and blocks (Fig 120). Sometimes pockets are cut (*P*, Fig 120) at interval 40 ft, in which to store drill, tools, etc. One man doing all the work has raised month (this is above aver speed. Author). In a 1-man raise of this kind, depth and consequent advance per round is adjusted so there will be time to keep the t up. It is usually best to blast on each shift; hence, holes are limited to from 2 to

Fig 121 shows method of driving a small inclined raise (113). Up to dip of 4 ber is required to support men; at angles above 30°, it is best to place small hor or 5 ft apart and 6 to 10 in above footwall. Muck collects behind these and fo which aid in climbing raise and supporting drills; the steps also catch steel a dropped accidentally. Support for tail-piece of stope drill may be obtained fro brace, as in Fig 121. (At Braden Copper mine, Chile, 2 short pieces of drill ste into plug holes drilled in footwall, carry a plank placed on edge across the raise, turn gives footing for drill. No back holes are required, as the ground break Christensen gives following data for aver conditions in a 1-man raise: "To clean get ready to drill need not take over 1 hr; to put in round of 8 holes, averaging takes 1.5 to 2 hr; if a nipper brings powder to miner, it is possible to shoot tw making 4 ft per shift; otherwise, max daily aver advance expected is 2.5 ft." raises on dips of less than 35° to 40° require shoveling (Table 22, Ex 5).

Several raises connecting levels for ventila-
distance 113 ft; cross-sec of raise 6 by 8 ft.
level below. Dry ore with considerable fines
up and block chute; at 34°, large pieces jump
slope of 34° for first 10 ft, flattening gradually
ope drills; 6 to 9 holes per round, loaded with
2.5 ft. Distribution of cost per ft: labor,
96; labor, loading cars and tramming, $0.93;
blacksmith, $0.24; explosives, $0.80; chutes,
07; candles and lubricating oil, $0.12; power
cksmith coal and drill steel, $0.06; superin-
ver wages: drillers, $3.20 per 8-hr; muckers,
ost less per ft than track in drifts, and required
rilling was done on day shift; mucking into
ght shift, even with forced ventilation (116).

very long raise starts 5 500 ft from the portal
inclined height, 930 ft. Rock section of raise
o strike of vein. There are 2 compartments,
t 4 by 5 ft for ladders, slide for bucket and air
oses, with idea of enlarging it later for a shaft
ound of 8 or 9 holes, 5.5 ft deep, in an aver of
during rest of shift. Holes were loaded with
long enough to allow men to reach bottom of
consisted of a row of 12-in stulls, between
in stulls, on other side of manway, supported
necessary. Stulls were 5 ft apart and set in
es in soft ground were 20 in deep; no head
htly. Sides of 12-in stulls toward the chute
-in plank fastened with 60 d spikes. Timber-
d of heavy timbers laid across stulls prevented
as usually on top pair of stulls and never more
rly full of muck to reduce wear on lining, but,
o footwall caused serious clogs. At 450 ft a
; then only 50 ft of muck was kept in lower
n lining showed considerable wear, and it is
vith lag screws should be used for heights over
raise by a small compressed-air hoist; a 12-in
l on topmost stulls. Work was done in 2 8-hr
e air per min) ran continuously for ventilation.
shift; sometimes 2 extra men on day shift for
le was used for tramming, 1 man on day shift
compressor men, 1 cook, and 1 waiter; outside
work. Aver speed, 100 ft per month; max,
from beginning to end. Due to time required
were drilled, indicating 5 ft advance per round.

as described by C. T. Rice, is done exclusively
with fewer holes, but 2 stope drills can readily
r over that required for 1 piston drill; with 2
ter than with 1 piston drill. A V-cut is used
2 by 6.5 ft, to take 2 square sets of 10 by 10-in
and girtwise (Art 47, 49). Holes are placed
a stope-drill round. Aver time for raising and
men per shift, two 8-hr shifts. On rush work,
rilling, blasting and timbering is done by con-

G: Mitchell furnishes following data for aver
lite, andesite and breccias; Waugh No 12 and
holes (= 80 ft of hole) per round; 30 to 40 lb
und, 4.5 to 5 ft, with 1 drill shift.

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Table 20. Treasury Tunnel Raise, Ouray, Colo (115) (fire ins and taxes n

Labor and supplies	May		July		A
	Drove 100 ft (448–548 ft)		Drove 92.5 ft (653–745.5 ft)		Drov (745.
	Total cost	Cost per ft	Total cost	Cost per ft	Tota cost
Machinemen	$465.50	$4.65	$468.80	$5.06	$478.
Timbermen and helpers	618.70	6.19	434.01	4.70	561.
Tramming	113.50	1.14	87.30	0.94	110.
Explosives	118.42	1.18	76.95	0.83	83.
Candles and oil	23.61	0.23	20.05	0.21	23.
Timber & timber supplies	128.74	1.29	171.28	1.85	169.
Hauling expenses	23.72	0.24	36.14	0.40	36.
	$1 492.19	$14.92	$1 294.53	$13.99	$1 462.
Salaries, office expenses	213.72	2.14	222.31	2,40	205.
Foreman	158.00	1.58	159.20	1.72	162.
Compressor men	254.00	2.54	261.40	2.83	262.
Blacksmith and helper	121.00	1.21	217.77	2.34	129.
Coal (*a*)	1 066.13	10.66	584.19	6.32	441.
Compressor supplies	22.30	0.22	24.24	0.26	23.
Tools and shop supplies	5.70	0.05	3.38	0.04	4.
Maintenance or repairs (*b*)	4.58	0.05	20.18	0.22	147.
	$1 845.43	$18.45	$1 492.67	$16.13	$1 376
Employers liability	.....	.....	.....	.....	273.
Cost of board	40.37	0.40	142.03	1.54	125.
Total cost	$3 377.99	$33.78	$2 929.23	$31.66	$3 238.

(*a*) In May, cost of heating several buildings is included. (*b*) In August, sp cost $30, repairs to boiler plant $75, otherwise, all items are for supplies.

Round Mountain Mining Co, Nev. R. H. Ernest, Supt, furnishes foll for aver conditions in 1916: cross-sec, 5 by 8 ft; rock, medium hard, tight Ingersoll-Rand CC-11 stope drill; 12 5-ft holes per round; 22 lb 40% gelatin per round; advance, 4 ft per round, with 1 drill shift of 1 driller and 1 muck

Melones Mining Co, Melones, Cal. J. A. Fulton, Supt, furnishes followi 1916. Raises 6 by 7 ft; ground, amphibolite schist containing irregular quart medium hard, fitchery, but breaks well; on the footwall is a banded quartz v difficult drilling; Ingersoll-Rand BC-21 Butterfly stope drill; 14 holes (= 80 per round, giving 3.25 ft advance with 2 drill shifts; 51.5 lb of 40% dynamite crew, 2 drillers. A winze, 7 by 11 ft, is kept 2 levels ahead of shaft, crosscuts the shaft raised from them. Shaft sinking costs about $46 per ft for break and timbering. Raising and subsequent stripping down to full cross-sectio costs about $36 per ft for same items. In the winze, ground is like that in holes are drilled in footwall quartz vein, giving more trouble than all of the r hamer and 1-man piston drills had a thorough trial in winze, but difficult gro use of 2-man, 3-in piston drill. In the winze, 24 holes (154 ft) are drilled per r advance per round, 5 drill shifts, 4.5 ft; 120 lb of 40% dynamite used per rou per round, 4 drillers, 4 helpers, 3 muckers.

Raising through an old filled stope at Silver Plume, Colo (Fig 122). One was next to solid rock, but same plan was used where raise was entirely surround material. Filling was well packed, and would not run in large amounts. Orig to 4 ft wide; walls, fairly strong. Timbering consists of 3 rows of short stulls apart, set in hitches, and dividing raise into a manway and chute. 2-in plan to middle row of stulls, to form a tight partition between compartments. Pl used for lagging the outside spreaders. Only half the face of the raise is advan time, topmost stulls on other side being covered with planks to form a roof for of miner when at work. Excavation is done with a bar, loose material falling below. When face has been advanced about 2 ft, temporary stulls *a* are put

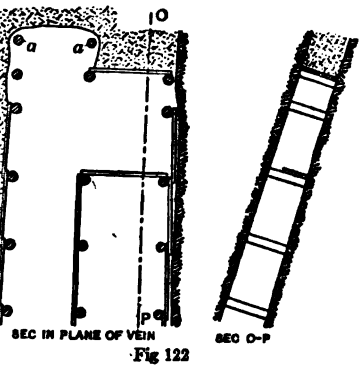

Fig 122

d Winzes (Hand-drilling)

advance, ft	Lb dynamite, 60% Ngl: per lin ft	Lb dynamite, 60% Ngl: per cu ft	Labor per ft: Miner-shifts	Labor per ft: Peon-shifts	Aver advance: per man-shift, lin ft	Aver advance: per man-hr, cu ft
04	2.78	0.042	4.81	0.93	0.17	1.43
31	1.75	0.027	6.69	0.82	0.13	1.10
57	3.36	0.061	....	....	....	....
04	5.56	0.086	5.27	0.79	0.16	1.36
33	1.95	0.039	7.30	0.48	0.13	0.80
74	3.30	0.050	7.17	1.13	0.12	0.99
94	1.20	0.018	10.71	1.66	0.08	0.67
76	4.46	0.068	5.23	1.21	0.16	1.21

drilling. (*a*) Friable quartz. (*b*) Fairly hard l quartz. (*c*) Moderately hard shale. (*f*) Hard fair in No 4 and 8; good in other openings.

(See also Examples of practice in raising)

3	4	5	6	7	8	9
4X5	6X12	6X10	5X7	17X4.5	6X7	5X5
1	2.33	3(*c*)	1	2(*d*)	2	1
8	8	8	8	8	10	8
5	6(*b*)	7(*f*)	5	6 to 7	7(*g*)	6(*i*)
S	*S*	*S*	*S*	*S*	...	*S*
1	2	1	1	1	2	1
....	0.234	0.250	0.126	0.069	0.276	0.272
12.5(*e*)	7.46	5.66	26.0	17.1	6.74	15.6(*e*)
5	1.93	2.27	6	3.6	3.21	5
1912	1911	1912	1912	1913	1911	1915

ard ground. 2. Cananea Consol, Mex, hard ium hard quartz. 4. Mammoth Copper Co, granite. 6. Erie Consol, Cal, slate and quartz. ilicified andesite. 8. Alaska-Treadwell mine, ugh crystalline limestone.

80 man-hr on timbering for advance of 58 ft. e, 20° to 30°. (*d*) 1 driller and 1 timberman. in ft for timbering. (*g*) Timbering, 20 man-hr ng. (*i*) Round required, 102 ft of hole, 14 to 20 2, 3, 6, 7, and 9, no data are available as to time to 8 (106), example 9 (112). *S* = stope-drill.

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man-hr is higher in raising than sinking; they also indicate a slightly duty in raising than in drifting and crosscutting (Table 15). Raise by machine drill, shows higher powder consumption per cu ft than is generally true of machine-driven as compared with hand-driven

Cost of raises and winzes. (See Examples of practice in raisi 23; also Sec 21.)

Park City, Utah (109). RAISE of 5.5 by 17.5 ft on a contact vein in quartz hanging wall; good drilling and breaking; dry; good air; mule tram, 1 soc record covers 20 days, in which 25 machine shifts were worked; total advance, 62 ft; one 1-man piston drill; for wages and supply costs see Costs of drifting, Art 21.

Raise at Park City	Amou
Machinemen	$81
Muckers	58.
Timbermen	68.
Pipe- and trackmen	3.
Labor cost	$211.
Cost of operating machines	50.
Explosives	33.
Lumber and timber	81.
Hoisting	127
Supplies	4
General expense	16
Total cost	$524

Cost of a WINZE for period of 7 days at same mine is given below. Cross-sec, 11 by 7.5 ft; sunk in contact vein in quartzite, limestone hanging wall; good drilling and breaking; dry; good air; drilling done with 1 large piston drill. Material hoisted 110 ft out of winze, mule-trammed 1 200 ft; hoisted up an incline 200 ft, and then up main shaft.

Utah Copper Co, Bingham, Utah, 1915. Raises 4 by 5 ft; stope drills; progress, 5 ft per round; breaking and mucking contracted at $1.20 to $ depending on length of raise; company furnished supplies and paid for timb Costs per ft as follows: drilling and tramming (see Table 18 for tramming data), $1.50; powder, caps and fuse, $0.85; compressed air, $0.15; miscellaneous, $0.90; total per ft, $3.40. (Compare with drifting costs at same mine, Art 21.)

Winze at Park City	Amoun
Machinemen	$28.0
Muckers	60.0
Timbermen	50.7
Miscellaneous labor	19.5
Labor cost	$158.2
Cost of operating machines	16.0
Explosives	8.8
Lumber and timber	9.7
Hoisting	20.6
Supplies	4.3
General expense	1.2
Total cost	$219.1

Giroux Cons Mines Co, Nev, 1909. Vertical raise; height, 948 ft; cross-sec, 13 by 21 ft; used later as a hoisting shaft. Rock is limestone; aver round, 36 7-ft holes; 3 shifts worked continuously; total time, 180 days; aver advance per day, 5.27 ft; max monthly advance, 190 ft; minimum, 133 ft; 2 stope drills (for a short time 3 drills operated simultaneously); raise timbered with square sets (Art 45), 14 and 16-in round posts, and 8 by 10-in girts; raise was 3 sets long and 2 sets wide. Costs per ft (r raising, $16.69; tramming, $5.09; timber framing, $1.76; blacksmithing, $1.1 shop, $0.11; teaming, $0.54; air compressor, $10; supplies, $17.24; hoisti proportion of general underground expense, $3.44; total per ft, $71.21.

Iron Silver mine, Leadville, Colo. G. O. Argall furnishes following cos raises, 4 by 9 ft in clear, 1915: labor, $3.243; explosives, $0.964; air, $0.507; st timber, $1.348; hoisting, $0.725; total per ft, $6.996.

Montgomery mine, Candor, N C, 1912. Record covers 6 months' work with 2-man piston drills; hard quartz veins, aver width 2 ft; country rock, diabase; largely negro labor, unskilled and irregular; 10-hr shifts; labor cost $1.75, for helpers and muckers, $1.50; dynamite cost, 15¢ per lb; fuse, $6 to $ ft, caps, 80¢ per 100; candles, $3.25 per case (125). Costs per ft were as foll

1	2	3	4	5	6
.979	$3.364	$2.237	$2.513	$2.00	$2.405
.347	0.897	0.705	1.110	0.27	0.650
.480	1.290	0.788	0.957	0.22	1.200
.204	0.913	0.520	0.698	0.48	0.630
.015	0.040	0.037	0.047	0.03	0.054
.038	0.117	0.092	0.117	0.07	0.095
.026	0.098	0.057	0.070	0.04	0.071
.240	0.240	0.240	.....	.....	0.240
.200	1.400	1.400	1.200	1.40	1.400
53	$8.36	$6.08	$6.71	$4.51	$6.75

Ann rep for 1914 gives cost per ft of 11 251 ft
(For wages, supply costs and other records, see
l crosscut at same mine cost $7.52 per ft.

es		Raises	Winzes
6	Other supplies (*b*).....	$0.442	$0.954
5	Total supplies.....	$1.607	$2.586
6	Engineering.....	0.208	0.386
5	Assaying.....	0.415	0.889
0	Mech and elec dept....	0.183	0.255
6	Sampling.....	0.011	0.072
2	Surface expense.....	0.063	0.024
0	Power.....	0.331	0.563
4	Total distrib charges	$1.211	$2.189
5	Total cost per ft.....	$6.616	$14.265
3			

ftmen, pumpmen, nippers, blacksmiths and
us. (*b*) Drill fittings, pipe and fittings, track,
, sundries.

ord shows cost of winzes and raises for 1903;
Wages: drillmen, $3.50; shovelers and tram-
smith, $4; 8-hr shifts. Cost per ft as follows:

	Winze	Raise
Timber.....	$0.33	$0.57
Hoisting, underground...	4.79	.....
Hoisting, main shaft.....	1.48	0.89
Compressed air.....	1.74	2.08
Fixed and gen'l charges (*a*)	10.14	7.94
Total cost per ft.....	$38.77	$29.97

repairs, and apportioned charges for electric
ramming, ventilation, pumping, assaying, sur-

of Development Openings

otal ft	Total cost	Cost per ft Max	Min	Aver
4 57	$1 412.74	$3.5217	$1 0671 (*a*)	$3 0409
3.35	822.02	4.5187 (*b*)	1.2793 (*a*)	2 2623
7.24	65.53	1 6272	1 0526	1 3873
5.16	$2 300.29	$4.5187	$1 0526	$2 6284

t timbered. (*b*) Through loose boulders.

; hoistmen, $1.24 to $1.36; blacksmith, $1.37·
19.9¢ per lb; timber, $19.88 to $24.85 per M;
970 (129).

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Drifts were mostly in footwall, a hard, dense rhyolite breccia, requiring no ti
Crosscuts were timbered where they passed through vein, but not in the foot or h
wall rock. Raises were in footwall or in firm vein matter, and required no tim
Segregation of costs, for 875.16 ft of openings, was as follows:

	Total cost	Cost per ft		Total cost	
Shift bosses (share)...	$57.57	$0.0658	Hoistmen.....	$10.17	$
Miners.....	367.53	0.4200	Topmen.....	7.83	
Contractors (labor and supplies).....	1 140.01	1.3026	Blacksmiths.....	40.63	
Timbermen.....	72.56	0.0829	Total cost of labor and contractor's supplies.	$1 885.74	$
Helpers.....	52.93	0.0605	Supplies, including lumber.....	414.55	
Laborers.....	35.87	0.0410	Total cost of development.....	$2 300.29	$
Powder boys.....	8.52	0.0097			
Trammers.....	49.74	0.0568			
Skip loaders.....	42.38	0.0484			

Standard Consol mine, Bodie, Cal, 1905. Hand drilling; miners, $4 per 8-h
Cost per ft: drifts, $4.38; crosscuts, $5.15; raises, $3.65; shallow winzes, $5.55 to
aver cost all development, $3.92 per ft, of which labor was $2.80 and supplies $1.1

Mining companies often publish footage costs of development work w
segregation into costs of drifts, raises, crosscuts, and winzes; these are use
preliminary estimates in same districts. Following costs per ft are mostly
from company reports.

Mary Murphy Gold Mining Co, Cripple Creek, Colo, 1915. 4 401 ft drifts;
ft crosscuts; 1 677 ft raises; aver cost, $7.22.

Elkton Consol, Cripple Creek, Colo, 1915. 2 939 ft drifts, $3.98; 175 ft raises,
207 ft winzes, $9.55. Costs cover labor and powder only. Contract prices for dri
$6, and $6.50; for winzes, $11.50.

Portland Gold Mining Co, Cripple Creek, Colo, 1914; drifts, $6.62; crosscuts,
raises and winzes, $7.77.

Liberty Bell mine, Telluride, Colo, 1911. Contract price for drifts 6 ft wide b
$8; for widths exceeding 9 ft, $11; 3.25-in piston drills; contractors buy powder, fuse
and candles from company, place stulls and lagging, and tram muck not over 500 ft

Nevada Hills, Fairview, Nev, 1914. Drifts and crosscuts, $8.15; raises,
winzes, $15.27; total footage of all kinds, 5 697 ft.

Nevada Wonder, Wonder, Nev, 1914. 11 276 ft drifts, crosscuts and raises
cost, $6.44.

Montana Tonopah Co, Nev, 1910. Drifts, $6.24; crosscuts, $5.84; raises, $5.1

West End Consol Mining Co, Tonopah, Nev, 1913. 1 546 ft drifts, $6.50;
ft crosscut, $6.44; 1 512 ft raises, $6.66; 383 ft winzes, $13.39.

Silver King Consol, Park City, Utah, 1915. Development and exploratory drifts,

Consol Mercur Co, Mercur, Utah, 1912. Aver cost 3 657 ft develop't openings.

Wolverine Copper Co, Mich, 1912. 4 293 ft drifts, $6.12.

Franklin Mining Co, Mich, 1912. 4 365 ft drifts, $5.79; 199 ft crosscuts, $5.57
ft raises, $9.30.

Mass Consol Copper Co, Mich, 1908. 6 410 ft drifts, $5.75; 466 ft crosscuts,

Alaska United, Douglas Island, Alaska, 1913. Ready Bullion mine, 3 997 ft dev
ment, $13.40; on 700-ft claim, 2 783 ft, $11.92.

Alaska Mexican, Douglas Island, Alaska, 1913. 2 464 ft development, $15.75.

Alaska-Treadwell, Douglas Island, Alaska, 1913. 5 980 ft development, $13.80

Le Roi No 2, Rossland, B C, 1913. 3 503 ft drifts, 628 ft crosscuts, 482 ft raise
ft winzes, total 4 633 ft; aver cost per ft, $15.87.

Hudson Bay mine, Porcupine, Ont, 1913. 298 ft drifts, $14.43; 446 ft crosscuts, $1

Kerr Lake Mining Co, Cobalt, Ont, 1913. 2 974 ft drifts, 1 663 ft crosscuts, 16
raises, 186 ft sinking; total 4 984 ft; aver cost per ft, $5.84.

ITATION

eneral

the means of supporting the COUNTRY ore itself, during the process of stoping. y or temporary support. In some cases, n or slabs of rock or ore; in others, large part of the method.

ing methods are: (*a*) Surface and over- PILLARS left in the orebody; chambers s, the removal of the ore being incom- the orebody and filled either contem- ste (as broken rock, sand or gravel). the pillars are mined; filling, if contem- and stope walls during mining. (*c*) A nd overlying material allowed or forced adjacent sections. Or, successive por- d then broken down by weight of the their own weight. CAVING METHODS above the deposit.

aried in many ways. In veins or masses, usually mined in descending order; to ated on as few levels as possible; num- ly depends on the size of deposit, distri- d. In large, uniform deposits, the ideal hat above is being mined, and at same w. Relative times required for these 3 orebody, determine sequence and extent ning. Development on a level is often rted on that level. In flat beds, mining simultaneously, the workings being ex- oint of entry; such methods are called TREATING SYSTEMS, development open- ndaries, where mining begins, the work- int of entry. These terms apply also,

f Mining Methods

Methods..... (Art 25 to 94)
..... (Art 95 to 101)
..... (Art 102 to 111)
..... (Art 116 to 127)

of metal-mining methods: shape, size, al and mineralogical character of ore; haracter of wall rocks and of overlying bodies on same property; kind of labor st of timber and material for filling.

varying importance. The method chosen t and extraction. Last 2 factors are closely of the orebody often yields maximum total so planned that low-grade ore and pillars, future if desired. Improvements in mining o lower the tenor of profitable ores.

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Classification of underground metal-mining methods. A logical c
tion, based on the factors outlined above, is impossible, because of their
relations. TYPE OF STOPE is used here as a basis of classification; the stop
selves are grouped, according to modes of supporting walls and men, as

I. Open Stopes..... (Art 29 to 44) IV. Shrinkage Stopes.. (Art 6
II. Timbered Stopes.. (Art 45 to 58) V. Caving Methods.. (Art 7
III. Filled Stopes..... (Art 59 to 67) VI. Combined Methods (Art 8

This arbitrary classification is adopted for presenting the details with a mir
duplication. For classification of metal-mining methods from standpoint of th
cability to different types of orebody, under different conditions as to charact
and wall rocks, see Art 94.

26. Breaking Ground in Stopes.

Breaking ground covers the work of blasting, including locating,
charging, tamping and firing drill holes. For charging and firing, and
explosive for different kinds of work, see Sec 4 and 5.

Fig 123 shows a drill hole AB in vertical section; top A and bottom B of a
called its COLLAR and TOE. Holes pointing downward are DO
or WATER HOLES, water being usually kept in them while drilli
holes, or those at a slight angle above or below, are FLAT HO
Fig 126); holes drilled steeply upward are UPPERS (UP, Fig
holes which will not hold water, and uppers, are also called DR
In Fig 123, the rock surfaces CD and DG are FREE FACES, a
the LINE OF LEAST RESISTANCE of hole AB. Length of lin
right angles to AB, is the BURDEN on the toe of the hole. (F
data on blasting, and remarks on Theory of blasting, see Sec
6, 7.) In practice, the arrangement of holes and weight of c
matters of judgment, based on experience under similar conditions and modified
ment with the rock or ore to be broken.

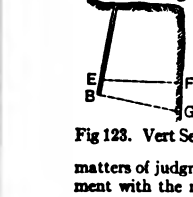

Fig 123. Vert Sec

Breast stoping. Ore is broken by flat or slightly inclined holes, dri
vertical face (or breast) of considerable lateral area, which is being adv
a horiz or nearly horiz direction; work resembles that of advancing th
a very wide drift. This method is used in flat, thin beds; also in parts o
or wide veins, to provide openings for other methods of attack (Art 30

Underhand stoping. Ore is broken in horiz slices, in descendin
miners stand on ore and drill holes downward; for different forms
which result, see Fig 147 to 153, Art 33.

Overhand stoping. Ore is broken in horiz or inclined slices, in a
order; miners work beneath the face of ore; holes may be flat, uppers,
holes. For different forms of overhand stope, see Art 36.

Underhand and overhand methods are employed under widely different con
to size and nature of orebody; in both, the stope faces are usually maintained i
benches (Fig 147, 159, 161). Ore is blasted in blocks from free ends of be
nearly all cases there are at least 2 free faces, to which a hole can break.

Caving (Art 71). A portion of the orebody is undercut and allowe
by its own weight.

Objects sought in breaking ground are: (*a*) to break ore from sto
(*b*) to break ore into pieces of a size suitable for handling.

Very large pieces of ore clog chutes and mill-holes and can not be handle
they must often be broken in stopes by sledging or BLOCKHOLING (Sec 5, Art 7).
require large amounts of explosive (see note m, Ex 8, Table 29). Shallow ho
spacing of holes, and high grades of dynamite, all produce fine breaking, but a
cost. Th[illegible] usually easily adjusted in narrow stopes; in wide dep

ease depth and spacing of holes and to block-
127). Economy of this practice depends on
ower; the manner only has greatly cheapened
n stopes to a smaller size than necessary for

Stopes by Hand Drilling

der following conditions (see also Choice
where ore is to be broken with minimum
an be carried by hand than by machine;
lo not equal hand drilling in this respect;
f high-grade ore, where the tonnage does
and machine drills; (*c*) where power is
; these conditions have caused extensive
l mines (Art 41); (*d*) in very soft ores,
in the early life of mines, where stoping
exploration has proceeded far enough to
) for various local reasons, for instance:

st be sorted in large stopes into smelting and
ork is favored because it allows careful break-
alition mine, the orebody is so flat and thin in
e used, and most drilling is done single-hand
a mine at Tintic, Utah, hand drilling was
stope-drills and consequent danger of "lead-

ium hard ore, a miner drilling 4 to 7 ft of
widely with character of rock; in many
rill a certain number of holes or footage
nd.

ertical water-holes, speed decreasing as incli-
riz holes or flat uppers. Speed increases in
is dry, so that cuttings run out freely; (*b*)
e is room for miner to get a full arm swing.
nd should be considered in planning stoping
th depth of hole. Economic limit for single-
pes, 2 to 3 ft; 3.5 to 4.5-lb hammers are used;

rd ground; increase in speed is not pro-
st applied to nearly vertical down holes,
r systematic stoping work in U S. Drill
-lb; ordinary depth of holes, 4 to 6 ft

or hand-augers are more efficient than single-
e best used in down holes, and under suitable
er hand drill; they have a very limited appli-
east and down holes; both auger and churn
s or nodules.

o those for machine drilling (Fig 124 to 128);
high; holes are roughly parallel to free faces,
d miner takes advantage of slips and irregu-
ground; hence, consumption of explosive per
nes.

an be carried by hand drilling is usually
e in thin, steeply dipping veins, with well
ork at a disadvantage in them; minimum

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on stoping by hand drilling (see also Art 41 for hand-drilling practice
STANDARD CONSOLIDATED MINE, Bodie, Cal. C. E. Grunsky gives fu
1912 (84). Narrow auriferous veins occur in firm, fairly hard andesi
steep, there is no handling expense in stopes except for shoveling to
ter is hard banded to soft, porous, granular quartz; sometimes a soft cl
quartz stringers. Drilling is single-hand. Stopes are back-filled with
aste being hoisted to surface. Stopes are kept as narrow as possible; m

.5 to 2 ft.
nd trammers,
8-hr; 2-shift
ower, $25 per
dynamite
), 16¢ per lb;
to 10-in diam,
lin ft; sawed
$35 per M.
shows effect
of vein on
per man stop-
labor cost of
and handling
stope chutes.
24 gives cost
ing supplies,
ag that aver
consumption
ut 2.4 lb per
ken, and that

Table 23. Overhand Stoping by Hand. Narrow

Width of vein, in			Vein filling	Dip of vein	Tons ore broken per man-shift
Aver	Variation From	Variation To			
3	....	....	Hard	Steep	0.38
5	4	6	Hard	Steep	0.42
5	4	6	Soft	Steep	1.21
8	3	17	Medium hard	Flat†	0.91
9	6	13	Medium hard	Flat†	0.99
12	8	18	Medium hard	Steep	1.46
18	14	20	Hard	Steep	1.44
28.5	9	42	Soft	Flat†	1.91

* Cost of miners only; aver stoping labor cost, i
distributed general labor, $3.986 per ton of ore. † Ha
was handled several times on account of flat dip.

with hard ore and walls it exceeded 3 lb per ton. Both tables are base
not on total tonnage of vein and wall-rock broken. A SMALL MEXICA
om R. H. Allen (129). Vein, 10 to 80 ft wide; footwall, strong rhyolite
wall, badly broken andesite; as ore is soft or cut by clay seams, it req
plosive. Overhand stopes are supported by square-sets (Art 45), filled w
n 1 floor of the back; vein is mined in 15 to 24-ft sections, along strike.

**Table 24. Breaking Ground by Hand Dr
Cost of Stoping Supplies per Ton of Ore**

Width of vein, in	Lumber	Explosive	Steel and misc	Distributed 40c†
Average*	$0.429	$0.388	$0.108	$0.544
7 (a)	0.608	0.500	0.207	....
12 (b)	0.337	0.539	0.135	....

* 1 year's figures for all veins. (a) Verti
medium hard quartz, firm andesite walls. (b) S
hard banded quartz, hard andesite walls.

ras; single-hand drill-
r shifts; miner's wages,
underground laborers,
o 62.1¢; rate of ex-
$0.497. Figures cover
s of 1911, during which
253 tons of ore were
Table 25 gives details.
, NEV. Very hard
vein, carrying gold and
ver width, 2-4 ft; dip,
33°; wall rocks, strong
; vein mined by over-
oping in open stopes;
lls wall rock is broken;

is single-hand, holes flat or pointing slightly down. A miner averag
er 8-hr shift; 331 pieces of drill steel drilled per miner-shift. In Apr.,
were broken in 861.25 miner-shifts = 1.16 tons per shift. DALY JUDGE M
Utah. In 1907, single-hand miners averaged 12 ft of hole per shift in
ft in quartzite. TINTIC, UTAH. L. A. Palmer gives data on work in 19
ex Consol mine, much ground is so soft that holes for blasting are m
steel ("ball-point"); 1 man breaks 25 tons per shift. At Iron Blosso
hand square-set stopes, medium hard ore, 1 single-hand miner breaks 7 t
ft. FRANKLIN FURNACE, N. J. Before machine drills were introduced
double-hand in ore like tough limestone, put in 3 holes totalling 8 ft
reaking about 8 tons, or 4 tons per miner-shift (58). NEGAUNEE, MICH.
drilling 3-ft water holes, double-hand, in soft hematite, was about 3 f
ed auger in the ore reduced speed greatly (158). CANOCE MINE, Virgin

hat harder than usual Mesabi ground. 2 miners about 2 hr. Air-driven augers averaged 1 ft per ole-hand drilling of 6-ft water holes in firm augite ishing bit, 1.25-in; ⅞-in steel. 2 men averaged

Overhand Stoping by Hand; Méxican Labor

	Total shifts	Shifts per ton	Tons per man-shift	Cost per ton broken
s (*a*)....	236.85	0.0138	72.80	$0.0213
.....	3 488.33	0.2021	4.94	0.1519
b).....	1 382.40	0.0802	12.48	0.0480
rs.....	311.25	0.0180	55.40	0.0089
s.....	163.50	0.0095	105.50	0.0109
.....	5 582.33	0.3236	3.09	$0.2410

	Quantity		Tons per unit	Cost per ton broken
	Total	Per ton		
40%...	1 194.3 lb	0.0692	14.49(*c*)	$0.0121
.....	7 826.0 ft	0.454	2.20	0.0015
.....	4 162.0	0.241	4.15	0.0009
o 15 L..	28 800.0	1.67	0.60	0.0139
.....	98.0 lb	0.0057	176.0	0.0004
.....	110.0 lb	0.0064	155.3	0.0012
.....	3.6 lb	0.0002	.....	0.0000
.....	4.0	.....	.....	0.0001
o 2 D...	34.0	0.002	507.0	0.0016
.....	6.0	0.0004	.....	0.0002
.....	.....	.....	.....	0.0007
al.....	4 090 lb	0.237	4.22	0.0008
.....	.....	.....	.....	$0.0334
ost, labor and supplies				$0.2744

tionate part of bosses' total time. (*b*) Shovelers (*c*) Equivalent to 0.069 lb dynamite per ton.

le per 8 hr. Minimum task per shift is 4.42 ft of ft of water hole in underhand stopes. ⅞-in steel 5 in (140). CHIKSAN MINES, Chosen, Korea (in to 8-ft quartz veins, dipping 65° or over; country ract miners, working overhand in shrinkage stopes, 5 to 0.57 ton per man-shift; consumption of 60%

in Stopes with Machine Drills

machine drills as used here, see Art 21)

t 33, 34). Fig 124 shows All holes are water holes, data on depth and spac- d Jackhamers or similar rk; former are mounted ar mounting in stopes up ches depends on width of rk economically and size

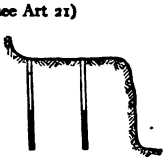

Fig 124. Vert Sec

ches can not be broken efficiently in narrow to 10 or 15 ft wide is 4 to 7 or 8 ft.

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Ordinary limit of depth of hole drilled by 3 to 3.25-in piston drills, 16 to 20 use of deep holes in a large stope, see Ex 15, Table 30. Deep holes make for breaking; objections to them are: (*a*) long steels are awkward to handle under (*b*) ore may break in large pieces difficult to handle along stope face. Objection not be serious; deep holes usually make steep faces, where large pieces come to rest at foot of conveniently for blockholing. Usual depth of drill holes in wide underhand stopes is less than economic limit of depth of hammer-drill holes, to 8 ft. Burden broken with holes placed as in depends largely on character of ore and must be mined by experiment. In easy ground, max d between holes and between rows of holes is 0.75 depth of hole (Art 26). In tight ground, at Alaska-Treadwell, 12-ft holes were dril apart in rows 2.5 ft apart, and staggered as in Fig 125 (133). For further de spacing and depth of holes, see Breast stoping, Art 30.

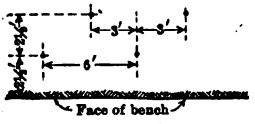

Fig 125. Plan

Overhand stoping comprises STEPPED-FACE and FLAT-BACK STOPE also Art 36). Fig 126 shows part of a stope face. Benches may be brol flat (breast) holes, running across stope, as at *FT*, or by uppers, *UP*. Uppers are often pointed as at *IC*, so that cuttings will fall clear of drill. In narrow veins, uppers are drilled parallel to dip to avoid breaking into walls.

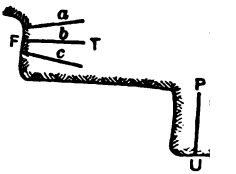

Fig 126. (In wide stopes, a vert, longit sec; in veins, a sec in plane of

Uppers *vs* breast holes. Uppers are well adapted to efficient use of stope-drills, but they shatter, more than breast holes, the back under which men must work in drilling the next slice. This is more important in wide than in narrow stopes, especially in presence of horiz slips. Breast holes can often be drilled as water holes, thus avoiding nuisance and danger of dust.

At Baltic mine, Mich, and in parts of Portland mine, Cripple Creek, Colo., holes are said to break ground more efficiently than uppers, besides being saf Broken Hill, N S W, in mining wide lodes in flat-back stopes, uppers are prohib a safety precaution (141, 142). Breast holes may or may not be harder to dri uppers, depending on character of ore.

J. A. Fulton, Sup't Melones mine, Cal, furnishes following data showing that ch of ore may influence in other ways the choice between breast and upper holes, filling at Melones is amphibolite schist, with irregular quartz stringers. Vertic are parallel to schistosity and will not break ground in wide stopes; when blaste leave long jagged pieces projecting from the back, which are dangerous and i impossible for following shift to place and drill their holes advantageously. breast holes across the schistosity is easier, and explosive acts much more effectiv

Uppers are common for overhand work in narrow stopes on steep dips; breas are used in such stopes on low dips. This is true of both hand and machine type of drill used also affects choice (see below under Choice of drills for stoping). may be drilled with either piston or stope-drills, usually the latter; breast hole piston-drills, which, in stopes up to 8 or 10 ft wide, are mounted on bars; in wider usual mounting is a column braced between back of stope and filling or timber.

Height of bench in machine-driven, overhand stopes depends on same as for underhand work. In narrow stopes, benches are 3 to 6 ft high; i stopes, breast-hole benches are 6 to 12 ft high, broken by 2 to 3 rows of 6 or 14-ft holes (*a*, *b*, *c*, Fig 126). Benches broken by uppers are usually 5 high and holes have corresponding depths. Tables 29 to 31 show prac to depth of hole. In square-set stopes (Art 46), depth of either breast or holes is determined by dimensions of sets (note *k*, Ex 16, Table 30).

Character of ore may influence depth of hole, and therefore height of bench; at Consol mine, Nev, 5 to 7-ft holes are drilled in very soft ore, because blasting chambers the bottoms of deeper holes, failing to break (143). Usual range of

For examples of drilling speeds, duty of labor and bles 29, 30, 31, 36.

reaking ground, combining caving with ordinary ge flat-back shrinkage stopes (Art 68) in several lifications at Homestake nsol mine, Ariz (note *D*, 10 to 15 ft wide in soft und. Pairs of 6-ft holes, 27), break shaded areas; rt and causes it to fall by Ex 5, Table 31. H. R. Braden mine, Rancagua, le copper-bearing zone of a volcanic plug. Part of topes, the aver width of of deposit. Stope-drills face), holes being placed as in Fig 127; distance are drilled per aver shift, which when loaded with 60 tons. (See also Table 26.) Powder consump- and 30). Large blocks of ore falling from back do p on ore filling and may be blockholed if necessary.

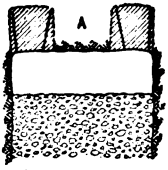

Fig 127. Vert Sec through Top of Stope

round, Braden Copper Co, 1914

Lb 34% gelignite		Aver depth of hole, ft	Ft per drill-shift
Per ton	Per hole		
0.26	0.945	4.4	67.2
0.33	0.950	4.4	69.00
0.29	0.902	4.4	72.0
0.23	0.988	4.4	58.0
0.39	0.857	4.5	50.0
0.31	0.881	4.5	74.0
0.31	0.925	4.4	71.2

m hard breccia, 9% soft fractured andesite, all easy asy breaking; 35% medium; 21% hard. (*c*) 50% t breccia; 25% medium blocky andesite; 18% soft oft fractured andesite, easy breaking. (*e*) 37% soft 32% soft breccia. (*f*) 63% medium breccia; 37% reccia; 26% hard andesite; 18% fractured andesite.

topes (see Art 36). Fig 128 shows sections stope faces (133). Holes are drilled in rows tope. All holes are wet, down holes, which ust and is favorable for hand drilling (Art 44).

stopes on the Rand, piston-drills on columns make holes. In a stope 6 ft wide, a 6-ft hole may carry rden; in tight ground, burden may be only 2 ft; ary stoping widths (5 ft), there are 2 to 3 holes in a ep holes in narrow stopes will "bull-ring" (ground hole is broken, but that on collar is not); prevented g shallower holes and placing a heavier burden on 36 give details of Rand work. No data are avail- nes where rill stopes are used in U S.

can not be used for breast stoping, as they this work, the choice depends on hardness of on-drills are required; under aver conditions r Leyner drills suffice; in very soft ores, air-

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operated augers, or plugger-drills with an automatic rotating device so tha can be used as either augers or pluggers, are the logical choice.

Many of the remarks on choice of drills for drifting (Art 21) apply here. Fo holes in underhand stoping, hammer drills are efficient. In steep-dipping overhand where the holes are uppers, stope-drills are best under aver conditions. They are li allow holes to be pointed as desired; delay in setting up, incident to piston-drill avoided and that due to changing steel is reduced. But, hammer drills produce mu unless used with a water jet, and are not suited to very hard ore; in the latter, pist are used for uppers, choice between 1 or 2-man sizes depending on difficulty of dri

Breaking ground with uppers in overhand stopes on flat dips (20° to 30°) re breast stoping; character of ore then determines whether stope or piston-drills used; in some dry ores, cuttings will run out of holes drilled on angles as low a 20° above horizontal; in moist or clayey ores, cuttings may pack around stope-d even in vertical up holes; all conditions are found between these extremes. E differ as to choice of proper drill for narrow, flat-dipping deposits. Modern shows an increasing use of stope and plugger-drills. In general, piston-drills a (*a*) in ground that is very hard or that fitchers badly, even if soft (Leyner dri times overcomes the latter condition); (*b*) in large-scale blasting with deep large stopes. MINIMUM WIDTH OF STOPE in which piston-drills operate effici about 4.5 ft; this width is apt to be exceeded, especially for large drills. St plugger-drills can be used in stopes about 3 ft wide on steep dips, but require c supervision to maintain this minimum. Minimum possible width of stope is increased by flat dips. In narrow orebodies, where clean mining is desired, these influence choice of drill and also choice between hand and machine drilling (Art

Examples illustrating foregoing discussion. LIBERTY BELL MINE, Tellurid (131). Quartz vein, aver stoping width 4.3 ft and aver dip 57°, was formerly m hand drilling, as vein matter was crushed and loose, and economy of air drills narrow veins was doubted. Table 27 shows effect on breaking costs produced by

Table 27. Cost of Breaking Ore, Liberty Bell Mine

	Year	1906	1908	1910	%Char
	Production, tons	92 221	116 133	134 321	+ 4
Cost in cents per ton	Dynamite	8.76	10.68	9.09	+
	Fuse and caps	2.20	2.53	2.48	+ 1
	Stoping labor	70.63	51.11	32.14	− 5
	Air pipe and compressor: labor	1.66	3.44	3.81	+12
	Air pipe and compressor: supplies	0.57	2.45	0.74	+ 2

from dominant hand mining in 1906 to machine mining in 1910. Stope-drills a for mining overhand by uppers; a saving of 54.5% in stoping labor result change, notwithstanding that drillers were paid more than hand miners. BERLI Nev. Hard quartz vein, aver width 2.6 ft, aver dip 40°, was mined clean and at by single-hand drilling in overhand stopes (Berlin mine, Art 27). An attempt to hand-drilling with 2.75-in piston-drills failed; stoping width with machines i to about 5 ft; mill had a fixed capacity and no sorting was done ahead of it; material sent to mill was lowered about 50% and tonnage increased but little; o resulted in a loss. FRANKLIN FURNACE, N J. Overhand stoping; ground h sistency of hard, tough limestone and is now broken by uppers drilled with sto in large stopes. B. F. Tillson gives data in Table 28, showing effect of changi all 3-in piston-drills in 1908 to all stope-drills in 1913; extensive tests on numero were made (98). MELONES MINE, Cal. J. A. Fulton, Sup't, furnishes followi (see also Ex 4 to 7, Table 30). Ore is schistose, containing quartz stringers. A 3-in piston-drill put in 8 6-ft holes per 8-hr; 1 man, with a Sullivan Liteweight piston machine, drills 7 6-ft holes in 8 hr. The 1-man drill is therefore chea 3-in drills are used in one orebody, where ore is mostly quartz, and for holes in quartz vein on the footwall of some stopes. (See Overhand stoping, Art 28, fo tions of stope-drill for uppers in this ground.) ALASKA-TREADWELL MINES Kennedy gives following data in 1916 (97); for method of mining, see Art 69. mineralized diorite (Ex 8, Table 29). For years, 3 to 3.25-in piston drills were

re at Franklin Furnace, N J

Per cent size of powder, 50% strength		No caps per ft	Powder, sticks per		Cost of explo per net ton	Net tons broken per man in stope
1⅛ in	1 in		Net ton	Drill excl B H shift		
...	...	....	0.76	17.7 19.3	....	....
...	...	....	1.01	20.7	....	....
16	84	0.310	1.025	39.0	$0.055	13.2
11	89	0.356	0.77	95.8	0.041	20.3
6	94	0.415	0.86	108.3	0.045	25.6
2	98	0.420	0.89	233.0	0 049	26.7

ills. P = Piston-drills. (*a*) Hammer drills are puts in about 75 holes in 5 hr. (*b*) 2 months' ft reduced from 10 to 8 hr in middle of 1913.

market were tried and found unsuited because r Leyner drills were then tried and averaged as on drills with about same air consumption. At rilled per machine shift by 3.25-in piston-drills , 31.7, 33.1 and 29.4 ft; in 1913, when all drills 4 ft was drilled per machine shift. One advan- f the water jet, it puts in flat holes difficult for p, Nev. In flat overhand stopes, both piston mined partly by dip, partly by dampness of ore; ed at fair speed by stopers, if cuttings are cleaned

ng. (See Art 44).

machine drilling. Following tables cover Breast stoping, Art 30 to 32; Michigan cop- , Art 41; and individual mining methods.

king Ground in Stopes

	Kind of drill	Ft of hole per shift	Length of shift, hr	Tons broken per shift (*c*)			
				In stopes			
				Per man breaking ground	Per man (*a*)	Per drill	Per man underground (*b*)
	S	45	8	42.8	13.5	45 5	9 35
	S	45†	8	13.1	6.18	35 0†	5 58
	e	*e*	8	41.3	34.6	57.0†	21.7
	S	90	9	9.1	5.4	10.2*g*	2.9
	3 in *P*	45	8	14.4*i*	5.11	38.0	3.9*i*
	3.25 in *P*	31	8	25.3	23.7	118.0	15.0
	S	25*k*	8	5.4	2.43	5.64	1.92
	3.25 in *P*	28	10	10.2	8.96	34.7	6.04
	S	77†	8	26.9	6.33	37.0	5.05
	S	45*m*	8	11.1	3 73	20 34	2.26
	S	102†	8	45.9	36.2	150 0	12.7

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Table 29 (*Continued*)

Example No	Dynamite, lb per ton	Labor							Date
		% breaking in stopes	% shoveling in stopes	% timbering in stopes	% in development work	% miscel underground	% on surface	Aver daily force (total)	
1	0.676	13.8	20.7	2.3	28.7	15.0	19.5	87	1912
2	0.66 to 0.9	30.7	....	7.7	26.9	34.6	....	26*	1912
3	0.227	....	....	....	19.1	17.3	12.5	104	1912
4	0.814	21.6	10.3	4.1	64.0	....	....	1 480*h	1912
5	0.454	17.5	18.9	13.2	15.6	13.3	21.5	302	1911
6	0.670	40.4	....	2.7	....	24.8	32.1	109	1912
7	1.330	29.2	21.4	5.2	11.0	33.2	....	195*	1912
8	1.470*m*	50.1	6.7	....	13.0	19.0	11.2	442	1911
9	1.980	9.9	3.3	28.9	24.8	14.9	18.2	121	1913
10	0.544*q*	18.0	18.9	12.6	20.0	17.9	12.6	470*p*	1915
11	*b*	22.9	....	6.1	17.9	40.9	11.2	358	1912

Output (Tons broken per day)

Example No	Ore		Waste		Total broken in stopes	Total broken	T was tor
	From stopes	From development	From stopes	From development			
1	500	40	13	100	513	653	o
2	80	20	25	20	105	145	o
3	1 833	133	.....	.....	1 833	1 966	....
4	2 578	458	322	1 013	2 900	4 371	o
5	765	66	.....	96	765	927	o
6	1 112	.....	.....	.....	1 112	1 112	....
7	306	35	.....	33	306	374	o
8	2 240	77	.....	47	2 240	2 364	o
9	323	177	.....	.....	323	500	....
10	874	56	.....	.....	874	930	....
11	3 766	268	.....	.....	3 766	4 034	....

† = Average. * = Underground only. *Oh* = Ore broken by overhand s
Op = Open stopes. *Sh* = Shrinkage stopes. *S* = Stope-drill. *P* = Piston-dri
= Caving method. *A*. Overlapping lenses of hard crystalline auriferous quartz
eral dips, 30° to 40°; usual widths, 8 ft or less, but reach 25 ft in some stopes; c
rocks are schist, alaskite, limestone, and diorite; ore carries $4 to $9 per ton i
hanging wall firm and supported by pillars of ore; stopes are large. *B*. Steep-d
quartz veins in slate; 25% of ore mined comes from overhand stulled stopes; rem
from shrinkage stopes. *C*. Massive deposit of medium hard shattered quartzi
pregnated with chalcopyrite; orebody, 400 ft long, dips 50° and has been opene
length of 450 ft and to a depth of 1 300 ft. A caving method of mining is used, fo
see Art 81. *D*. Large massive deposits of low-grade copper ores in diorite and
stone; ore is easy drilling, stopes are practically vertical. *E*. A series of fla
tabular masses of pyrite and chalcopyrite in alaskite porphyry; largest depo
max width of 228 ft and is about 500 ft long; ore carries 3 to 12% copper, 2 to 3 o
and $0.60 to $1 gold; ore breaks easily and is mined by top slicing. *F*. Quart
4.5 ft wide; dip 23°; vein is mined in overhand stulled stopes. *G*. Quartz vein;
wide; aver dip 57°; easy breaking; overhand stulled and shrinkage stopes (see
and Sec 21, Ex 6). *H*. Irregular ore shoots occurring in silicified zones in daci
latite; common dips, 40°; widths, from pinch to 50 ft; data compiled from Ann
1915. (For other details, see Sec 21, Ex 2; for square-set practice, see Art 4
Large intrusive mass of mineralized monzonite porphyry, aver 1.49% copper.
ground work was done only to keep mill running while capping was being strip
steam shovels. Stopes were vertical shrinkage stopes, 16 ft wide and 450 ft long.

(*a*) Includes men breaking ground, timbering and shoveling in stopes. (*b*) Exp
cost 9.1¢ per ton, which, at prevailing cost of dynamite, indicates powder consu

waste. (*d*) Total ore and waste broken in both es all underground labor. (*e*) 3 3.5-in piston-e-drills are in use on an aver shift. In stopes, deep or 50 ft of hole per shift; stope-drill puts July, 1912, 5% of ore was mined in square-set , 8% in shrinkage stopes, 68% by caving (top d 88). (*g*) Includes some ore broken by hand n, remainder mostly American. (*i*) On basis occurrence and method of mining see Mother time, 5.67 hr. (*l*) For detail of ore occurrence) Consumption in stopes. Including develop-n of rock broken. R. A. Kinzie in 1904 (105) r "bull-dozing" per ton rock broken. (*n*) Bib-ed comes from filled square-set stopes, 10.8% stopes, 6% from development. (*p*) Does not electrical, mechanical, milling and power plant Sec 21.

und in Stopes, Machine Drilling

	Meth-od of break-ing gr'nd	Kind of stope	Width of stope, ft	Dip	Type of drill
	Oh	*Sh*	30	*A*	*S*
	Oh	*Sh & Co*	30	*B*	*Ly*
	Oh	*Sh & Fi*	12	90°	3¾-in *P*
	Oh	*St*	7	25°	3-in *P*
	Oh	*Sh*	10	70°	*S*
	Oh	*Sh*	25	50°	*S & P(f)*
	Oh	*Sh*	40	70°	3-in *P*
	Oh	*St*	10	26°	*S*
	Oh	*Sh & Sq*	3 to 50	20° to 70°	3¾-in *P*
	Oh	*Sh*	16(*i*)	74°	3¾-in *P*
	Oh	*Sh*	6(*j*)	*G*	*S*
	Uh	*Op*	15 to 40	55°	3¼-in *P*
	Oh	*Sh*	6 to 20	70°	*S*
	Oh	*Sh*	15 to 50	*I*	*S*
	Uh	*Gh*	.....	*J*	3¼-in *P*
	Oh	*Sq*	*K*	*K*	*S*
	Oh	*Sh*	10 to 60	*L*	*S*

Tons broken per drill per shift	Dynamite		Date	Bibli-ography
	% Ngl	Lb per ton		
167	30	(*a*)	1913	(143)
(*c*)	30	0.45(*b*)	1909	(114)
14.4	60	0.61	1906	(146)
16.8(*e*)	.....	1.41	1916	(*d*)
24.7(*e*)	.....	1.47	1916	(*d*)
26.6(*e*)	.....	1.12	1916	(*d*)
34.2(*e*)	.....	0.73	1916	(*d*)
17.5	.....	1.0 *h*)	1914	(116)
10 to 15	.....	.....	1909	(147)
11.8	.....	.....	1909	(148)
8.01	.....	1.866	1915	.....
25.8	.....	.....	1913	(149)
28	.....	.....	1913	(149)
40	40	.....	1910	(150)
60	40 & 50	.....	1909	(121)
(*k*)	30	(*k*)	1913	(80)
(*l*)	35	(*l*)	1911	(151)

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Oh = overhand stoping. *Sh* = shrinkage stopes. *Ca* = caving. *Fi* = filled *St* = stulled stope. *Sq* = square-set stope. *Uh* = underhand stoping. *Op* = stope. *Gh* = glory-hole work. *S* = stope-drill. *Ly* = Leyner drill. *P* = pisto

A. Lead-silver ores, with much iron and manganese, occurring as a bedded repla deposit 180 ft thick; dip 17°; stopes are vertical; ore carries 4% lead, 3 oz silv $0.20 gold. *B*. Large massive deposit of disseminated copper ore in monzonit phyry; vertical stopes. *C*. Nearly vertical lodes of quartz stringers in amphibolite s ore occurs in shoots; mined by shrinkage stopes and rill and flat-back filled *D*. Lodes of quartz stringers in amphibolite schist. In Ex 4, vein filling is quartz; in Ex 5, ground is fitchery, but breaks well, and there is a very hard seam on footwall; in Ex 6 and 7, ground is fitchery but breaks well. *E*. Mine zones in schist. *F*. Replacement fissure veins; wall rocks, chiefly monzonite por ore minerals, pyrrhotite and chalcopyrite; principal gangue mineral is biotite, witl quartz and calcite. Ore occurs in shoots 50 to 500 ft long. Ores of district ca to 2% copper and $10 to $20 gold per ton. *G*. Steep-dipping, narrow veins i glomerate, quartzite and schists; ore, hard to medium hard. Data from annual for 1915. *H*. Large lenses of magnetite in gneiss; ore breaks badly at both Washington ore is harder than that at Mt Hope, but holes in the latter tend to f *I*. Vertical mineralized zones of rhyolite and andesite breccia surrounding a large c core of same material; ore over 200 ft wide in places. Stopes are vertical. description of orebody and present method of mining see Art 42. *K*. Mineralized zone in granite; easy drilling and breaking; individual stopes are vertical and 50 ft in horiz section (see Art 46). *L*. Steep-dipping, irregular shoots of gold ore, pying fractured zones in granite near dikes; ore occurs in sheeted zones 6 to 10 ft also in chimneys where horiz dimensions have reached 45 by 60 ft. Oxidized soft and easy drilling; unoxidized ore is hard breccia of granite and basalt.

(*a*) 2.5 to 3 sticks of dynamite per hole. (*b*) 4 sticks dynamite per hole. (*c*) 8. of ore broken per man underground per shift. (*d*) Data furnished by J. A. F Sup't. (*e*) Includes Jackhamer drills used for blockholing and drills in manwa ends of stope. (*f*) 50% stope-drills, 50% 2.75-in piston-drills. (*g*) For piston only. (*h*) Approx figure, 3.3 sticks per hole. (*i*) Or less. (*j*) Usual widths; stopes in low-grade ore much wider. (*k*) 4 to 5 holes break room for 1 square-set, ft capwise and girtwise and 8 ft high (see Art 47, 49). Burden on toe of holes, abou about 4 sticks of dynamite used in an 8-ft vertical upper. (*l*) In soft ore, a stop can make 40 ft of hole per hr and breaks up to 100 tons per shift; common cha sticks dynamite per hole.

Table 31. Breaking Ground in Stopes. Drilling Speeds (see also Tables 29, 3

Example No	Location	Character of ore	Kind of stope	Type of drill	Depth of holes, ft	Ft hole drilled per shift	Length of shift, hr	Date
1	Findley Consol mine,	*A*	*Oh*	*S*	5 to 8*a*	40	8	1907
2	Cripple Creek, Colo ...	*A*	*Oh*	2¼-in *P*	5 to 8*b*	25	8	1907
3	Tonopah Mining Co, Nev..	*B*	*Oh*	*S*	5(*a*)	60 to 90	8	1911
4	Coronado mine, Ariz.....	*C*	*Oh*	*S*	..(*a*)	90 to 110*a*	8	1910
5	Ray Consol, Ariz.....	*D*	*Oh*	*S*	6(*a*)	72 to 108	8	1915
6	Baltic mine, Mich.....	*E*	*Oh*	3½-in *P*	7(*b*)	35 to 42	9	1911
7	Goldfield Consol, Nev ..	*F*	*Oh*	2¼-in *P*	4.5*F*	36	8	1911
8	Minn mine, Soudan, Minn.	*G*	*Oh*	3½-in *P*	6 to 10*b*	8	10	1906
9	Miami Cop Co, Ariz	*D*	*Oh*	*Ly*	10 to 15*c*	60 to 80	8	1910
10	Creighton mine, Sudbury, Ont	*H*	*Oh*	*S*	6(*a*)	30 to 40	..	1913
11	Burra Burra mine, Tenn..	*I*	*Oh*	*S*	4.4 to 6*a*	40 to 50	9	..
12	Ariz Cop Co, Morenci Ariz	*J*	*Oh*	*S*	5.5(*a*)	68	8	1910
13	Mich Cop mine, Rockland, Mich	*K*	*Oh*	3½-in *P*	6 to 8.5*b*	37	8	

Oh = Overhand stope. *S* = Stope-drill. *P* = Piston-drill. *Ly* = Leyner-drill. Silicified zones in porphyry, tuff, and breccia, varying greatly in hardness. *B* Qu

Copper ores occurring in a vertical fault zone; d diabase; greater part of ore of medium hard- te deposit in schist; easy drilling; (see text in at Ray). *E.* Amygdaloid copper lode; fairly ing ground see Art 64. *F.* See Ex 10, Table 29, with stope-drill; ground generally easy drilling; o avoid difficulty of drilling flat holes (parallel hematite; difficult drilling; in some holes a half few inches, to prevent bit from sticking; each Hard nickeliferous pyrrhotite. *I.* See Art 34. d amygdaloid. (*a*) Drilled in 6 hr. (*b*) Breast Uppers.

STOPES

is a stope in which no timber or filling is d stope is an open hole. Various authorities , shrinkage stopes (Art 68) and some other the term comprises stopes in which walls stulls and other simple forms of timbering.

Gophering

to mining in irregular drifts or other open- thout regard to maintenance of a regular also called "coyoting" in western U S; in all-size, irregular, unsystematic workings.

used in small excavations where walls require e employed in portions of veins, beds or masses, If used to mine rich seams in a large orebody, ofit but eventual loss, as the irregular openings e remaining low-grade portions. On the other small irregular portions or isolated parts of an opment is not justified by probable tonnage to , high-grade, spotted deposits, and where Mexi- work, is available.

ica, India, and elsewhere, are adepts at gopher- ore is followed up, down, or laterally, as values te. Miners carry ore to surface in baskets or inzes or ladders connect workings at different ers," logs 10 to 15 ft long, 8 to 10 in diam, set de. Breaking ground in gophering is generally nbered or filled. Gophering sometimes follows ive work in mining soft, gold-quartz veins out- of drift-tunnels, 10 to 20 ft above one another; backs between tunnels, pillars being left to sup- ater level or hard ground is reached, workings

east Stoping

nethod of mining horizontal or flat-dipping rk in deposits less than 10 to 12 ft thick, or breast, the height of which is the thick- eneral method of working thicker deposits, by true breast stoping and lower portions better term for latter method would be ither case, roof is supported by permanent nates a method of breaking ground by ad-

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Type of orebody to which breast stoping is applicable is indica
the following description of Missouri lead and zinc deposits, in which the
has been extensively developed.

S E Missouri. Disseminated galena ores occur as flat or slightly incline
dips 3° to 10°) tabular masses in dolomitic limestone; thickness of deposits, o t
depth, 200 to 600 ft; aver tenor of ore for district, about 5% lead. These depo
irregular in horiz outline and cover considerable areas (Fig 130). Limestone
generally good, with local shaly and fissured areas. JOPLIN, S W MISSOURI.
stoping is used exclusively for mining flat or nearly flat bedded deposits of spha
chert, locally called "sheet ground"; depth of deposits, 75 to 250 ft, aver 175 ft
ness of beds varies from a pinch to over 20 ft; aver thickness of workable depos
10 ft. Overlying rock is strong limestone; in some cases, ore is overlain by 2 t
hard flint, which stands well and makes a clean roof to break to. Ore is lo
averaging a little less than 2% zinc. Methods of exploring these deposits are
Art 11. See also Wisconsin zinc deposits, below, and Mineville, N Y, Art 31.

Development. A vertical shaft is sunk to bottom of orebody; pra
no lateral development precedes mining; preliminary boring outlines the o
in a general way; stopes follow the ore outwar
shaft (Fig 130).

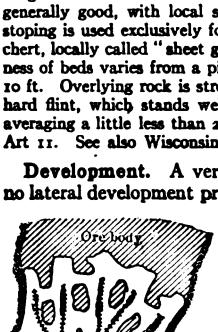

Fig 129. Plan of Breast Stope

General plan of work. Fig 129 shows
work; in deposits less than 10 ft thick, it con
driving wide drifts, the intervening ore bei
through at intervals to form pillars. Fig 129
different stages of work; obviously, the face
advanced in any direction. Fig 130 is a plan
of a mine in S E Mo, showing application of
over a large area (55).

Breaking ground. Methods vary locally and with thickness of or
as illustrated below (see also Ex 16 and 17, Sec 21).

S E Missouri. H. A. Guess gives following data (55). Where ore is less than 8 ft thick, face is advanced by breast holes; drills are mounted on columns; where there are well-marked bedding planes on top and bottom of ore, 2 holes in a vertical plane suffice, otherwise 3 holes are required; top and bottom holes are in same vertical plane, center hole is set 6 in nearer the face to be broken. All holes are approx parallel to face to be broken (compare with Fig 132). The 2 lower holes slope enough to carry water, top hole points a little upward. Aver depth of holes, 7 to 7.5 ft; in starting a stope or in turning around a pillar, face is broken into by 3 cut holes 4 to 6 ft deep. Where ground is carried with 2 holes, burden is usually 3 ft; with 3 holes, burden on middle hole is 3.5 ft, on others, 4 ft. In thicker orebod upper part is mined by breast holes as above; lower portion, called the stope, is

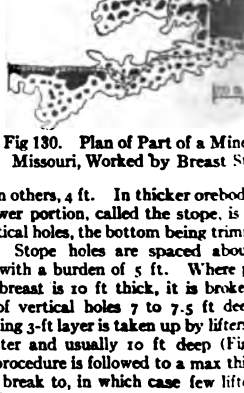
Fig 130. Plan of Part of a Min
Missouri, Worked by Breast S

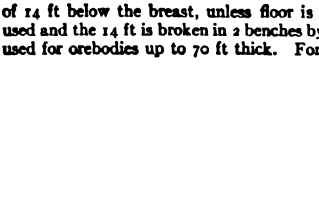

Fig 131. Longit, Vert Sec

by vertical holes, the bottom being trim
lifters. Stope holes are spaced abou
apart, with a burden of 5 ft. Where
below breast is 10 ft thick, it is broke
series of vertical holes 7 to 7.5 ft dee
remaining 3-ft layer is taken up by lifter
ing water and usually 10 ft deep (Fi
Same procedure is followed to a max thi
of 14 ft below the breast, unless floor is good to break to, in which case few lift
used and the 14 ft is broken in 2 benches by vertical holes. Same general method h
used for orebodies up to 70 ft thick. For breast work, the 2 rear holes are each

; middle hole with about 7 lb, and is fired first. In drilled in a row between pillars, the end holes are e, the middle one with 9 lb; middle hole is fired eneral is much lighter per ft, as the burden is less

d in pointing upper holes to avoid breaking into ortions of the back are removed by moils, using ning pillars, holes are placed to shoot away from g it. 95% of drilling is done with 1-man piston- to use self-rotating plugger-drills for down holes p on tripods. About 29 ft of hole are drilled per per ft of hole. Doe Run Lead Co, Flat River, o 40 ft thick; headings or breasts are about 30 vance of bench; benches are 8 to 10 ft high; drill

ast stopes are carried with a single vertical face sometimes to heights of 18 ft. Fig 132 shows f holes, which ; number and hape of block ound; typical pth of heading ace. Heading ls on columns, production per g on character of ground. Owing to toughness of (locally termed "squibbing") to make room for holes are squibbed at end of one shift and blasted from dust and powder smoke. In thicker ore- 15 to 20 ft in advance of bench or stope; bench illed from tripods. Sometimes 2 or more benches, ach about 10 ft high, are carried; in other cases, ingle 16-ft benches are broken by a row of stope oles at bottom and a row of "splitters" above hem; stope holes look down about 10°; splitters ook up same amount; stope holes are usually 12 to 4 ft deep, sometimes 17 or 18 ft; one machine drill- ng stope holes will keep up with from 6 to 10 drills n headings. 33% and 40% ammonia and gelatine latter for wet holes; in some mines, 80% gelatine of explosives is 20% to 30% of total breaking cost. s, first with 4 to 5 sticks of powder, increasing to uch a chambered hole will hold 100 to 200 lb dyna- d 2 splitters, all 18 ft deep, loaded with 3 to 4 boxes igh across a width of 10 to 12 ft, producing 35 to 45 ombined heading and bench work ranges at various

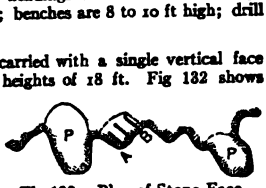

Fig 132. Plan of Stope Face, Joplin, Mo (P = pillar)

f breaking used at one mine (158); upper 12 ft of broke easily. Heading was driven in softer ground described above; tum was broken wn; splitter holes s drilled, as squib- nake it dangerous; ad of stope. This an the usual plan.

Fig 134

per 8-hr. (This wide veins; it is not applicable to thick beds, be- men and drills; nor does it allow careful scaling of, Davey No 3 mine, Joplin, Mo. Sheet ground is 16 ks well; heading, 7 to 8 ft high, is kept about 18 ft

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in advance of bench. Five 8-ft holes in heading break a width of 12 ft. Bench is with 16 to 18-ft lifters, 3 holes usually making a width of 40 ft. These holes a chambered and then loaded with 50 to 150 lb 40% dynamite; 3.25-in piston-dr used; aver powder consumption for both heading and bench, 1.43 lb per ton Otto Ruhl (161) gives data in Table 32, the record covering 4 weeks.

Table 32. Breaking Ore in a Sheet-ground Mine, Joplin, Mo (191

Week ending.....	Supplies				Tot. ave
	Nov 11	Nov 18	Nov 25	Dec 2	
Total sticks powder.....	.....	.....	6 808	7 452	...
Sticks used in shooting.....	5 647	5 935	5 296	5 650	22 5
Sticks used in squibbing...	.....	.....	1 512	1 802	...
Number of holes drilled....	186	192	165	195	7
Ft of hole drilled.....	1 471	1 544	1 328	1 522	5 8
Aver depth of holes.....	7.91	8.04	8.05	7.81	
Sticks per hole, squibbing .	.....	.....	9.16	9.24	...
Sticks per hole, shooting....	30.38	30.90	32.09	28.98	
Total sticks per hole.....	30.38	30.90	41.25	38.22	
Sticks per ton, shooting.....	3.160	3.052	2.61	3.03	
Sticks per ton, squibbing...	.....	.....	0.744	0.966	
Total sticks per ton.....	.....	.....	3.354	3.986	...
Tons broken per hole.....	9.61	10.12	12.31	9.56	
Labor					
Hours mill run.....	58	56	61	58	2
Tons per hr.....	30.80	34.70	33.29	32.18	
Machine men, shifts.....	52	54	53	54	21
Helper, shifts.....	52	54	53	54	21
Shoveler, shifts.....	80	86.25	86.5	82.5	33
Other labor, shifts.....	65	60	64	65	25
Total shifts, underg'd labor	249	254.25	256.5	255.5	1 01
Total shifts, surf labor (a)..	105.5	106.12	107.5	109	42
Total shifts.....	354.5	360.37	364	364.5	1 44
Tons per shift: per drill.....	34.38	35.98	38.30	34.53	3
Tons per shift: per shoveler.....	22.34	22.52	23.48	22.61	2
Tons per shift: per man undergr'd...	7.18	7.64	7.91	7.30	
Tons per shift: per man, total labor.	5.04	5.39	5.58	5.12	

(a) Includes mill-men.

Wisconsin zinc district. Deposits of ZnS and PbS occur in pitches and fl limestone; some are so irregular that mining is practically gophering; larger de are commonly mined by breast stoping. Orebodies are from 5 to 70 ft high, 20 t ft wide, and 500 to 7 000 ft long; roof is generally solid, thick-bedded limestone; de lie at depths of 75 to 200 ft. George (162) gives following d breaking ground: Fig 135 shows method; in headings 6 to 7 ft 8 to 10-ft holes are drilled from umn; heading is kept 15 to 2 advance of benches, which are 10 ft high; bench or stope hol 10, 12, or 14 ft deep; both p and plugger-drills are used. Ac ing to character of ground, a mac man drills 50 to 100 ft of hole per shift in headings, and 30 to 60 ft in stope work headings, at least 1 ton of ore should be broken per ft of hole; in stopes, up to 1 per ft; aver varies from 1.5 to 5 tons, depending on height and width of stope and ence of openings at the top " flat " of ore in the heading. Fig 136 shows plan of hea

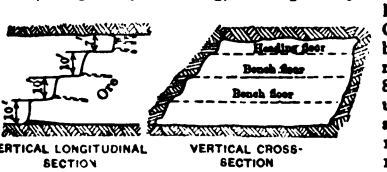

Fig 135

holes 1, 2, 3, 4, which break out hatched areas; C, C', C''. Fig 137 is a cross-sec of an irregular ked out as a breast stope, g mining on 2 levels. In bench are worked simul- ing connected by an in- A, or in rock at right of narrow orebody, the roof over entire width; for would be left (Fig 129).

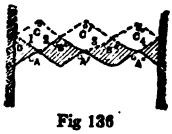

Fig 136

in breast stoping is by pillars; in low-grade as calcareous shale beds used for making d uniform size of pillars are feasible and n Missouri lead and zinc mines.

cing of Pillars, Breast Stoping

Distance bet pillars, ft	Percent ore-body left in pillars	Thickness of ore-body, ft	Depth of ore below surface, ft	Authority
30	15	.....	.....	(a)
20 to 50	17 to 35	0 to 70	300 to 600	(b)
.....	10 to 15	.....	.....	(c)
20 to 35	10*	.....	.....	(d)
30	7*(k)	10 to 40	.....	(e)
25 to 50	12*	6 to 26	75 to 250	(f)
30 to 50	17*	.....	.....	(g)
40	13.5*	16 to 20	240	(h)
40	10*	9 to 10	155	(i)
.....	10	5 to 70	75 to 200	(j)
20 to 50	17*	5 to 40	100 to 200	(l)

conditions in district in 1914 (55). (b) C. T.). (d) R. D. O. Johnson (164). (e) A. H. (g) E. H. Leslie (174). (h) J. H. Polhemus systematically in good ground; areas of poor aller intervals than in table (175). (j) W. L. * Approx aver, computed from data in table.

of pillars vary widely with local conditions; pil- ground and are left where possible in low-grade with regard to haulage tracks to avoid awkward roof between pillars cause serious danger and ons of deposit, slabs are supported by vertical nd is removed from heading roof before bench roof-men are constantly employed in Mo lead areas near stope faces or over traveling ways; n bars and gads; this costs more than blasting ng; holes drilled by hand or plugger- or piston- pport men in trimming high backs, see (163). d by accumulations of water under pressure in ines vertical holes ("drain holes"), 8 to 10 ft very 8 to 10 ft, to allow water to escape (164).

size (Fig 130); often roughly circular in top and bottom to increase area of support 1). Fig 138 shows another means of deal- scales on exposure to air; it is feasible only

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for light pressures and in strong, low-grade ore. Where roof is fissured wi joints or cracks, stoping may unkey large slabs; this danger is partiall by a systematic staggered arrangement o

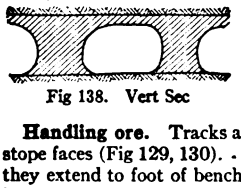


Fig 138. Vert Sec

In some S E Mo mines, ore occurs in ov beds, separated by barren limestone. worked from separate levels, care being t the pillars in upper and lower beds shall b over each other.

Handling ore. Tracks are laid on floor of deposit, from point of en stope faces (Fig 129, 130). In thin deposits, tracks run into headings; they extend to foot of benches. Where more than 1 bench is worked, ing ore is shoveled to foot of lowest bench; general slope of benches is to minimize handling. Broken ore is shoveled into cars, and tramme by hand, or by animal or mechanical haulage, depending on distance an in small mines, buckets are mounted on light trucks, reducing cost o and hoisting equipment. Air hoists are used to haul cars up ste caused by local rolls in footwall.

At some Joplin mines, shoveling plats of 2 by 12-in oak plank are laid at foot over 16 ft high. Attempts have been made to use steam shovels of Thew t ated by compressed air or electricity, for loading into cars (160). For use Whalley machine at Flat River, Mo, see Art 92. H. A. Guess (55) states th men shoveling from rock floor into 1-ton cars averages 18 tons per man-day i Long handle, round point, No 2 shovels are used; a 1-ton car takes 130 sh 15 lb per shovel; aver cost of loading in 1914, 13¢ per ton. Records for Jop on contract work vary from 18 to 22 tons per man per 8 hr, shoveled into buc tract prices for 1913, 4 to 8¢ per bucket of 800 to 950 lb (111). Contract prices work in Wis zinc district in 1915 were 5 to 7¢ per bucket of 1 000 lb (162). See

Percentage of ore left in pillars depends on following factors: (*a*) CH OF ROOF determines max allowable width of heading which will stan ported, and therefore the distance between pillars; (*b*) CHARACTER O a soft floor requires pillars of large area, or closely spaced, so that thei power will not be exceeded; (*c*) STRENGTH OF ORE determines minimu of pillar to withstand pressure of overlying rock; (*d*) DEPTH OF DEPOS mines total wt to be carried on pillars, where deposit has such later that the overlying rocks do not arch, and become self supporting (see Su Art 115). These factors are interrelated; their effect cannot be defi termined in advance in new districts; it is obvious that the percentage in pillars may be fairly constant in a given district, but vary widely in localities. Table 33 gives percentage losses in pillars, representing e of different engineers in Missouri lead and zinc districts.

Recovery of ore from pillars in a systematic way is rarely attem mining method like breast stoping is justified by its simplicity and and because, where properly applied, it yields a max profit in spite of of pillar ore. Use of method should be confined to low-grade deposit value of ore in pillars will not repay cost of replacing them by artificial

At Joplin, in some mines leasers clean up floors and pillars in worked-out area is not systematic; as pillars are trimmed as far as men dare, several disastr have occurred. Attempts have been made to take out pillars, starting at b and retreating toward shaft; such work is costly and dangerous; the caving of r results may destroy the remaining pillars and may extend over large areas. deposits, where ore is soft and weak, stopes are small and pillars contain as 50% of orebody; attempts to mine the ore in pillars have resulted in loss or v profit. Following methods have been tried in these mines for pillar recovery: timber cribs were built in stopes around pillars, to support roof while pillars w mined; timbers failed, much ore removed was mixed with waste and much

mine, in 1914, an attempt was made to sup-
lars were mined. 6-in churn-drill holes were
square were built under each hole; concrete
with a gasolene-driven mixer poured concrete
l tamped it. Hitches were formed near top of
were placed to support roof between; where
oints by
roof and
5 in and
ing pur-
oncrete;
6 bulk-
nd 25 ft
lable on
9 shows
ody was
aft was
e dotted
k below
of each
handled
ough to
through

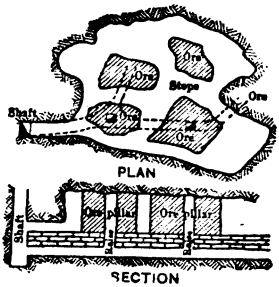


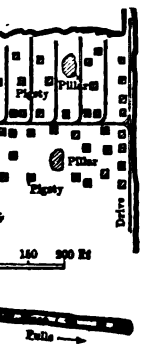
Fig 139

ethod: cost of subdrifts and raises, unavoid-
ars and incomplete extraction (166).

in Flat-dipping Beds

rried parallel to either dip or strike; in
s called DIP WORKINGS; as dip increases,
becomes the same as that in underhand
ethod is limited to deposits with strong

frica. W. L. Honnold gives following
of 7°, is broken by dikes and faults;
; both reef and hanging wall are harder
at cohesion and tenacity. Ore is in large
irregular shoots, gen-
erally elongated along
dip; one measures 700
ft on strike by 4 000 ft
on dip. Stope widths,
57 to 70 in. In 1915,
workings had reached
a vertical depth of
4 100 ft (for character-
istics of Rand banket
see Art 41).



Method of mining (Fig 140). There are 2 vertical shafts, 4 400 ft apart in direction of dip, connected at bottom by an incline on the dip. Levels are 500 to 800 ft apart; varying dips give
inzes connect levels at points near middle of
el to dip; cars are run to face and handled
d timber cribs support the roof.

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Breaking ground. 2.75-in and 3.25-in piston-drills are used, the smaller fo faces under 5 ft high. 1 white man on contract supervises 4 drills, each being 2 Kafirs; 1 drill carrier serves 2 machines. Hammer drills are being introduced fully (1915) for uppers. All holes are drilled wet where possible. Generally, on th much stoping is done by hand. Machines are used at Brakpan because: (a) nati is often scarce, and is inefficient for flat or upper holes, required by the flat dip; is wider than in most Rand mines; hence, the increased stope width inevitat machine drills does not break excessive amounts of waste (see Art 41).. 57% of e used is blasting gelatine (92% Ngl); rest is gelignite (60% Ngl). Large drills t aver of 11 tons per shift; small drills, 7 tons; aver stope widths, 70 in and 5 spectively. Aver stope widths exceed aver reef width by 22 in.

Stoping. Fig 140 shows face of a well opened stope. Work begins at corner block, at both top and bottom of a winze; at first diagonal cuts are taken, giv appearance of overhand stoping (Art 36) at bottom, and underhand stoping (at top, of winze; but when face has advanced 100 ft it is parallel to winze (Fig

Support of roof is by pillars of ore and 10 by 10-ft timber cribs (pigsties) being sometimes filled with rock. Both cribs and pillars are irregularly spac respect to faults and haulageways. The strong quartzite roof settles gradually; faces advance right and left from winze. Eventually pillars are robbed, much crib timber is taken out and the stope allowed to cave. Pack-walls (Fig 170) pro levels, and must be maintained through worked-out areas.

Handling ore. A track is laid in the winze, with a small hoist at top and halfway down. Lateral tracks are run from winze track to the stope face ever Ore from upper half of stope is hoisted to level above, that from lower half is to level below (Fig 140). Winze tracks are shifted toward stope face as roof set requires. Cars hold 1 ton (20.5 cu ft); native muckers receive 8 to 10¢ per car and handle about 6 tons per 8 hr.

Park City, Utah. Fig 141 shows the dip workings of Silver King Coalit J. Humes gives following data (134): silver-lead orebodies occur as replac in limestone; thickness, 1.5 to 25 ft; dip, 10° to 30°; oreshoots are irreg outline and make off from fissures which gave access to mineralizing solu

Method of mining. Exploratory drifts, in the ore-bearing bed, follow the when a drift cuts ore, stopes are started to the dip and carried as far as broken r be shoveled out. If ore co a small hoist is installed. is laid on floor of stope an cars run to stope face. If is thin, enough of footwall out to admit car. If the rary inclines strike barre the tracks are turned and the ore. In orebodies of la eral extent, small drifts (s els) are run to right and le the main incline. Stopin ceeds both up and down di sub-levels, in which run sm

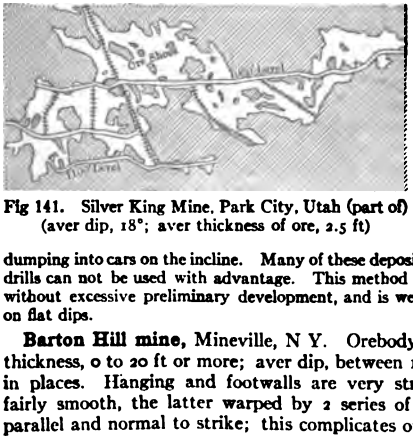

Fig 141. Silver King Mine, Park City, Utah (part of) (aver dip, 18°; aver thickness of ore, 2.5 ft)

dumping into cars on the incline. Many of these deposits are so flat and thin that m drills can not be used with advantage. This method permits mining irregular ore without excessive preliminary development, and is well adapted to handling brok on flat dips.

Barton Hill mine, Mineville, N Y. Orebody is a bed of strong magn thickness, 0 to 20 ft or more; aver dip, between 15° and 25°, rising to 35° in places. Hanging and footwalls are very strong tough gneiss, the f fairly smooth, the latter warped by 2 series of ridges, the axes of whic parallel and normal to strike; this complicates ore handling.

Method of mining. Deposit was first attacked at several points along outc breast stoping to the dip. Mine cars were hoisted to surface on inclined tracks o

s for roof support (note that inclines were swung temporary horiz tracks were also laid to reach

oping
lines;
abows
drift-
stope
e laid
g dip;
into
oisted

Fig 142. Barton Hill Mine, Mineville, N Y (part of)

nderground electric hoists. In places, the stope ear incline, to give headroom for dumping into Brakpan mine in that the roof is supported permanently by pillars of ore; irregular footwall makes stope tracks crooked; steeper dips prevent handling stope cars to the level.

Breaking ground. Following data were furnished by the management: holes, 4 to 6 ft deep; in 1912, an aver of 26.58 ft of hole was drilled per shift with 2.75-in piston-drills on tripods; records for 7 mo show aver production of 15.53 tons per drill-shift in stoping and development, and 19.91 tons per drill in stopes alone. In 1913, prizes were offered for best footage drilled per shift. In a 14-weeks' run, 1st, 2nd, and 3rd prize-winners drilled averages of 41.2, 36.36, and 35.14 ft hole per shift; aver of next 12 men after prize-winners, 27.17 ft. Hammer drills, substituted for many large machines, average 40 to 50 ft of hole per shift. 40% dyna-

N Y

t 1.03 lb per ton ore; aver for first 4 mo of 1916, r man-day) for May, 1916, was: per man under-ons; per man stoping, 10.10 tons. Drilling was er of 23.75 tons ore was produced per drill-shift.

nd Pillar Methods in Beds

rs many different methods of cutting up a ich sense it includes breast stoping (Art 30, r from breast stoping mainly in being more ectangular and laid out with almost mathe- be left for permanent support, or recovered olliery methods applied to mining salt, iron plified in these deposits because of less rigid ils of coal-mining methods, see Art 102 *et seq.*

on by room and pillar are flat or slightly character, and of large area. Cheap, abun- roof and floor are necessary if permanent is recovered by robbing, a very strong roof ause trouble by dropping suddenly (Art 103).

ion of coal mining terms, see Art 102): Bevis ed of salt 18 ft thick is mined; depth, 1 065 ft; (167) describes methods in 1911: From a vertical *B*, were driven (Fig 144). Rooms *R*, 50 ft wide entries and advanced indefinitely. Pillars, 50 ft 80 ft by break throughs *C*. Roof is good and

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no timber is used; pillars stand well; to avoid contaminating salt with sl layer of salt is left on roof and 4 to 6 in on floor. Fig 145 shows mode o ground; bed is mined in 3 sections; bottom 7 ft is undercut with mechanic depth of 3.5 ft, then holes are drille 22 to 27 holes are required to break a room. Holes are next drilled to overlying 5 to 6-ft slice; top layer drilled, but left in place until face o advanced 100 ft, when it is all blast 50 to 75 holes being sometimes fir neously, breaking as much as 3 000 t Compressed-air augers are used, e about 100 ft of hole per hr; blasti with 20% dynamite and electric fuz are not robbed. ROYAL SALT CO, Kan, mined a 10-ft bed at depth of similar methods. Whole height o broken in one operation; a good part of bed made undercutting unneces

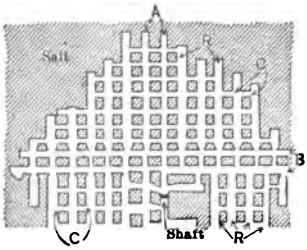


Fig 144. Bevis Salt Mine, Kan (part of)

of rooms having been opened by a V-cut, vertical rows of side holes were drill in a row breaking from floor to roof. Air-driven post augers were used; bits, and 2 to 8 ft long; aver speed of drilling, 3 ft per min, including setting

WABANA, Newfoundland. Practice here illustrates application of room and pillar to flat-dipping beds. Three workable beds of hematite outcrop on Bell Island, averaging 6, 8, and 16 ft in thickness (7); dips, 8° to 10°; roof, strong sandstone. Deposits are opened by slopes 0.5 mile apart; main levels are driven each way from slope at intervals of 250 ft. At points along levels, headings are driven up the dip; on each side of headings rooms are opened in direction of strike. Rooms are 15 ft wide with 20-ft pillars. Face of a room, 15 ft wide by 8 ft high, is broken by 12 7-ft holes; a 3-in piston-drill makes 75 to 80 ft of hole per 10 hr, breaking 44 tons; charge per hole, 3-4 to 3.6 lb 50% dynamite, or, 0.75 l No data are available on pillar robbing. Cars are trammed in rooms by hand;

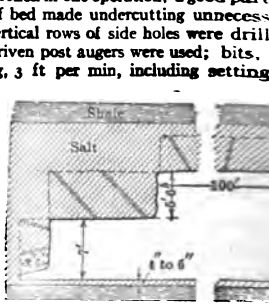


Fig 145. Bevis Salt Mine (vert, through room)

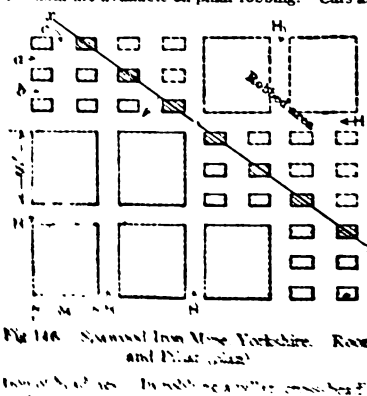


Fig 146. Stanmoil Iron Mine, Yorkshire. Room and Pillar (plan)

hoists lower loaded cars to which rope-haulage is used. on mucking, see Table 18 (WOOD MINE, Yorkshire, Engl from L. Mayer (169). Depo ular bed of oölitic iron ore, 7 dip 0° 48'. A 2-ft bed of in stone overlies the ore, and w mined with it; roof and floo max depth of bed, 700 ft. 1 been worked over a large are ing headings *H* (Fig 146), which cut bed into pillars (cf Bord and pillar method This extracts 37.5% of tot lars are robbed by a retreat too, starting at any point w not prevent maintenance o ways, face 17 of an area bei makes an angle of 40° to 45°

[illegible] are driven through [illegible] This forms 6 small p [illegible] are then broken down,

of around them. Double-hatched areas indicate
ll pillars are left to aid in controlling subsidence of
ion. Posts are pulled systematically to make the
thus recovered. When robbing begins in a new
and gives trouble from sudden falls; but after the
y controlled.

and Stopes, Narrow Veins

derhand stoping are shown in Fig 147, 148.
aking ground and handling ore and waste.

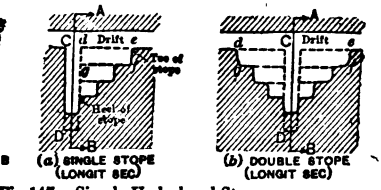
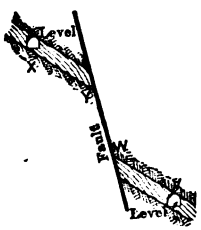


Fig 147. Simple Underhand Stopes

ep-like faces, converging toward bottom of
derhand work. Stopes like (*a*) Fig 147 are
OPES; those like (*b*), DOUBLE STOPES. Terms
EEL sometimes designate top and bottom
Fig 148 shows a better plan, known as the
METHOD. Stope is worked around a raise
etween 2 levels. Where work is carried on
147, all ore except that from first slice is
into buckets and hoisted out to the level
n Cornish method broken ore falls through
e level below, and is loaded by a chute into
stopes are self draining. This method is
ad of that in Fig 147 wherever possible.

nd stoping as in Fig 147 is convenient for mining
a level without preliminary development, but its
s cost of mining by the cost of hoisting ore out of
Though such stoping is not used for system-
on a large scale, it is useful: (*a*) for mining small,
often such deposits can be explored only by stop-
ring (Art 29); (*b*) for mining isolated or faulted
here development open-
would be too costly or
work, lacking funds for

ows underhand stoping
a Copper Co. Stoping
t up in the oreshoot 25
rking floor is kept cone-
logs, over top of chute,
g and clogging it.
for methods suitable



28). Fig 149. Cross-sec

toping, carried on as in Fig 147, 148, destroys
unication with parts of level beyond stope

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break the arch pillars into the open stope below, beginning at boundary and toward shaft; in old workings this is apt to be dangerous.

Examples. DUCKTOWN, TENN. Orebodies are large, lenticular masses c phides, carrying 1.5% to 2% copper; aver width, 60 ft; dip, 70° to 80°; ore, tough; country rock is schist; both ore and walls stand well. At Burra B an inclined shaft is sunk in footwall about 100 ft from orebody; level inter at each level a crosscut is driven to orebody and drifts run in each direction; put up from drifts at intervals of 50 ft or less to within 20 or 30 ft of level abo ings are opened from tops of raises, and benches are broken in steps 6 to Details of development vary. On upper levels a raise was made near point w cut entered orebody, and stopes started each way from it (180). Since 1910, t has been replaced by shrinkage and other methods. ARMINIUS MINE, Mineral, tabular mass of pyrite; width, 8 to 60 ft; dip, 60°; the strong slate walls stan no timber was used (181). Level interval is 50 to 100 ft; drifts, 10 ft wide. put through to level above at a 45° angle; heading is opened 12 ft below level worked in 4 to 6-ft steps. A pillar 12 ft thick is left over back of lower level dropping ore to haulage level pierce this pillar at 25-ft intervals. Pillars are necessary, amounting to 33 to 50% of the orebody.

35. Underground Glory-hole Method

General. This method, also known as UNDERGROUND MILLING, is tation of opencut methods of same name (Art 100). The top of a raise i in all directions, making a funnel-shaped opening (GLORY HOLE, or MI which is widened and deepened by underhand stoping. Face of a glo usually carried in benches, forming in plan rough concentric circles around the raise as a center (Fig 156). General slope of face must enough for broken ore to move by gravity to the raise.

Often, before milling begins, sub-levels are run through the orebody to con at vertical intervals of 25 to 50 ft. F. W. Sperr states that, for economical raise interval on each sub-level should not exceed 25 ft. This condition c by inclined raises branching from the main vertical raises. Economic interv raises and sub-levels depends also on hardness of ore. (For full discussion o raise method, see Bib 182.)

Applicability. Method is limited to large orebodies, as masses veins, with strong walls and ore. Back and side walls of a glory hol inaccessible as stoping progresses, with great danger from falling sla but strong ground.

Fayal mine, Minn (183). Hematite, underlying a cover of 65 to 90 ft drift, was formerly mined by underground milling in rooms 24 ft wide, 60 ft hi to 100 ft long. A drift was run under center of proposed room, with raises ore at about 50-ft intervals. A wide drift was then run over top of room an with saddle-back stulls (Art 36) and heavy lagging; under this protection ore into raises. Walls of finished rooms were vertical; pillar widths, 23 ft. Wh was mined out it was filled, intervening pillars being mined by top slicing (Art of timber for supporting backs of glory holes is rarely feasible.

Application of method to inclined deposits is limited to those having dips ste for the footwall to clear itself by gravity. Ore breaking and handling are c benches being carried, with deep holes. Since no sorting is possible in stopes, must be uniform. Following examples show applications to different types o where all or most of ore mined is from glory-hole work. The method is also auxiliary for cheap mining of rooms between pillars mined by other methods, a mine; see also Ohio Copper Co's method (Art 81).

Section 21 mine, Marquette Range, Mich (153, 184). Fig 155, 156 sho of mining a steep-dipping lens of medium hard hematite; footwall, diorite jaspilite. Shaft is in footwall. To open a level a crosscut is driven from foo ing; walls are then followed until drifts connect (see plan, Fig 155); crosscuts between foot and hanging-wall drifts at intervals of 50 to 60 ft. Footwall rais

development has advanced far enough for a with other work on level; other raises are as

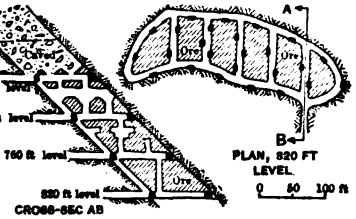

Fig 155

rted near top of each raise, leaving a 6 to 10-ft rts the level until ore above has been removed. east stope is milled into raise, and milling con- ward to lines *abcef*, where ore will no longer slide ope. Robbing the V-shaped pillars over level begins from raises *r*, the pillars being thinned ey will just support caved material above. Ore to open stopes below. Holes are then drilled in illars and in level-pillars *L*, all being fired to- ctions, beginning at boundary and retreating. rom pillars is drawn through chutes *b* and *e*; lost by mixing with waste, which falls when roken. At this mine a rigid geometrical plan llowed, because of numerous intersecting dikes. is made to locate development raises to reach ion of pillars on level above. This method has omical and safe; no timber is required except nd ventilation is good. It is stated that the per- of ore through contamination with waste is low.

nine, Ariz. P. B. Scotland gives data for 1910 odies, of low-grade oxidized and sulphide copper ckworks in granite-porphyry, containing shoots ss fissures. Where ore is hard and stands well, raise is put up to top of ore and extended to ning begins at 50 ft above level. Ore is milled ill hole is widened to afely. Holes are deep, ary to roof from shocks. with waste through the p as filling progresses; rly attacked.

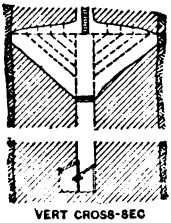

Fig 157

Contact metamorphic rite, and specularite, in n crystalline limestone ody is a lens, 2 500 ft r 900 ft wide; dip at o 15° to 30°; footwall, ercial one, ore grading Ore is strong, of very oz Au, 0.3 oz Ag. R. H. Allen (120) and C. M. sit is opened by shaft and tunnels; level interval, are run on each level in direction of strike; 7 by drifts at intervals of 25 to 60 ft, usually 45 ft. and the raise is widened until it connects with es is taken out for height of 10 to 15 ft, above

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which the raises are carried up 20 to 25 ft wide. At 60 ft above the level, an nection is made between raises, which are continued to level above, leaving of pillars throughout the stop holes are next opened aroun raises, ore being broken in Drilling is done with 3-in ai piston-drills; in preliminary w umn mounting is used and hol 10 ft deep; in bench work, ma tripods drill holes to 20 ft dee also work in connections betw No stope timber is used, pil left between glory holes to sup In steeper parts of deposit, p nearly normal to walls; in fla wide portions, raises are made and when hanging wall or leve reached, glory holes are beg vertical pillars between them. shows part of workings betweer 3rd levels. In 1909, 500 men (4 underground) mined 3 300 tons

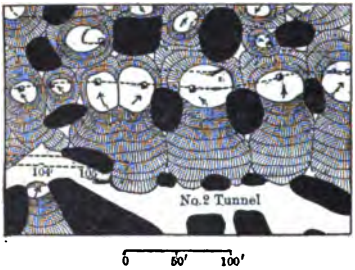


Fig 158. Granby Mine. Glory Holes. (Pillars are black; slopes of glory holes hatched to show contour; level below in dotted lines)

output per man underground per day, 8.25 tons; output per miner, 20 tons First underground mining, done with square-sets, showed that orebody was le uniform, and of enormous size, and that ore and hanging were strong and wou over large areas, conditions which led to adoption of the open stope method d This work is closely allied to chamber workings (Art 42). For further detail, se

36. Open Overhand Stopes, Narrow Veins

General plans of mining are shown in Fig 159 to 162. Overhand st practically inverted underhand stopes (Art 33), miners working upward und the ore to be broken. Details vary with modes of breaking ground, handling ore, and supporting walls and men; dip of vein and distribution of ore are the principal factors causing variations. Open stopes are limited in general to deposits having strong walls and ore strong

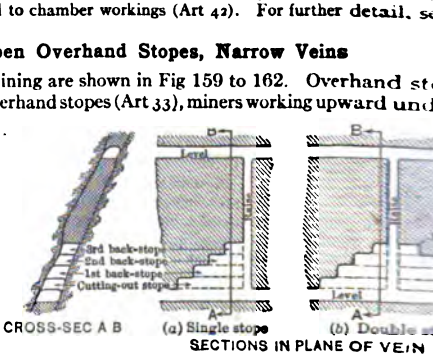


Fig 159. Open Overhand Stopes

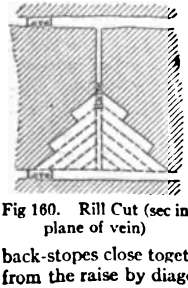


Fig 160. Rill Cut (sec in plane of vein)

enough to stand unsupported over back of stope. are preferably started from bottom of a raise, suc horiz slices being taken (Fig 159). First slice, direct level, is the CUTTING-OUT STOPE; succeeding slices 2nd, 3rd, etc, BACK-STOPES. Stopes may be SING or DOUBLE STOPES (*b*) (Fig 159). Terms TOE and sometimes designate bottom and top corners of stop The drift and cutting-out stope may be excavat gether making a DRIFT STOPE (Art 40). RILL STOPE a longitudinal sec like an inverted V (*b*, Fig 159); in faces may be produced by keeping the faces of succ back-stopes close together, or by using the RILL-CUT, in which the stope is c from the raise by diagonal slices (Fig 160). For choice between horiz and

more useful in filled than in open stopes
r 40° to 45°. In FLAT-BACK STOPES (horiz
anced in a general line parallel to the level,
topes far
between
The num-
aneously
ack, and
n, deter-
a stope.
, miners
ith each
ERHAND

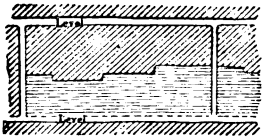


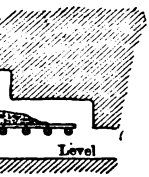
Fig 161. Flat-back Stope (sec in plane of vein)

ntermediate forms between this and Fig 161.
opes of same form in wide orebodies.

be opened in the back of a drift, without first
of vein in Fig 149, or in case of small irregular
e the level. Raises insure natural ventilation,
s, and provide entry and facilities for lowering

pical development in a vein, providing
ng.
8).

ls. Broken ore from an open overhand
nence, a barrier is necessary between level
and stope to protect men and keep broken ore off haulage tracks. This is more important on steep than on flat dips; protecting the back of level also allows broken ore to be loaded through chutes (Art 91) and avoids shoveling. Practice varies with dip and width of deposit,



VEIN
l Timbering

STULLS usually protect back of level in
ugh to support them; Fig 162 shows com-
e is taken before
n stulls and back

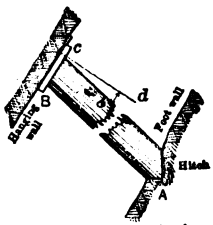


Fig 163. Stull Timbering

to 24 in or more;
6 ft; occasionally,
Art 40). LAGGING
ks (see Timbering
n open stopes, a 3
g to protect it from
en stope is finished.
all with a moil or
h is flattened as at
Hitch may be only
twall may require hitches 6 or 8 in deep or more;
tting hitch. Head of stull is square and to dis-

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tribute pressure rests against HEADBOARD *B* (sometimes called a cap) of 2 t
where necessary, wedges are used between headboard and hanging. Headboa
blocking are compressed by initial creep of groun
stull from splitting. Stulls are set at a steeper a
normal to the hanging wall. In Fig 163, line
dicular to hanging wall; angle *dcb* s the ANGLE
Setting of stulls varies. W. E. Sanders states
underlie should be about 0.25 × angle of dip
T. Johnson favors a max underlie of 10° for di
over, and a ratio of about ⅛ between angle of
angle of dip on flatter dips (187); usual practi
between these limits. Object of setting stulls wit
is to prevent their falling under weight of ore
to cause them to tighten under small settlement of hanging wall. Foregoing
to placing stulls in both overhand and underhand stopes, as well as to stulls
If footwall is weak, a FALSE STULL may be set underneath main
stull (Fig 164). In steep-dipping or vertical veins with poor
walls, stulls over levels may be reinforced, Fig 165 (186).

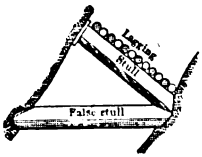

Fig 164. Stull Timbering

Fig 165. St

Maximum economic length of stulls, placed as in Fig 163, varies from 12 to 20 ft, depending on timber supply, facilities and room for handling timber underground, lateral pressure from walls, and relative cost of alternative methods. Usual max width of vein in which single stulls are used to protect levels (also for supporting slabs and men in open stopes) is about 15 ft. SADDLE-BACK STULLS (Fig 166) may be used in veins up to about 25 ft wide. They lack lateral rigidity, and are apt to fail by buckling sidewise; will support vertical wt of broken ore or waste, but do not resist wall pressure; joint at apex must be carefully framed and fitted. They have a limited application, and many engineers obj
under any conditions.

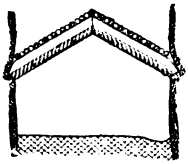
Fig 166. Saddle-back Stull

Advantage of stull timbering for levels is that th
below can be readily broken through to level above
out disturbing the waste and debris which accumul
under exhausted stopes.

Stulls and posts may be used in steep-dipping veins, 12 to 20 ft wide, with
The stull is set with a flat underlie (generally in hitches), with posts under bot
at intermediate points if required.

Drift sets (Art 21) are common for protecting backs of levels under stopes in narrow veins with weak walls; like stulls, they are placed and lagged after the cutting-out stope is broken. HALF-SETS are used where one wall is weak; THREE-QUARTER SETS, where both are weak.

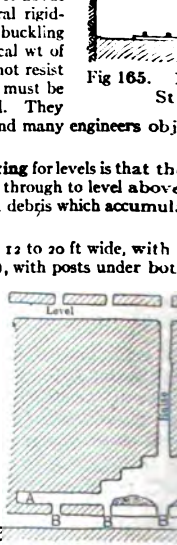

Fig 167. Level-pillars
Stope (sec in plane

Pillars of ore (level-pillars) may be left over level at bottom of overhand stopes; Fig 167, from Truscott (30), shows Rand practice (Art 41). In this case, stopes are opened from a sub-level (stope-drift) *A*, and connected with it by short raises *B*, in which chutes are built; wedge-shaped piles of ore, collecting between chutes on top of level pillars, are cleaned off by shoveling when stope is finished. Pillars may sometimes be recovered prior to abandonment of level. For factors governing choice on the Rand between level-pillars and timber, see Art 41. In open stopes, choice usually depends
ative cost of timbering, extra cost of the sub-level and net value of ore left in p

Packwalls may be built above level on flat dips and holes left in them
for chutes (for detail, see below under Support of walls). Need for protecti

rotection is necessary where dip is less than 20° 0° to 40°, see Art 38 to 41.

rs of ore are left where it is necessary to keep lars may be also required to support walls. See

an 40° to 45°, men can stand on and work upport being necessary; on steeper dips, tim-

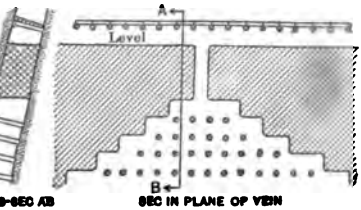

Fig 168. Stulled Stope

rt 37, 38). Square-sets and other timbering ng veins, too wide for stulls (Art 45 to 55).

stopes, in veins worked by open overhand on of open stope to timbering, pillars of ore s are commonest timber support in narrow evels, for economic limits of length). When ose slabs. Stulls do not adequately support ugh off on exposure to air; for such cases *d seq*), allowing use of lagging along walls, not adapted to veins with weak walls; filling ferable. Timbering in general is to support til the stope can be abandoned and allowed port the weight of rock overlying a deposit. Art 37, 38.

veins waste is piled on stulls (Fig 152). Fig rted. H. C. Hoover states (20): "This system emand continuous support; it effects economy in stulls by using waste which accumulates underground; artificial pillars also apply to cases where stulls alone are not sufficient support, and yet where complete filling or square-setting is unnecessary; under propitious conditions they have the comparative advantage over timber systems of saving timber and over filling systems of saving imported filling." Inverted V shape of pillars (Fig 169) allows broken ore to slide to the level without especially built passes; when this system is employed, more special staging must be provided for miners than in square-set or filling systems.

for roof support in flat deposits; for examples, t, but are adapted to temporary support, prior vaste.

ilt on the upper side of levels in veins dipping 30° e hanging wall when ore is completely removed; pillars in stopes. T. Johnson (187) comments should be built nearly at right angles to plane

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of deposit, and not with vertical end walls, which may fail by slumping "upbr
the dip); in long packs, middle portions of walls bulge and eventually fail. S
placed as in Fig 170, with or without lagging to steady the walling until weight c
small timbers serve this purpose. Small waste should be thrown in while walls
built, to form bedding for large
and give solidity. It is advisabl
walls to inside of pack with piec
pipe, rails, rope or boards, partic
corners. See Brakpan mines, Ar
Quincy mine, Art 40.

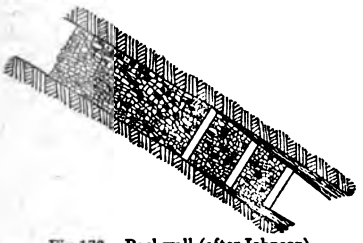

Fig 170. Packwall (after Johnson)

Pillars of ore may be left in
hand stope, where the hanging
stand unsupported between th
where net value of ore in pillars
pay for alternative modes of
Low-grade or waste portions of
shoot are usually left as pillars;
ary pillars of ore (as level-pillars)
mon; use of pillars for permanent support is confined to low-grade orebodies, us
extensive deposits lying on flat dips. For examples see Art 39 to 41.

Control of hanging wall over entire mine, in veins worked by open ov
stoping, may be effected as follows:

(*a*) In veins where ore is in shoots, the barren areas often serve as pillars; su
individual stopes is obtainable by one of the above methods, depending on size
and character of walls, and stopes will remain open indefinitely or cave after th
been abandoned. For permanent support to protect shafts, surface buildings,
hausted stopes may be filled. (*b*) Where ore is continuous over long distan
hanging wall may be supported on permanent pillars of ore. Flat-dipping, lo
amygdaloid copper deposits (Mich) furnish an example of this practice (Art 40).
continuous orebodies, stoping on each level may be carried to property line or to
ore, leaving temporary pillars, often of large size, to support roof. As much as
of the pillar ore is then mined, retreating from boundary toward shaft, and the hang
caves over robbed areas (see Clinton iron ores, Art 39). (*d*) In continuous oreb
good grade, or at great depth causing heavy pressures, levels may be driven to bou
before stopes are begun above them. Ore is stoped in blocks, starting at boundar
being mined before work begins in next one toward the shaft. Size of block is adju
pressure, kind of roof, and speed of work, so that the stope can be kept open with
until finished; the hanging wall is then allowed to cave and work repeated in a
block (see Calumet & Hecla, Art 38). The foregoing remarks apply to open stop
support of walls by filling, see Filled stopes, Art 59–67.

Handling ore. In STEEP-DIPPING VEINS, ore broken in open overhand
goes by gravity to level below. Loading chutes are built in openings i
timbering or pillars at intervals of 15 to 30 ft (for details, see Art 91).
mum angle on which ore will slide depends
on character of ore and footwall; soft ores
often slide less easily than hard (see Bir-
mingham, Ala, Art 39). Angle of friction
varies with ore; an irregular footwall
causes ore to hang; minimum angle, usu-
ally about 40°; varying from 35° to 45°.
In open, overhand work, no ore is stored
in the stope except the small amount left
on level-pillars or to protect level tim-
bering. WING STULLS (Fig 171) (winged
stulls or wing chutes) are sometimes used
to facilitate handling ore in steep-dipping
veins, where all the vein matter is sent to surface. They permit some s
in stopes, obviate necessity for lagging all the stulls, and permit use of

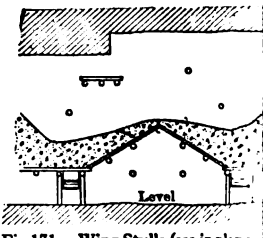

Fig 171. Wing Stulls (sec in plane

can be put in any stulled stope, preferably
s not required for wall support; their use
temporary plat-
Liberty Bell and

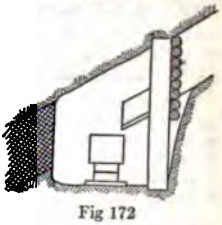

Fig 172

of ore from stope
by shoveling. In
d on these dips.
eled into cars, or
gh enough for ore
th a low lift (see
West Australia,
172, drifts being
for cars below lip
filled stopes also (83). In flat deposits (less than
veling and cheapen stope transport (see Art 92).

storing waste sorted from ore in open over-
g it on stulls (Fig 169).

hese stopes are not suited to ores containing
out underground. In flat stopes, some waste
bs, or it may be left on floor in middle of stope,
this point is usually prohibitive, and if it is left
with transport to the level and becomes mixed

en Overhand Stopes (Stulled Stopes)

d with stulled stopes are narrow and nearly
e, tuff, breccia, etc, and strong, often high-
Wolcott gives following data (142). Fig
173 shows method of stoping and timber-
ing in veins up to 8 or 10 ft wide, where
back-stopes are broken with uppers. A
flat-back stope is carried, with stulls 5
to 7 ft below back, and enough lagging on
top row for a working platform.

Formerly (1906) 2.25-in piston-drills were
used and height of slice taken in one back-
stope was 4 ft. Permanent stulls were say 7
orizontally; 2 slices were taken across a stope
t of the broken ore was left on lagging, or tem-
o men up to the back while mining the second
used,
es are
nay be
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s were
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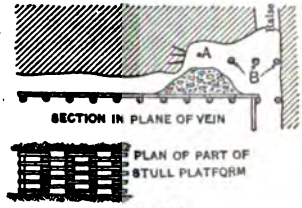

Fig 174

for machine men to stand without staging, and
injuring stulls below. A flat-back stope was
l, lagging at the rear was removed and muck

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dropped down, large boulders being broken before they were allowed to fall. was reused on higher rows of stulls (*B*, Fig 174).

Liberty Bell mine, Telluride, Colo (131). Quartz vein, aver width, aver dip, 57°; wall rocks, andesite, tuff and breccia; pay ore occurs in of variable size. Much mining is done in stulled overhand stopes, practice and details varying with dip and strength of hanging wall.

C. A. Chase gives following data for 1911. Ore is broken with vertical or pitched holes, drilled with stope-drills or hand augers, miners working on a part near back of stope. Wing stulls (Art 36) deflect broken ore to chutes at 25 intervals. Stulls, 8-in diam up, are placed 5 ft apart in floors 7 ft apart. Worki is chiefly of 6-in round timber; stoping floors, of round or split lagging, 10.5 For steep dip and strong hanging wall, shrinkage stopes (Art 68–70) are preferab

Fig 175

Tonopah Mining Co, Nev. Much mining wa in the early history of the camp in stulled stopes occurs in quartz veins carrying gold and silver. F is a cross-sec, showing former method used in upp of Mizpah vein; width, 5 to 30 ft; dip, about 70°; w is dry andesite, which will stand for a long time. broken averaged $15 to $25 per ton; no sorting done ground. Entry was by vertical shaft; level interval,

First row of stulls was horiz, 8 or 9 ft above level, a ported by 2 or more posts on sills wedged between walls of drift. Succeeding rows were set as usual at vertical i of 6 to 15 ft. · Plank lagging was used for platforms protect stulls at level; wing stulls (Art 36) were often em Stulls were 8 in or less in diam, braced in wider part vein by struts to stulls above and below (Fig 175). stope broke through to level above, the sills there were up from the stulls below. Some stopes have remained o 8 years; many are filled with waste to protect later workings. Recent practice square sets for wide orebodies (110, 188).

38. Examples of Practice, Open Overhand Stopes (Stulled Stop

Calumet & Hecla mine, Mich (67). Calumet conglomerate bed, d to 39°, width 10 to 25 ft, has been worked a distance of about 2 miles al strike. Shelly diabase hanging wall gives trouble, especially at the great of present mining (Art 17). Tenor of ore mined is 1.75% to 2% copper For ore occurrence, see Art 40.

General plan of mining in the conglomerate bed differs from amyg practice (Art 40), because of the weak hanging wall and great depth of a retreating system is required by these conditions and also by high te tures at depth, which necessitate good ventilation at working face; in a ing systems, much of the air current is apt to short-circuit through old close to shaft. Stoping is begun either at the boundary or midway betwe jacent shafts, retreating toward shaft; small blocks are stoped rapidly open temporarily by stulls and then allowed to cave; the only pillars l the shaft pillars, extending 50 to 75 ft along strike on each side of shaft; are robbed after hanging wall has settled around them, beginning at and retreating up the dip.

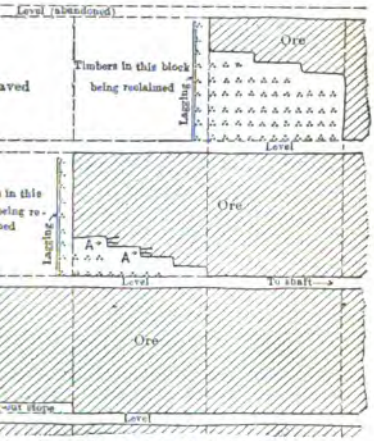
Development. For practice in locating shafts, see Art 17, 18. Le terval is about 100 ft; higher stopes can not be worked out quickly eno allow control of hanging wall in lower portions of stopes.

Drifts are 8 by 8 ft, and follow the footwall to take advantage of better grou well-marked slip; double-track drifts in Red Jacket workings are 7 by 11 ft; drif

ficult and costly to maintain. A 7 by 7-ft raise is n the end stope on the level, for ventilation and to as stoping progresses there is enough open space ese purposes.

stepped-face stopes are carried, a cutting-out th of lode for full length of stope; successive

d out and caved



Stoping, Calumet & Hecla Mine (diagrammatic sec in plane of vein)

een stoped is lagged to prevent waste (from caving e. Stoping in any block on a level begins as soon mined out (Fig 176). Alternate method would be slices; objections here are: (*a*) pressure develops g in a stope face running up the dip would make tical face causes more work and delay in carrying ND. Drilling is more difficult in conglomerate me is also lost in barring down and setting up. conglomerate mines average 24 ft of hole per shift ed as at *A*, Fig 176; aver burden, 3 ft, ranging s of holes to break a 6 to 8-ft slice off a back-stope; e widths; in S Hecla mine, 2 holes are sometimes lower holes are drilled wet; back hole, dry. In wide is advanced on footwall and ore in back is oof-holes are necessary, men and drills stand on An 8-ft hole takes 6 to 8 sticks 45% dynamite. l (189) which has been substituted for inclined ups of 3 (batteries); upper stull of battery is cut -plate, placed parallel to strike between adjacent t slabs, lagging, running up the dip, is placed on ood, the wall-plate is omitted and a block wedged nbers are used, diam, 16 to 30 in; the short stull es are used in bottom of stope, smaller for upper o in diam, flattened at ends to give good bearing edges are large, so that blocking pieces over bat- ised into stope by an air hoist (Fig 177). Main tom to next lower battery, to prevent knockin-

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down by blasting before weight of hanging comes on Often, the short stull is b of pieces recovered from old stopes. Batteries are spaced 6 ft apart along strik 6 to 9 ft along dip; they have been used at Tamarack mine in stopes 42 ft wide; i stopes, stulls are blocked at bottom and braced at middle with struts or wire rope. T

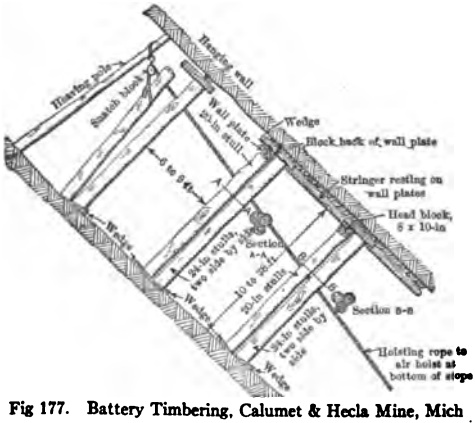

Fig 177. Battery Timbering, Calumet & Hecla Mine, Mich

GANG comprises a boy to run hoist, timbermen; on ar age this crew sets a 2 batteries per shi Hecla, in stopes 14 ft wide; at Red where hanging poorer and stopes to 25 ft wide (aver aver duty is 3 batt 2 shifts (sometimes battery per shift). (hemlock) in 1912 to $8 each. Grou stulls in batteries use of smaller and c timbers. Batteri well adapted to fl and conditions in conglomerate mine in stopes much ove 25 ft wide they ar tionable. PARTIAL ERY OF TIMBER is possible if stope length is such that excessive weight does no on lower part of stope before the block is mined out. Amounts of timber recovere widely; in some stopes up to 20% of the total is reclaimed. Recovery is done by b footwall under batteries with pop shots; before shooting, rope from the hoist is tied timbers so that they may be pulled out without danger to men; much lagging is under loose rock and lost. Timbermen receive $1.50 to $1.75 for recovered stulls 14 in good condition; they work at robbing either after shift or after finishing their timbering. HANDLING STOPED ORE. Broken ore slides or is shoveled to first batteries above level, between which chutes are built and ore scraped from the cars; bottom of chute is sometimes lined with steel plate, or entire chute is l hardwood. In S Hecla, 1st or 2nd row of batteries above level is lagged from 4 ft above footwall, to prevent occasional falling roof slabs from injuring trammer

39. Examples of Practice, Open Overhand Stopes (Support by Pillars o

Clinton iron ores, Birmingham, Ala. The orebodies are beds of hem occurring as interbedded members of a series of sandstones and shales. extend many miles along the strike and have been opened over 2 000 ft o Leaching has removed lime from upper parts, producing an enriched s for varying depths down to 400 ft on dip; below this, ore is hard. W. R. (190) and E. C Eckel (191) give following data: thickness of beds, 10 ft; dip, 8° to 50°, but usually between 18° and 30°; hanging wall, largely stone; footwall, shale.

Method of mining. Entry is by inclined shaft in the deposit; level interval 65 ft. Drifts, 12 to 15 ft wide, run from shaft each way 60 to 100 ft; thence, the tinue as drift-stopes (locally, rooms), extending up the dip 20 or 30 ft above the Raises, connecting levels at points 60 to 75 ft away from shaft, serve as manwa airways and also inclose pillars to protect shaft; other raises connect levels at intervals (Fig 178) (192). Drift-stopes are driven to limit of economic haulage levels; then pillars between levels are robbed, retreating toward shaft. Robbing m depend on character of ore, dip of bed, and the level interval. In very high lifts ore, which does not slide readily on footwall, horiz slices are taken off lower side of

ig 179). Hard-ore pillars are robbed by cutting off
l joints divide orebed into diamond shaped blocks

nton Iron Ore Mine, Birmingham, Ala; Sec in Plane of Orebed (after E. F. Burchard)

he bed worked is from 2.5 to 4 ft thick, be-
30°.

Method of mining. Entry is by single-track inclined shafts in footwall, with drifts at 100-ft intervals along dip. First mining consists of driving small stopes (rooms) up the dip at intervals of about 30 ft, with 20-ft pillars between them. An arch pillar 10 to 20 ft thick is left below levels, broken by small raises for ventilation. Most of ore is robbed from pillars on retreating. 8 shafts were operated in a distance of about 4 miles along strike; length of shafts, 400 to 1 200 ft; 370 men mined 700 tons per day; production per man-day, 1.5 to 2.2 tons; aver cost, about $1 per ton. 10 to 20% of total ore is left in pillars, or is unrecoverable because of faults or other disturbances.

Open Overhand Stopes (67, 153, 170)
rt by Pillars of Ore)

r deposits, in Keweenaw Peninsula, carry
f vesicular basalt (amygdaloid), interbedded
he whole series being tilted on dips from 30°
oint, the individual beds are very regular and
at several horizons. Distribution of copper
less than 1%. Hanging wall is diabase, usu-
rregular.

en overhand stopes are used for beds on dips
roof support. For mining steep-dipping beds
e, Art 64.

entry and development, see Art 16 to 18.

wall permits lateral development openings
e cost for maintenance. Hence, the develop-
ope are often excavated together in openings

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Details. Size of drift-stope depends on strength of hanging wall; in Osceola v of C & H Co, South Kearsarge, Osceola Consol, Ahmeek, and Centennial mines, it 9 by 19 ft; at Allouez, where hanging is s in many places, size is 9 by 18 ft; at W where hanging is strong and lode wide, it 25 ft; at North Kearsarge, 10 by 16 ft. advance is slower than in drifts, but cost pe ore broken is less owing to larger size of also stoping can begin at once. Better ve in levels results from this practice, and bette tunity to follow lode. Disadvantages: (*a*) break low-grade ore which should be left; where poor ore is encountered, the area of is reduced to about 8 by 10 ft; (*b*) stoping a stoping sometimes crowd and interfere w other, owing to slow advance of the latter.

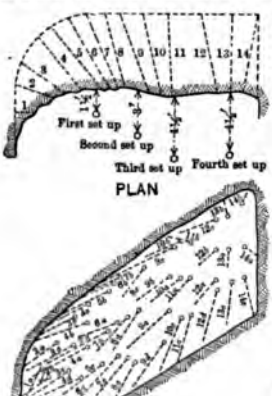

Fig 180. Drift-stope

shows a typical drift-stope in Osceola lod round resembles the side-cut for drifts (Art cut is on hanging wall, which has a well-defi and is first advanced 6.5 or 7 ft by 6 to 8 to 3 ft deep; 30 to 40 holes are required to round, depending on ground. Drills are on columns; all holes are blasted at end of sh Osceola lode, the face is squared up about per month (52 shifts); aver advance, 48 ft. sarge lode, about 8 cuts per month are blaste the ore breaks better, the aver monthly ad about 52 ft. Table 34 gives aver cost of a 12 drift-stope at Mohawk mine, in 1909 and 1910. In this period, ore broken per 2-ma drill per 2-shift day was 30.3 tons in drift-stopes and 22.1 tons in 9 by 16-ft cros

Stoping (67). Stope widths vary from 8 to 20 *ft*. GENERAL PLANS o (*a*) A stope of profitable width, usually 8 to 12 ft, is first opened along l wall; later, any ore occurring below the arbitrary footwall of first stope is broken in benches; this method is followed in Osceola workings of C & H Co; (*b*) Stope follows footwall. Ore less than 12 ft thick is broken in one operation; where thicker, a stope 10 to 12 ft high is advanced a short distance on footwall, then ore on back is blasted down. Opinions differ as to best method: under plan (*b*) it is difficult to reach and trim the hanging wall in stopes 16 to 18 ft wide; under plan (*a*) the mining of f ore starts at top of stope, for breaking ore onto the fairly smooth floor stope; this leaves less ore to be shoveled out when stope is finished tha floor is mined up the dip.

Table 34. Drift-stope, Mohawk M

	Per ton broken	P a
Labor:		
Machine men	37.4¢	
Drill boys	1.8¢	
Shoveling and tramming*	21.0¢	
Supplies†	10.9¢	
Totals	71.1¢	

* 11 tons per man-day @ $2.50 per shift
† Dynamite, 40%, costs 12¢ per lb.

Drilling was formerly done with 2-man piston-drills; 1-man Leyner drills have la placed these since 1912. J. MacNaughton (171) states that, due to reduction in l in delays in setting-up and handling, and lessened wear on steel, light drills have of drilling to ⅓ of former cost, in spite of 15% to 18% increase in wages. Sm chines drill 40 to 48 ft of hole per shift. Drilling speed varies with rock; pre coarse copper reduces speed and may cause abandonment of hole. Drilling sp 9 hr shift in 1914 (C. T. Rice); Kearsarge mine, aver 40 ft; Allouez, about 30 ft; in better drilling ground, 40 to 55 ft, aver 47 ft. Drills are mounted on column

ht columns are desirable for 1-man work. Some
s are separate, with a socket for posts of different
sual to work 2 drills close together, so that men
UTPUT PER DRILL in amygdaloid mines, in 1912,
6 days (52 drill shifts); at Mohawk mine, it was
nderground per shift of 4.15 tons and consumption
l ore broken; monthly output per drill (2-shift
de was 1 000 to 1 050 tons; at Allouez, and at
averaged 900 tons. Contract drilling is the rule
et to 4 men who work 2 on each shift with one
MENT OF HOLES. Breast holes about 8 ft deep are
aced evenly, near footwall, near middle, and near
idths of 12 ft; in places in Osceola lode, 4 holes
drilling ore is often hard to break. In Kearsarge
ft; in Osceola lode (easy drilling), burden is 24 to
influenced by the fact that in this district wet holes
rs and flats are used for layers of ore left on backs;
fficult to drill, and are more apt to break into hang-
ke; but latter requires staging for men and drill.

wall in stopes; their size and spacing depend
oping (Art 30).

40 ft along strike, less along dip; interval ranges
is of pillars is at right angles to dip. Pillars flare
er sides shaped to shed broken ore falling from
or more. Rice estimates probable loss of ore in
per is used in stopes; a stull or two may be put
for supporting men and drills. Systematic recov-
g wall is strong and does not cave promptly when
areas extends its effects to the remaining pillars

f amygdaloid lodes is great enough (generally
to fall to the level below; fine ore clings to the
tter hold up ore, demanding some shoveling.

plank platform (SOLLAR), laid alongside track in the
Since chutes are not used, fine breaking in stopes
n size; large pieces are lifted into cars by hand;
Cost of installation and maintenance of chutes,
high; due to this and the flat dip, local opinion
e below under Stoping for variations in handling
D before stope is abandoned, generally by shovel-
k, by a small power-scraper operated by a com-
ore remaining on stope floors is shoveled into steel
n the broken ore, and supported on wooden horses
ey are moved along the stope floor as it is cleaned
e 42 tons per shift.

RAISE-STOPE, say 24 ft long by full width of
r as possible from the shaft. It is driven at
machine; slow progress is due to slow drilling

NE MINE is on an amygdaloid lode with strong
idth, 14 ft; Fig 181 shows stoping method; level
rift-stopes. Ground is mined in blocks 75 ft long,
s stoped by 4 miners on contract in about 4 mo.
top of drift-stope in each 75-ft block, with an 8 to
support level above. A raise *R* is put through
ion. MOHAWK MINE. Aver width of lode, 20 ft;
Stopes are 100 ft long, separated by pillars (locally,
eft, and a pillar about 13 ft diam above dri-

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in middle of each stope (Fig 182). A flat-back stope is carried, and a cut must be out of back for each slice. Ventilation is poor in upper parts of stope. NORTH

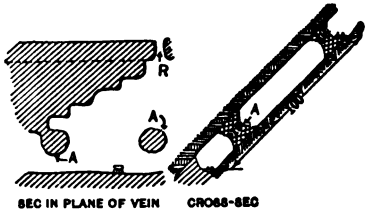

Fig 181. Stoping, Wolverine Mine

SARGE MINE. Aver width of lod dip, 38°; level interval, 125 ft. stoped in blocks 20 ft along st half the level interval on dip, o broken in slices parallel to dip; is mined in 1 month. Upper similarly mined; several lower st usually finished before the uppe gun. Pillars are usually left in stopes at top of drift-stope and a of upper stope, but their spaci strike depends on hanging wall. 10-ft chain pillar is left, except w els above are stoped out. ALLOU

Width of lode, 16 ft; dip, 38°; level interval, 125 ft. Lifts are mined in 3 series o each say 40 ft high, with pillars at bottom of each stope; poor hanging wall often pillars as close as 25 ft along strike. HANCOCK MINE. Width of lode, 8 to 10 ft; di 45°; level interval, 100 ft. Drifts, 6 by 7 ft, are enlarged by a cutting-out stope Back of level is timbered with stulls 4 to 6 ft apart, lagged with 4 to 6-in poles. A 2 by 4-ft hole is left in lagging at 25-ft intervals, with a sollar under it about 4 ft above track. Some broken ore remains in stope to protect lagging and aid in setting up drills; the rest is drawn onto sollars, sorted and loaded. Ground is good, requiring no stope timber. Pillars are left where ore is poor, also a chain pillar, 6 to 10 ft thick. QUINCY MINE. A lode about 10 ft thick, dipping 38° to 54°, is worked at great depth. Drifts 7 by 6 ft are run partly in footwall, followed by a cutting-out stope with back 18 ft above floor of level. Back of level is timbered with stulls, from 15 to 24-in up to 3 ft diam, set apart, without lagging; this is stated to be cheaper and safer than wider spaci lagging. A 5-ft space is left between stulls every 15 ft, and partially lagged, lea by 5-ft chute opening which delivers ore onto a sollar over the track. Stopes contai ous pillars at irregular intervals; stulls support slabs as required; some artificial waste on stulls are used. A stepped-face, overhand stope is carried. When stope is rock is blasted out of footwall and piled on stulls over the level to a height of 30 as the hanging wall settles, stulls are compressed until the rock filling takes the

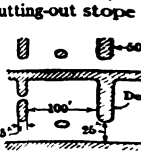

Fig 182. Stoping, Mine (sec in plane

Retreating systems are attempted in most amygdaloid mines; es as depth and consequent pressure from overlying rocks increase. drifts or drift-stopes must reach the boundary before stoping begins abov output requirements sometimes prevent this. At Allouez, it is pla extend drifts to boundary before more than the first series of stopes is at other mines, single stopes are opened successively, as far out from development allows, until the drifts reach property lines.

Besides increasing the difficulty and cost of maintaining drifts, the practice ning stopes before drifts reach boundary has following drawbacks: (*a*) blasting injures tracks on level; (*b*) mucking in stope and in drift or drift-stope inte small pieces of ore rolling from stope onto rails often derail cars; (*d*) danger fr falling from stope into drift; (*e*) sometimes a stope caves, and ore from point must then be run to level below; this is troublesome and costly, especially on fl

41. Examples of Practice, Witwatersrand, South Africa
(Open Underhand, Overhand, and Combined Stopes)

Orebodies comprise several gold-bearing silicified conglomerate beds ETS), interbedded with quartzite and slate.

Main Reef series, about 90 ft thick, is the most important; within it, 3 b been extensively mined: (*a*) Main Reef, aver width 6 to 7.5 ft, on footwall

3 in, lying 1 to 10 ft above Main Reef; (c) South
, but pinching out or becoming lean to E and W, is
Leader. Wide variations from above figures occur
dikes and faults; but their tenor, thickness, and
ndividual mines. Dip in Central Rand averages

mples of Rand Mines (194)

	M R L		S R		Stoping widths, in		
	Width, in	Oz gold per ton	Width, in	Oz gold per ton	M R	M R L	S R
	35.0	0.57	18.0	0.52	66	56	60
	13.4	1.06	21.3	0.63	71	30	51
	11.0	1.41	19.0	0.55	47	39	52
	12.0	1.27	14.0	1.22	61	46	50
	28.0	1.02	23.0	1.07	70	70	57
	23.9	0.83	21.6	0.84	63.8	63.8	63.2
	....	....	6.4	2.57	61	....	40

, = Main Reef Leader. S R = South Reef.

st row of deep-level mines; in East Rand, dips are
n West Rand, dips are 80° and over. Both hanging
rtzite, which stands unsupported over large areas
able 12).

e dictated largely by local dip; open stopes
ing wall permits stoping outward from point
y deep workings, sand filling (Art 93) is used,
bly be necessary. On flat dips, breast stopes
); on dips of 20° to 50°, mining is by under-
ombined stope (Fig 184). Steep portions of
re mined by shrinkage stopes (Art 68).

entry, see Art 17. In Central Rand, aver
ts usually follow reefs, but important haulage-
drifts (*A*, Fig 167) are often 5 or 6 ft above
to 500-ft intervals (Art 20; **Brakpan**, Art 31).

a winze is sometimes used where a dike dis-
is preferable with hand drilling, as native labor
and uppers. Underhand stopes are usual in
where steep dips permit down holes and easy

Trus-
ler these
om face

t opened
e left and
chute are
(Fig 183).
o of raise.
p of stope
ed, and a
n stope is

Fig 183. Underhand Stoping, Rand; Sec in Plane of Reef (after S. J. Truscott)

l above is supported on stulls; in soft oxidized ore
rner at toe of stope is avoided by driving a stope-
ttens, underhand stoping loses its advantage, due to
stope face.

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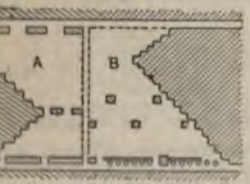
rhand stoping is common on dips of 35° and less, as it facilitates l
e and storing waste. Stopes are opened on each side of a raise,
being taken (Art 28), permitting down holes. There is no chain p
in soft ores; the level below is protected by waste-packed stulls, or
formed by driving a stope-drift, during either development or sto
tulls are used, a cutting-out stope is sometimes made over the whole le
e, timbers are placed, and chutes built before the stope is carried u

ott (30) states that, in Rand ore of aver grade, chain pillars are generally bette
ecause of their solidity in connection with the large stoped areas; they
e protection for the level and may be partly recovered. On dips of 25° or less,
l protected by leaving equal lengths of chain pillar and open spaces.
n stopes is packed on level-pillars, the chutes being maintained by rough w
ls are required for storage in upper parts of stope, as in underhand stopes.



. Combined Stope, Sec in Plane
ef; *A* = steep dip, say 55°. *B* =
lip, say 25°

Combined stopes. Upper part of
mined underhand and lower part over
(Fig 184). On steep dips, underhand
is larger than the overhand, and *vice*

Arguments in favor of combined stope
Hoover (20) and Truscott (30): (*a*) on
dips, the upper part of stope has all the a
tages of underhand stopes, with added fac
for storing waste in the overhand stope b
(*b*) with a low dip, considerable ore can be
eled from the underhand part to level a
thus reducing transport distance in stop
shaft; (*c*) pillars, if required, are easily fo
er (heel) of stope; (*d*) men may keep close to face of stope in going to and
(*e*) face of stope is carried back on a general line parallel to raises; hence, b
on stope floor in lines roughly parallel to dip; this is advantageous with me
pe transport, as swinging chutes (Art 92). (Combined stope is little used o
Rand; advantage (*e*) is the most important. Author.)

aking ground in stopes. Aver stope width is about 5 ft; mini
about 2.5 ft. Factors governing widths: (*a*) thickness of reef; as
ck as possible is mined; (*b*) dip; narrower stopes are possible on
n flat dips; (*c*) bedding planes near reef may form economical wal
to; (*d*) method of drilling; narrower stopes are possible with hand
with small drills, than with large piston-drills.

lls. In underhand stopes, holes are parallel to dip, but are flat where
is frozen to wall. In overhand stopes, hand miners point holes slightly down
hammer boys are inefficient in drilling uppers); a rill-cut (also customary wit
drills) is taken with a much flatter inclination than in Fig 180. From 2 to 5
break from foot to hanging, the footwall hole being fired first, to open the face.
round, in stopes of aver width, hand holes are about 3 ft deep; machine holes, de
per less than 5 ft wide, hand holes aver 2 ft deep. Drilling is single-hand
hoe; natives drill one 3-ft down hole per shift in aver ground, or two 2-ft ho
round; speed is less for dry holes. A 3-ft hole breaks an aver of 1 200 lb.
city of labor makes machine drills desirable in all but narrowest stopes.
drills are in general use, but constant attempts are made to replace them by li
suitable for narrow reefs. In 1909–10, competitive tests were made by ru
different size and make for 215 shifts in stopes of varying width; a 2¼-in, a 2
to 2.75-in light piston-drills survived the competition; their aver drilling speed
rock were 61.2, 58.4, 52.8, and 52.6 ft per 8 hr. Each worked 43 shifts in an
width of 30 in, 3 shifts in width of 33 to 36 in, and 1 shifts in 36 to 40 in, a
all they can work in narrow stopes. Compared with hand drilling, tonnage ho
to 5 in favor of machines; cost of explosive, 6 to 5 in favor of hand work;
machine and hand drilling were about equal, but labor cost of hand drilling
than for machines. Banket is harder to drill than aver vein quartz in U S.

ssures (70 to 75 lb) were used; due to these condi-
pe-drills could not compete with piston-drills (195).
f all kinds were in use on Rand, 58.4% large piston-
% stope-drills, and 3.1% plugger-drills (196). In
s was still done with large piston-drills, but use of
nd hammer-drills is increasing. In regular stoping,
and 6 tons of ore broken per shift; 1 white miner
36 gives data (see also Brakpan mine, Art 31).

aking Ground, Rand Mines

drilling	Ft of hole per shift	Stoping width, in	Tons broken per man or machine-shift	Explosive, per ton		Date	Bibliography
				Cost, cts	Weight, lb*		
and	3	60	0.6	....	....	1906	(30)
"	6 to 9	60	1.2 to 1.8	....	....	1906	(30)
"	....	54	.....	....	0.6	1906	(30)
"	....	48	1.0	15.8	0.56	1910	(198)
"	3.3	54	1.2	14.0	0.50	1912	(198)
"	2.75	44	0.5	30.3	1.07	1912	(198)
"	4	63	1.5	18.2	0.64	1912	(198)
"	....	65	.....	14.6	0.52	1913	(197)
"	....	43	.....	24.8	0.88	1913	(197)
in *P*	30	60	6.0	....	....	1906	(30)
chine	....	72	10.48	16.1	0.57	1910	(199)
"	....	54	.....	....	0.98	1906	(30)
in *P*	....	61	10.0	26.0	0.92	1910	(198)
in *P*	....	43	5.5	33.4	1.18	1910	(198)
in *P*	....	43	5.5	33.4	1.18	1910	(198)
ge *P*	....	70	10.0	21.0	0.74	1912	(198)
ge *P*	....	55	7.2	36.5	1.29	1912	(198)
all *P*	....	47	3.7	40.5	1.43	1912	(198)
ge *P*	....	83	19.5	20.2	0.71	1912	(198)
all *P*	....	66	.....	....	....	1912	(198)
chine	....	92	.....	14.8	0.52	1913	(197)
"	....	58	.....	32.4	1.15	1913	(197)
in *P*	24 (*d*)	57	7.0	....	....	1915	(79)
in *P*	24 (*d*)	70	11.0	....	....	1915	(79)

(*b*) Aver conditions, oxidized ore. (*c*) Central
tom to 4 6-ft holes. *P* = piston-drill. * Approx

cost in 1915 $7.33 per 50 lb for 60% gelignite
5% gelignite is also used (79); for consumption per
97) cites figures showing that in a given mine 34%
er ton in machine than in hand-driven stopes; ex-
dth of stope, and footage drilled per shift increases.

anging wall is very strong, and is supported
opes, chain pillars above and below drifts, shaft
mum width of latter is fixed by law at 10 m).

nonly 7 to 8 ft diam; in large stopes, where reef is
nly in low-grade or locally disturbed areas; under
out 40 ft apart; few pillars are left in small stopes;
are placed with respect to them. T. Johnson (187)
with dip may be allowed to pass through a pillar
tical strike-slips should have a pillar left under the
a shows two cases. Usually, to facilitate robbing,

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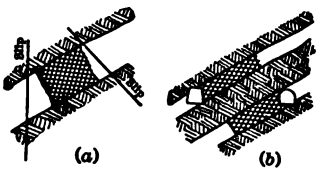
more pillars are left in lower than in upper part of stope. Machine stopes require pillars than hand-driven, due to shattering action of heavier blasts. Under aver tions, CHAIN PILLARS are left; in weak ore or under poor roof, both above and below minimum thickness, 5 ft. Work in adjacent reefs may make it desirable to chang tion of chain pillars, to insure their position (Fig 185 *b*). SHAFT PILLARS to 12 ft wide. ROBBING OF PILLARS be soon as the area containing them can be doned; owing to robbing in outcrop and increased pressures in deep-levels, movements of hanging wall have occ crushing stope and chain pillars and ir cases affecting shaft pillars (81); sand has been adopted to prevent further t (Art 93). STULLS are used locally to s slabs near working faces. Imported in 1915 cost about $83, inferior local t



(*a*) (*b*)

Fig 185. Arrangement of Pillars with Respect to Slips, and in Adjacent Reefs (after T. Johnson)

$26 per M; hence its use is minimized (79). ARTIFICIAL PILLARS, in form of pac (Fig 170), often replace chain pillars. In stopes, packs are also built of coarse Waste-packed stulls (Fig 152) are common supports, especially in steep-dipping unde stopes of outcrop mines; they also protect men in bottom of stope from falling Waste-filled CRIBS or PIGSTIES are often used, forming convenient support of roof a pillars while being robbed (Art 31, 36). For levels some engineers prefer them to pack

Handling ore in stopes. Where dip allows it, ore slides on footwall t level; on the usual dips of 25° to 30°, it is handled by mechanical de to avoid shoveling (see Art 92; also Brakpan mine, Art 31).

42. Pillar and Chamber Workings

The term denotes methods of mining in which large chambers are excav leaving pillars of ore for permanent support. Chambers may be mine underhand stopes, glory holes or overhand stopes (see also Mother Lode below). Support for men in overhand stopes is afforded by accumulati broken ore (Shrinkage stopes, Art 68), or by square-sets (Art 45 *et seq*); chan are rarely filled with waste to support men (see Rimogne quarry, below).

The method is a development of room and pillar work in beds (Art 32), for masses or wide veins. In typical cases, workings are a series of large open stopes arated by a network of vertical and horiz pillars of ore. Synonymous term CHAMBER WORKINGS, PILLAR WORKINGS, ROOM AND PILLAR, PILLAR AND STALL, RO (see index for other use of same terms; also Art 104–6).

Chamber workings are a very old form of mining, obsolete for metalliferous de except in rare cases; they are useful for exploiting cheap and abundant minerals, a or slate, where much of the deposit (40% to 70%) can be sacrificed in pillars for sec cheap support and low mining cost. Recovery of pillars is generally costly, ofte possible, and workings are left in such shape that a change to other methods is dif

Requirements of method: (*a*) large deposits, as masses, wide veir thick beds; (*b*) deposits of strong mineral or rock, with strong walls, will stand unsupported over back and on walls of chambers; (*c*) cheap erals, which will not pay cost of installing other methods; (*d*) deposits of form value, as no sorting is done in chambers.

Examples of practice which follow show variations and applications of the meth

Tilly Foster mine, N Y. Orebody is a steep, lenticular mass of strong magn embedded in gneiss; in horiz sec, it is about 200 ft wide by 300 ft long; dip, about It has been mined to a depth of 600 ft. Fig 186 shows method of mining part o deposit. An inclined shaft was sunk in footwall, with wide drifts at 100-ft inte along footwall to ends of orebody. From drifts, rooms 24 ft wide, with 20-ft pillars, carried across to the hanging wall. They were excavated overhand as shrinkage s (Art 68). A timbered crosscut in middle of each room connected with footwall

re raised to within 15 or 20 ft of level above, leaving r whole area of orebody. When a room was finished, oom ore- fill pil- part l by

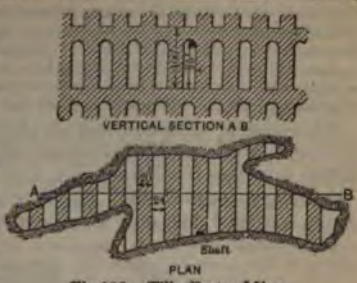


Fig 186. Tilly Foster Mine

nge, for tite ex- ical 70- ven cted zes, wall to wall, leaving 30-ft pillars. Top of chamber , ore being mined underhand, starting at top of a winze. At places, the glory-hole method (Art 35) was used. Where a level-pillar was left, the heading method (Art 34) was employed (Fig 187, vert sec *CD*), the chambers being 50 to 150 ft high. At other places, 2 series of 30-ft rooms were driven at right angles to each other, leaving 30-ft square pillars. About half of ody was left in pillars; some of this ore has since recovered after filing old stopes with waste. For ore, shrinkage stopes are now used, with subsequent g (Art 69).

io Tinto mine, Spain. Orebodies are lenses of pyrite, enough chalcopyrite to carry an average of about Cu. Length of deposits, 1 200 to 6 500 ft; width, rally proportional to length, reaching a max of 250 ft; ually occur along contacts between porphyries and weakness in either rock. Nearly all have at some chamber (locally, wing details (203). ternate slices being ngles to each other; as left a solid floor ing on strength of by 13 ft; 16.5 by o tried, but proved aller openings were er on entering good . Face of galleries ng, leaving a bench thod, Art 34). Fig one deposit; entry levels 41 ft apart, ate levels are used rosscuts, 10 to 11.5 s on each level, at square, the pillars

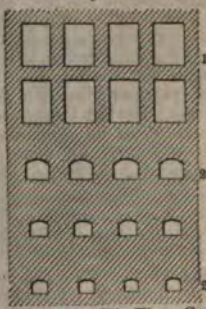


Fig 188. Rio Tinto Copper Mine, Spain (vert sec)

Chambers are then made by heightening and widen- ore, pillars are about 10 ft square and chambers 30 to

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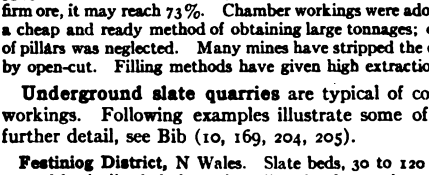
33 ft high; under aver conditions, pillars are 15 to 20 ft square, chambers 20 to 25
Vert sec (Fig 188) would be the same, whether taken parallel or at right angles to
it shows successive stages of work in different levels. At many mines in the distri
33% of the total ore is won by this method; in case shown, extraction averages
firm ore, it may reach 73%. Chamber workings were adopted largely because they
a cheap and ready method of obtaining large tonnages; ore was so plentiful that
of pillars was neglected. Many mines have stripped the overburden and mined ol
by open-cut. Filling methods have given high extraction in much recent work.

Underground slate quarries are typical of correct application of ch
workings. Following examples illustrate some of the methods employe
further detail, see Bib (10, 169, 204, 205).

Festiniog District, N Wales. Slate beds, 30 to 120 ft thick, dipping 20° to
opened by inclined shafts or by adits; levels are about 50 ft apart. Transverse
bers are opened 30 to 50
along strike (Fig 189), w
lars of about same size.
hand drilling, sides of pill
low planes of weakness
as "pillaring") at right
to cleavage; these are
askew to dip and deviate
from perpendicular. W
channelers are used, pill



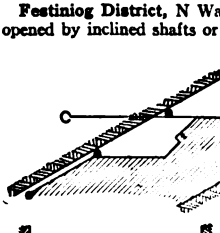


Fig 189. Underground Slate Quarry, Wales

vertical and normal to dip. A chamber is started
by 4 or 4 by 6-ft raise (called a "roof") along hangin
at center or corner of proposed chamber; the raise i
widened to full width of chamber, widening and
usually proceeding simultaneously, as small raises ar
to ventilate. Floor of widened raise is carried back t
hanging wall, by cutting off slices parallel to cleava
bottom cut is first made across width of chambe
bar channeler drilling close-set 2-in holes; ribs b
holes are broken with a broad chisel bit. Other c
made higher up and also along chamber walls, and
blasted down in sections with small charges. Cha
and pillars on successive levels are under and connec
each other. Vertical distance between floor of ch
and hanging wall may reach 120 ft. Main drifts are destroyed by work in the lift
communication being maintained by run-around drifts in hanging, or by bridges;
mediate drifts are also run. Slate from development is all lost, and about half
total is left in pillars. Further loss in splitting, etc, reduces net yield to about 13%
deposit. L. Mayer states that in 1908 the Oakley quarry, with full crew of 1 50
produced about 6 000 tons of slate per mo (169). One chamber may provide worl
small crew for 10 or 15 years. Elsewhere in the district, conditions allow chamber
100 to 150 ft and pillars 24 to 50 ft wide. No means have been devised for mining
Men work under high unsupported roofs, but, though pillars often collapse, fatal ac
are rare. Backs are examined and trimmed from ladders erected in 30-ft section
held by guy ropes; 3 to 6 men take 1 week to build up a 100-ft ladder.

Rimogne quarry, French Ardennes. Principal slate bed dips 22° to 45°; thi
0 to 200 ft. Long chambers (30 to 50 ft wide) are excavated along the strike, i
of across it, as at Festiniog. They reach from foot to hanging, with pillars abou
wide along dip, sides of pillars being normal to dip. Chambers are mined practic
overhand filled stopes (Art 60); much of the waste is used as filling (205).

Anjou, France. Slate beds 130 to 500 ft thick stand almost vertical. Outcr
generally covered by a considerable thickness of worthless slate. They were fo
worked in large isolated chambers, each with its own entry from surface. A ver
was sunk to good slate, and an arched room excavated over top of proposed ch
by breast stoping. Floor was mined underhand in 10 to 13-ft benches. Slate
were handled in a stone skip attached to a trolley on a cable brought down the sha
fastened to an eye-bolt in floor of chamber. Horiz sec of chamber was a circ

500 to 27 000 sq ft; some chambers were 360 ft deep, ining rock salt at Marmoras, Hungary (205).

posit is a dome-shaped mass, in places over 2 000 il, and has been worked on 2 levels at 90 and 160 s methods in 1896 as follows: Chambers are 80 ft e reduced by crosscuts to 60 by 60 ft. Chambers ercut"), 7 ft high by 200 ft long, the roof of which is alt mucked out. Successive slices are then taken off or. Drillers work from scaffolds on ladders, which, ned and finally discarded as chamber is heightened. middle and 50 ft at walls, the broken salt fills the ariant of shrinkage stoping (Art 68). 2 men bore s; 0.75 lb of 18% dynamite is used per ton.

B C. Orebody is an irregular lenticular contact gnetite, in a gangue of actinolite, garnet, epidote, limestone, is well marked; hanging wall is a com- e minerals or of altered granodiorite. Length of diminishing with depth (on 400-ft level, it is about 40 ft. In 1913, a depth of 650 ft had been reached. n lower half of deposit. Ore mined averages 18.7 Little or no timbering is needed, except for chutes. levels are 60-ft level his, rooms were form- e that of Present d by him bers (Fig ces, those work re- t for per- each level an inter- (Fig 191).

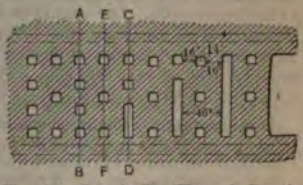


Fig 190. Longit Sec Through Developed Orebody, Showing Pillars and Chambers

s, footwall raises and inclined chute-raises are made, o hanging or level above (Fig 191, 193). Slope of lides with little handling and men can work without mined by breaking down the ribs between inclined n are close to the back. Down holes are drilled by

92. Pillar Developed ady for Drilling and sting (Sec *EF*)

Fig 193. Chamber Partly Mined (Sec *CD*)

being anchored with ropes. Broken ore falls into ect haulage drifts, their shape aiding in drawing ore 92 shows preliminary work. Fewer raises are run ft deep) are drilled in top, bottom and sides of in- lar and all fired together. Drilling in both pillars drills and plugger-drills; former make about 32 ft

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per shift. In 1911, for blasting out 1 pillar, 2 433 holes were charged with 40% dynamite and fired simultaneously, breaking 175 000 tons.

Cost per ton, for mining 377 510 tons in year ended Nov 30, 1912, was: mi timbering (labor), 13.8¢; general underground and tramming labor, 9.56¢; e 9.97¢; candles, 0.46¢; drill parts and hand tools, 2.1¢; shops (except drill parts tools), 0.63¢; compressor and power, 2.69¢; lumber, 0.28¢; sundries, 1.58 41.07¢. Wages, $3 to $4 per 8 hr. Breaking ground is thus very cheap; cost sives is increased by necessity for blockholing large pieces in chutes.

For successful APPLICATION of this method, the orebody must be large, fairly of too low grade to permit use of filling or timber, and with footwall steeper t ADVANTAGES: (*a*) little mucking is necessary; (*b*) no timber except for chutes; nomical use of explosives (Table 29, Ex 6); (*d*) drills have large capacity in pil as they drill continuously without blasting delays on each shift; (*e*) method is

42. Sub-level Stoping

This method, also known as sub-stoping, was developed in Michig mines about 1902. It is an open-stope method and distinct from s caving (Art 76, 77). Following data and drawings are from F. W. Sper McDonald, and F. C. Roberts (208).

Type of deposit mined by sub-level stoping in Mich includes "p and narrow ends of large lenses; in general, steep-dipping, tabular d 12 to 100 ft wide. Ore should be free from slips, and strong enough fo to stand after trimming and benches to stand under their own weigh hard ore increases cost of driving sub-levels and is a disadvantage). must be firm, or they will cave and mix with ore, and overlying capping be strong, so as to permit stoping of large panels or blocks.

Development. Shafts or other entries should be outside of the oreb mining eventually causes walls to cave. Level interval, 100 to 150 f cally, each level a main haula from which cr are run to the w outline the depo

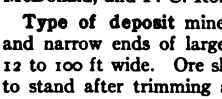

Fig 194

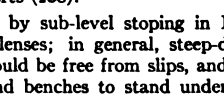

Fig 195

Chute-raises are from drift and cr to about 25 ft abo floor. Drift and cr are planned so th ters of raises are 15 to 30 ft apart; in 25 to 40-ft orebodies, raises alternate on o sides of haulage drift; some wider bodies have 2 parallel haulage drifts, connected a vals by crosscuts, to facilitate motor haulage. At end of deposit (or at end of panel to be stoped) a raise is put through to level above. Fig 194 shows procedure when end of orebody slopes forward. The triangular block *ABC* is broken by underhand stoping and handled to the level through the end raise; and ladderway and sub-levels 1, 2, and 3 are driven. If end of deposit slopes backward or is vertical, sub-levels are started from end raise as well as from ladderways (Fig 195). sub-level (No 1) is at tops of chute-raises; vertical interval between sub-levels, 25 ft; they are closer in soft than in hard ore.

Fig 196

nsists of a single drift in middle of the deposit. sub-level No 1 (Fig 195) is first widened to bout 12 ft wide and of same height as sub- deposit, and worked back toward ladderway.

f chute-raises, and uppers in back of breast stope, unnel shape to tops of raises (Fig 196). One round nigh across width of deposit and about 12 ft along raises and to chute gates in main level. After

97. Plan of Sub-level Stope in Wide Orebody

ormly, being kept 20 to 30 ft apart. Each main the series. When the ladderway is reached a new ve been opened to the rear. Thus, men on each nough back from open stope to be safe from falling o orebodies 50 to 100 ft wide. A stope about ½ through middle of deposit, leaving a pillar on each 60 to 100 ft, crosscuts *K* and drifts *M* (Fig 197) e extended and form benches along face of pillars 198) and benches *B* (Fig 197) are carried across of pillars to walls. Pillars are then mined by holes d from benches *B*, as in the central stope; each h is kept a safe distance in advance of the one e. A vert sec through *DJ* or *D′J″* (Fig 197) would ke Fig 196; vert sec through *D′J′* would show entral stope worked back to a vert face.

long orebodies, it is usual in Michigan to make s about 250 ft long, separated by pillars contain- the ladderways. Length of stope is adjusted to ent premature caving of walls; horiz pillars can be left underneath main levels. Ore in pillars be mined later by top-slicing (Art 71), or by sub- caving (Art 76, 77), provided stopes are first . Filling for stopes between 2 levels may be n down and reused in lift below, allowing walls e to cave. (See Miami method, Art 87.)

afety and small timber consumption in suit- ous points of attack, a large tonnage can be Small charges break large tonnages from breast stopes is more costly);* aver powder DISADVANTAGE is the large amount of narrow ost of which increases with hardness of ore. rrow orebodies than in wide.

rinkage methods (Art 68–70). In sub-level stop- round, but where the back will stand unsupported atter would generally be preferable. (*b*) Square- oice exists, it would be based on relative cost of mber for square-sets.

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Field of use seems to be in orebodies with good walls, which are to or otherwise unsuited to caving (Art 71 *et seq*), and in which the ore is to to use shrinkage methods and yet not soft enough to require square-sets

44. Summary of Open-stope Methods

Applications. These methods, employing pillars of ore for per support of walls, are best suited to low-grade orebodies; they sacrifi of the deposit to secure low mining cost. Caving methods yield hig traction than open-stope methods, and compete in cheapness with them fo low-grade deposits. (For limitations of caving, and types of orebody to it is suited, see Art 83.) Open stopes are in general limited to orebodi strong walls; in overhand methods, the ore must be strong enough to unsupported over backs of stope, as is true also of most underhand w wide orebodies (Art 34, 35). Strong ore is also required where ore pill relied on to support walls. Field of open-stope methods employing ore for permanent support is in beds and veins dipping less than 35° to 45° massive deposits where vertical pillars can be left. Stulls for supportin slabs, etc, are usually limited to a max length of 20 ft (Art 36 to 38); se vidual methods, Art 29 to 43, for their applicability and limitations.

Underhand methods. ADVANTAGES: (*a*) all drill holes are down holes, drill this avoids dust, is advantageous for hand-drilling with unskilled labor, and use of efficient plugger-drills. In general, breaking ore costs less in underhand overhand stopes; (*b*) miners stand on ore while working, which is safer than ov stoping in a few cases in narrow veins of weak ore with strong walls; (*c*) und methods, where applicable, require less timber than overhand; (*d*) in general, fines is less in underhand than in overhand work. DISADVANTAGES: (*a*) danger, men work under inaccessible backs and walls, the height and area of which incr stope is extended, and loose pieces may fall; (*b*) due to above conditions, the leve val must usually be smaller than for overhand methods; (*c*) facilities for storin in stopes are poor; (*d*) broken ore from face collects at a single point, which ma fere with economical handling and loading.

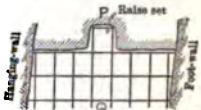
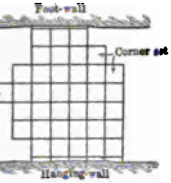
Overhand methods. ADVANTAGES (not indicated above): (*a*) miners wor back of stope, where they can examine face carefully and take down loose groun are not exposed to danger from pieces falling long distances; (*b*) waste or ore is stored in stopes; (*c*) a greater variety of working plans is available than in ov stopes; any type of machine drill may be used, and by using a rill-cut (Art 36) holes may be drilled; (*d*) an open, overhand method may, if required, be chang square-setting, shrinkage, or filling method, far more readily than underhand s (*e*) gravity aids in breaking ground; (*f*) broken ore falls down the dip under the i of blasting; this allows handling of ore to the level below without mechanical on flatter dips than is possible in underhand stopes. DISADVANTAGES are largely t verse of advantages of underhand; in addition, support for men is necessary in ov stopes on dips over 40° to 45°, whether required by walls or not (Art 94).

TIMBERED STOPES

The term "timbered stope" is used here to denote stopes in which tim is the predominant feature of the method. Stulled stopes (Art 36 to 3 also types of timbered stope.

45. Square-set Method

Square sets were first used in the U S in 1860 by Philip Deidesheim Ophir mine, Comstock Lode, Nev. The method is often called the N square set system. It is used chiefly in overhand work; for its use in hand stopes, see Art 46.

dvanced by successive small excavations, each Timbers in adjacent sections are framed into s forms a continuous structure of horiz FLOORS, ted at their corners by POSTS (Fig 199). One hose at right angles to caps being GIRTS (ties, ft high, are 5 to 6 ft apart c to c, both capwise



Diagr of Sq-set System

and girtwise. Caps are often stronger than girts, the principal function of girts being to hold the posts in position. In veins, caps usually have their long dimension at right angles to the strike; in masses, in the direction of max lateral pressure.

as well as the stope walls. Lowest floor of a highest working floor is the MINING FLOOR and by laying planks across caps or girts (Fig 200). OR, etc (Fig 201). The first set erected on any and 2 girts. First set in a new row on a floor d 1 cap, or 2 posts, 1 girt and 2 caps. In work- CORNER SETS are used, consisting of post, cap nearly all sets added are corner sets (Fig 200). ich the sets alone are relied upon to support the te filling is used for added support.

llows general overhand practice (Art 26 to 28).

Square-set Stopes

are based upon the use of square-sets, ary overhand stopes. DEVELOPMENT RE- A drift, either in wall or in orebody, is a In wide orebodies, there may be a grillage e raise to the level above is necessary in ing of timbers and filling.

hand stope is illustrated by practice at cDonald gives following data for 1903 (209).

fissure veins in silicified monzonite; max vein country rock are generally hard and stand well, uare-sets are used for widths over 12 to 15 ft. ment drifts, 6 by 8 ft, usually untimbered, are to the level above. Sill floor is breasted out of 12 or 13 ft. This gives a space of 2 or 3 ft shion of old timber on which ore from first floor e floor, the first being near footwall and parallel and sills covered by planking. Work on 1st or 2 sets wide and 1 set high across vein, and them. Fig 201 shows a stope of several floors;

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the face on any floor is advanced by a cut 1 set wide across the vein. Brea parallel to strike and about 7 ft deep, are drilled with large piston-drills. Br falls on the floor, and is shoveled to chutes in alternate sets or in every third set. move across face from wall to wall, followed by muckers and timbermen; or

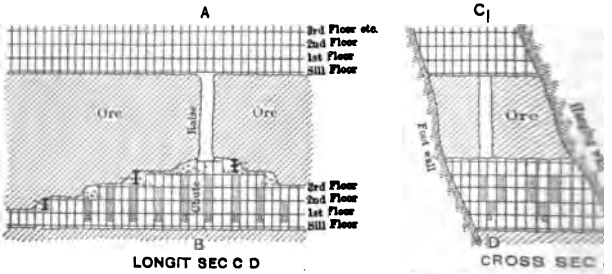

Fig 201. Square-set Stopes, Le Roi Mine, B C

may alternate with muckers at opposite faces. Faces of successive floors are ke 60 ft apart, to allow simultaneous work on each without interference (this di readily reduced to 20 ft at various mines in U S). Waste from development is into the finished stopes, but systematic filling is not attempted.

Details of open square-set work in stepped-face, overhand stopes ve stope-drills are used, ground is broken by uppers drilled from shovelin Main function of timbers is to support slabs, broken ore, and men. I ground, waste filling is generally required to prevent failure of timbers; of work is then modified as follows:

Square-sets are used for development openings, which can be laid out in straig corresponding to lines of stope timbers; crooked drifts are usually timbered wi sets, replaced with square-sets as sill-floor work proceeds. The floors are wid sections small enough for ore to stand unsupported until sets can be erected. The kept filled with waste, usually to within 2 or 3 sets of the back; under heavy pr only top floor sets may remain open. Haulageways are kept open on sill floor by the sets through which they run. Art 47 gives details of handling ore, waste, filled stopes. Occasionally, in very soft ground, the stope faces are advanced by

Domed or pyramid stopes. The work is so arranged that the general of the stope back is dome-shaped or pyramidal. They are commonly stopes, and have been used in massive orebodies of both strong and we under strong hanging walls. The arched back is partly self-supporti reduces pressure on timbers.

Fig 202 shows this method at Centennial-Eureka mine, Tintic, Utah, for mi irregular chimney, 350 by 100 ft in cross-sec, dipping 45°, and over 800 ft deep.

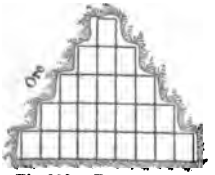

Fig 202. Domed Stope, Cross-sec

are fairly strong limestone, but in many places ore mined with a pick. Level interval is 100 ft vertical raises between levels near one end of orebody. Set 8 by 8-in timber, with sills 2 sets long laid parallel dimension of orebody. Work on sill floor starts at t and when a square, 3 sets on a side, is completed, a set is placed in the raise. Sill and 1st floor are then simultaneously; when the 1st floor is 3 sets square, floor is begun, and so on. Crib bulkheads are nece keep stopes from caving (Art 52) (210).

L. S. Cates describes similar work at Bingham, U irregular replacement orebodies in limestone or bet quartzite foot and limestone hanging wall (211). Ore is in shoots, 50 to 200 ft wi to 300 ft long, and in places 600 ft deep; dip, 0° to 90°. Open pyramid stopes a

vall firm. Space for only one set is excavated in set long. From a drift near center of orebody, ngitudinally through it. The sill floor is opened then a row of lead sets is begun on the 1st floor, ning on any set. Each wider than broken ore

ngitudinal t Old Do-

set ahead of the filling, *b*, made by bing is built ccumulates better foot- nced 1 set, e raises are ise *d*. At opes up to ore of uni- ossesses the g handling

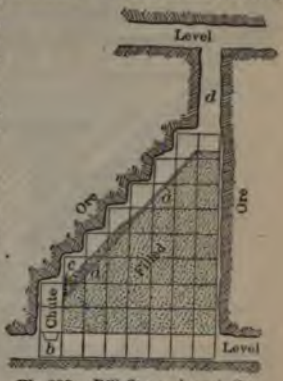

Fig 203. Rill Stope, Longit Sec

hoveling of both is necessary. The number of simultaneous work on adjacent floors interferes. he Broken Hill Proprietary mine, N S W (213).

shows practice in open stopes at Bingham, e Domed stopes. Stope faces are vertical e nature and extent of ore are not known.

se to hanging wall, followed at a convenient dis- ete, stoping begins at *B*, and sets *C* are carried g wall. Successive vertical slices are taken until worked out, or, if the weight becomes great, the ed and a new one started alongside. Sills and ight angles to strike. Advantages (L. S. Cates): always a solid breast of ore on one side of stope, y relieves pressure on timbers. (*b*) Should the nexpectedly, only the ore on floors and in chutes new stope can be opened by driving a row of ill floor, next to the caved stope, and then carry- to the hanging as before."

od in filled stopes is illustrated at Bunker Hill & e, Idaho. Large irregular shoots of heavy galena le waste, occur in shattered quartzite; dip, about , 300 to 700 ft long and 40 ft wide, have been car- lifts of 200 ft. Hanging wall and ore are heavy, k or stepped-face stopes expose areas of back too ty (214). Work begins on a level, Fig 205 (215); ce, 3 or 4 sets wide, is carried up under hanging ed toward footwall by mining vertical slices 1 or e dropped through footwall raise *C*, and trammed le. Waste is distributed in stope by temporary driven directly over *A* to serve the next floor; each being abandoned as a new floor is opened. rower stope, worked up to 6th floor, showing the floors, and arrangement of tracks for handling fill- hrough a raise along hanging wall; but a footwall

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raise is preferable in heavy ground. Rock chutes to footwall drift (Fig 205) are apart, cribbed or lagged chutes being carried up from them through the filli

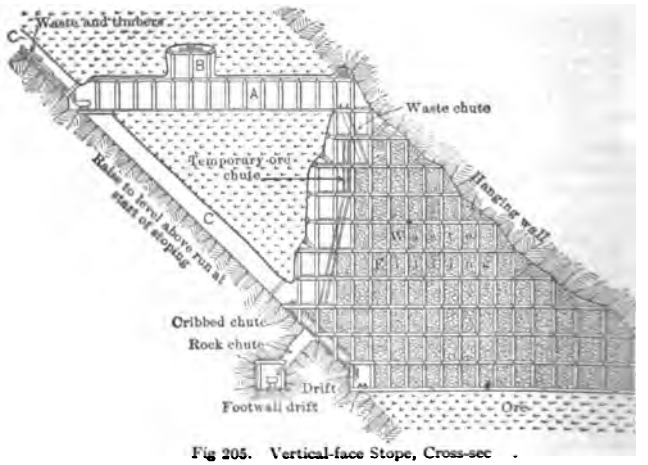


Fig 205. Vertical-face Stope, Cross-sec .

stope advances. This method is safe and provides prompt support for the wall. Most of the weight of ore is carried by footwall.

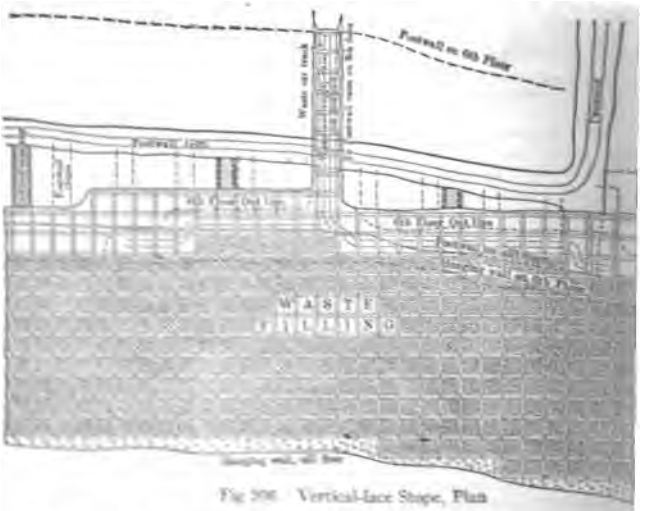


Fig 206. Vertical-face Stope, Plan

Block system is used for wide, weak orebodies, with weak walls. blocks are stoped and filled; then corresponding blocks are mined al Size of block should be adjusted to strength of the ground, so that wor

ve pressures; individual blocks are usually . Timbers on any floor are eased before ing the floor above. This permits use of of reinforcement needed.

. Massive orebodies. Blocks (Fig 207) are d ling to strength of ground. To reduce develop-, around e it im- blocks. (216). , miner- e (along Each is l stopes ions are , so that gh arch ack. A djacent se raises val, usu- e 100 ft

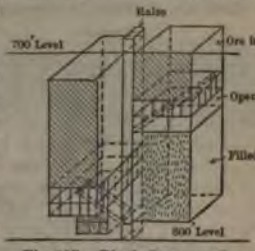
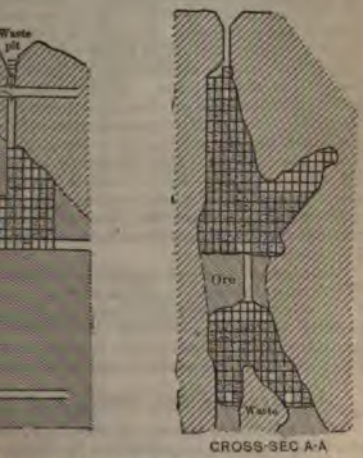


Fig 207. Block System, Copper Queen Mine

h special timbering for leveling stope floors (80). 208 (P. B. Scotland) shows block system applied o ft wide (178). CENTRAL MINE, Broken Hill, g from hanging to foot in a vein several hun- d as open square-set stopes. Attempts to mine



stem, Arizona Copper Co

. Later, blocks of same size were mined success- nt. blocks are mined and filled without square- ev. Attempts to mine in large filled square-set n caves and led to use of block system. Blocks and extend from wall to wall. Ore is in shoots . Walls and ore are generally weak. Work on

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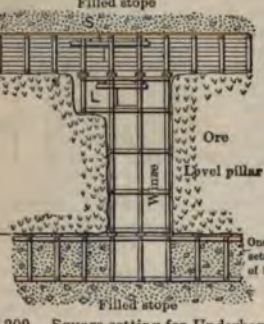
r in a block begins by crosscutting to hanging wall, along which a slice
mined. Hanging-wall sets are lagged, filled with waste, and remainder
oward footwall. Two floors are kept open, but the shoveling-floor sets,
side of stope, are filled as face of mining floor advances toward footwa
width of back supported on timber alone, and gives prompt support to
s between hanging wall and ore. Chutes are 25 ft apart, carried up
ver slope of 50°. Ore is sticky and clogs in long vertical chutes. Becau
econd line of chutes is necessary to serve upper parts of stope (145).



209. Square-setting for Underhand Stope

Underhand square-s
are unusual. They are
at Broken Hill South min
for mining chain pillars, 3
under large filled stopes.
is done in vertical slices,
from wall to wall; whe
adjacent slices are compl
are filled before more
opened. Work starts fr
a winze, by an excavatio
set *L* (Fig 209). This un
sets in a crosscut above,
supported by a boom *S*.
erected and caught un
boom from the winze
210). The 6 by 10-in c
A supports the 2 caps an
supported by boom *B*, wh
s rear end being wedged under *D*. Dogs *K* tie posts *F* to caps and girts
blocked, the ground beneath is excavated, another set is suspended by
repeated until the filled stope below is reached. The bottom set is then blo
may be removed. By repeating these operations
ft wide (1 set) and 30 ft deep is carried across the
lar. Broken ore falls to and is handled on bottom
the underhand stope. Mining in the slice along-
rts from a lead set (Art 45), at one end of the top
t. Work under old stope bottoms is difficult and
es requires spiling. It is found that timbers
lices over 60 ft long; for greater widths of lode
in pillars are sliced in 2 sections. Object of this
is to expose only small sections of old stope floors
ime. It is stated that, though slices are narrow,
depth gives capacity and considerable economy.
ails, see Bib (217).

Fig 210. Details of Und
Square-setting

derhand square-set method is used at the Colonel
nine, Leadville, in a large orebody of shattered
ulphides, under a heavy crumbly hanging wall;
l data could be obtained. P. B. Scotland states
s method has been used at Morenci, Ariz, to extract a set or two of ore und
t has not been used on a large scale and is considered uneconomical and da

. Handling Ore, Waste, and Timber in Square-set Stop

s handled in barrows, on slides, or by shoveling to chutes or pockets buil
lat holes may be drilled from the mining floor and ore sorted and handle
g floor, without interference between miners and shovelers. Vertical
drilled from the shoveling floors. Close filling may require both drill
g to be done on mining floor.

rs are of 2 to 5-in plank, long enough to cover 1 or 2 sets; the
is more convenient; flooring is taken up and reused; 2-in plank
o stand heavy blasting, and should be doubled, or protected by
, or 3 or 4-in plank used.

flooring lasts for a 200-ft lift (80). Floor plank ith the grain; for barrow work, planks usually lie re essential to prevent loss of fines, which may ant with high-grade ore. Canvas may be used t Goldfield, Nev, in high-grade ore, the shoveling n the floor is finished the planks are taken up and f down to the caps and sent to the chutes; this gh joints between planks (145).

life of stope. Types: (*a*) A cheap chute the outside of a row of sets, but lagging is winging of sets or by pressure of the filling, oying the chute; (*b*) The inside of a row ined with vertical planks. This is common ckness of lining (2 to 4 in) is proportioned able 2 or 3-in lining may be used to secure square timber, built independently or just not be lagged outside. Independent chutes s at an angle to the sets. Chutes inside of strong resistance to wear and pressure of "bricked" with timber blocks.

ows of 10 by 10-in blocks, 14 in long, laid close; the chute, and is wedged tightly between the es the chute sets; it is used for the lower 100 kept open through old stopes for passing waste. e of lining several times (80).

by offsetting to reduce vertical drop. At Center fset every third floor. Wear is also lessened by ates, see Art 91.

et 8 to 10 in apart, are often placed over tops of night block chute or gate. They are especially s in filled stopes.

s from 15 to 50 ft or more; it is closely dling. Cheapest handling is furnished by upper parts of stopes ore can be thus de- wer parts of stopes started with this chute

ack of stope, the smaller the area served by a slide. es, 50 by 25 ft in horiz section, filling is kept 2 sets

Fig 211. Wing-chutes, or Slides

floors of 3 to 4-in plank, spiked to the sets. ng floor and thence is dropped or shoveled to the w the back. If sorting is unnecessary, both slides

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and filling may be moved 1 set higher. Slides may converge in 4 directions tow and many variations are possible. Their use does not entirely eliminate sho

Pockets or bins for storage are sometimes used in open square-set stopes. Th by putting lining around from 1 to 5 sets, terminating at the bottom in a may extend through the whole height of stope. Pockets reduce shoveling di may obviate necessity for wing-chutes.

Access to open square-set stopes is by ladders from level below. In f there may be a raise to level above, or chutes built to include 2 rows of sets, compartments being a MANWAY. In heavy ground, these are harder to keep single compartment or "blind" chutes, but the manway is not only a tra through filled sets, but facilitates repairs to the ore chute and clearing it w

Filling is obtained in part from sorting, or from breaking was grade inclusions in the orebody. For running in filling from outsi a raise to the level above is necessary. Filling is distributed by shovel or cars, depending on size of stope and distance between waste raises.

Practice in handling filling (see also Rill stope, Art 46). At METCAL friable, massive orebodies, stopes are kept full of waste to within 1 set of the ba (from surface workings) is distributed from a small raise in a corner of the st venient arrangement, because, by beginning mining of each floor at this point can be kept close to the face without interfering with shovelers (152). At H MINE, Leadville (statement by G. O. Argall), orebodies are massive repla limestone and porphyry, with flat dips. For opening stopes, main haulag driven under the orebodies, with raises 50 ft apart. Waste comes from d work, where a raise exists for dumping it into the stope; if there is no rai obtained by driving inclined raises into back of stope, or by drifting into wa ARD MINE, Butte. Wide vein; soft ore; block system of stoping; stopes, (Art 46). A 2-compartment raise is carried to level above in middle of each A car on a temporary track, laid lengthwise on shoveling floor, in middle of tributes filling, which is kept within 2 sets of back. Side walls of stope ar prevent runs of filling when adjacent blocks are mined. Posts of the outer se top and bottom, to those of next row by spiking on 2-in planks. Lagging, 2-in plank, is placed on stope side of posts. Planks are spaced 5 to 6 in apart cheaper and have longer life, but take twice as long to place. Open lagging i to allow fine filling to run through and fill space between lagging and walls. space prevents sets from swinging, and reduces loss of ore on sides of stopes filling when the block alongside is mined. One man places lagging and distrib in a 50 by 25-ft stope.

Timbers are handled most cheaply when lowered or dropped t raise from the level above; 1 compartment of a waste raise may be u Lacking access to stope from above, timbers are hoisted from below b lass, or electric or air hoist. In filled stopes, TIMBER CHUTES are s kept open through the filling by lagging and lining a vertical row of se a manway or ore-chute; or a timber slide may be built in a corner of

48. Dimensions of Square-sets

Horizontal dimensions. Table 39 shows that practice favors s 5 by 5 ft in plan, with floors 7 to 8 ft apart.

C. T. Rice states (80): (*a*) Posts should be far enough apart to give shovel to allow for right and left-hand shovelers, they should be equidistant capwise wise. A clear space of 4.5 ft is about minimum for efficient shoveling; hence, timbers, posts should be at least 5 ft 4 in centers. (*b*) Strength of ore limit of set, as the ground must stand unsupported over area of 1 set while timbers placed. Heavy pressure from weak ore may require smaller sets (see *e*). should be proportioned so that 1 round of holes breaks room for a new set, thu prompt placing and blocking of sets. (*d*) Large sets are desirable, in reducin of timber per ton of ore, but, in large sets the timbers are heavier and cost more and a stope started with a large post interval may become unmanageable if t

between posts and the area of stope floor are
dly in small sections, the pressure on timbers is
nd smaller timbers. (*f*) A compromise between
ng of posts to 6.5 ft and minimum spacing to 5 ft.

sts could often be increased without over-
es, where the weight comes onto timbers
ts through lagging and caps, a comparison
f different size can be made.

s method used by him at mines of the Ariz
s in Fig 228. Round timber cost $19.09, square
eas of sets = ratio between total pressures they
taken as ultimate strength of cap. Scotland
aried to suit weight of ground.

e-sets, Ariz Copper Co, Morenci, Ariz

ft 8 in by ft 8 in by 7 ft 2 in	6 by 6 ft by 7 ft 2 in		6.5 by 6.5 ft by 7 ft 2 in	
8×8	8×10	10×10	8×10	10×10
6×8	6×8	6×8	6×8	6×8
12	12	12	12	12
1.28	1.44	1.44	1.69	1.69
0.88	1.26	1.63	1.18	1.51
117	128	138	133	144
230	258	258	303	303
13.4	20.6	20.6	24.1	24.1
6.3	6.2	6.7	5.5	5.9
13.4	13.0	14.4	12.0	12.9

room is about 6.5 ft in the clear; in many
are cut to give a clear 7 ft. Length and
d to withstand pressures.

e due to rapid work in small stopes allowed use
12 by 12-in. and also an increase in height from
fects no saving in timber, unless the increase is
floors in a lift. Long posts probably increase
enough to counterbalance the saving in timber.
ed posts from 6.5 to 9.5 ft; this increased the
of ore, and reduced total timbering cost 15%.
ch greater than transverse strength of cap and
es not materially reduce strength of set. It is
ing when pressure comes from the side (178).

ng Square-sets

s. Practice is about equally divided be-
be based on theoretical grounds alone,
e-set stopes generally exceed safe bearing
ructures and hence there is considerable
of pressure is rarely parallel to caps or
ever known; (*c*) distribution of stress in
centric loading can not be calculated, be-
e members do not fit exactly.

across the grain is from 10 to 20% of its com-
In framing post-butting sets, the area of the
f the joint; in cap-butting sets, the area of cap
int. Hence, under pressures within the limit of

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safe bearing values, square-set joints are always strongest in the direction in tenons butt. For such conditions of loading, post-butting sets should be used direction of max pressure is vertical; cap-butting sets, where the direction of sure is lateral. Where pressures exceed safe bearing values, the joints are cr In cap-butting sets, ends of posts bear entirely on wood in cross grain, and und pressure the rate of compression is equal over the whole area of joint. In po sets, under heavy vertical pressure, the post tenons or HORNS must fail before grain wood in caps and girts can aid in transmitting pressure to the post. As compression is not the same in all parts of a joint, settlement in a stope is often C. T. Rice cites evidence (218) to show that settlement is greater and more in post-butting than in cap-butting sets, and that more auxiliary timbers are nee former to keep the floors level. Hence, many engineers prefer cap-butting sets vertical pressure, and practice is almost uniform in using them for heavy lat sure also, though arguments like the above indicate the superiority of post-but But, in heavy ground, filling follows mining closely and takes most of the lateral the smaller and more uniform rate of vertical settlement of cap-butting sets is argument for their use in such ground.

Square-sets may also fail by "jack-knifing"; posts swing out of line and ev girt or cap drops, allowing adjacent sets to collapse. Hence, wide seats for caps are desirable, and are commonly given in cap-butting sets. In post-but wide seats are obtainable only by reducing the size of the post horn. Long small cross-sec are always objectionable, as they break too easily. With timbe by 10 in, post-butting sets with adequate shoulders and horns are readily framed times, to give a wider bearing for either girt or cap, horns on posts are not section; they require a complicated framing machine. Note that girts are mo pull away from their seats under light than under heavy pressure.

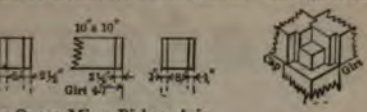
Empirical rules proposed for framing details: (*a*) length of the p should be less than its least cross-sectional dimension; (*b*) girts a should have a seat on the post not less than 2 in wide; a wider should sirable (218). SIMPLE FRAMING is best, especially for hand framing butting sets are generally simpler and cheaper to frame than cap-butt but the difference is small when framing machines are used. Com joints resist distortion better, because of the numerous shoulders, advantage is doubtful, as corners are points of weakness at which begins, and such joints are difficult to assemble if the timbers have b slightly crushed. SYMMETRICAL FRAMING is best. It costs less to place when the posts can be set either end up; caps and girts also should be s that either of two opposite faces may be placed upwards.

Examples. Fig 212 to 219 show post-butting types; Fig 220 to 227, ca types. At Copper Queen mine, Ariz (Fig 212), both cap and girt are dappe framing is used at Broken Hill, Australia, with a post horn only 4 in square. A 8 by 10-in girts are framed square on each end, the joint thus becoming the same Cananea, Mex (Fig 217). Girts fail at their bearings on the post; an 8 by 10- is cheaper and has same bearing area as the dapped 10 by 10-in. But, narrow vide smaller support for floor plank. Fig 216 shows same type of framing, but timber. The narrow shoulders supporting caps and girts indicate the limit 8-in timbers, when the posts butt; they should be used for light pressures only

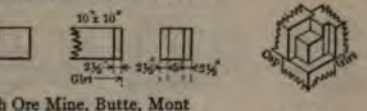
At the High Ore mine, Butte, the post (Fig 213) is unsymmetrical, with a 6-i top and 4-in at bottom. This secures a shallower mortise at top of set, whic to clean out and allows "flirting" the post (Art 53), if sets have swung sligh line; long, slender horns make flirting of posts difficult. Another way to reduce horns on posts of post-butting sets is to cut them of equal length and short leave a SQUEEZE-SPACE between them (Fig 215). At Vulcan mine the same used for 10-in and 16-in timbers, with post tenons 4 and 8 in square respectiv Squeeze-spaces are usually 2 in thick; if they are filled with a SQUEEZE-BLOCK of the posts bear entirely on cross-grain wood, as in cap-butting sets. Neithe spaces nor blocks are in general use in high stopes, as they allow excessive se Squeeze-spaces at tops of sill-floor posts may lengthen the time during which will remain open without reinforcement.

nd symmetrical; post horns are short and strong; t, so that its horn takes only part of the lateral 30), turned 90°.

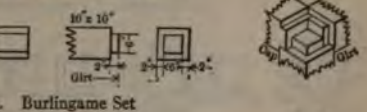
simplicity of post-butting sets when of large timber, uld be prohibitive in most places. These joints are



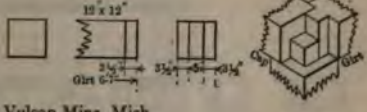
r Queen Mine, Bisbee, Ariz



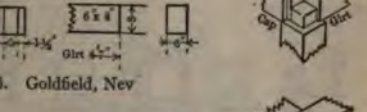
h Ore Mine, Butte, Mont



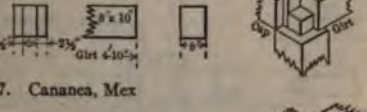
. Burlingame Set



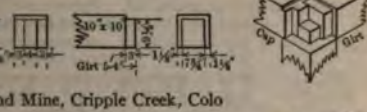
Vulcan Mine, Mich



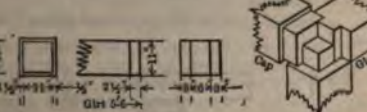
). Goldfield, Nev



7. Cananea, Mex



nd Mine, Cripple Creek, Colo



Homestake Mine, S Dak

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like those used by Deidesheimer in the Savage mine, Comstock Lode, Nev. an example of odd framing, possessing no apparent advantage.

Fig 224 is the original Eureka cap-butting set. The eccentric cap tenon p of a squeeze-block. Other forms of Eureka framing are used at Butte. F

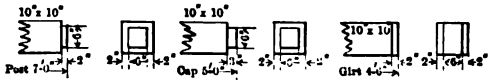


Fig 220. Anaconda, Butte, Mont

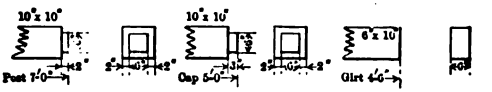


Fig 221. Anaconda, Butte, Mont

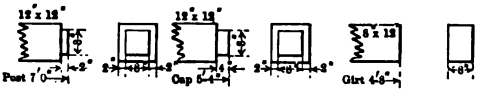


Fig 222. Leonard Mine, Butte, Mont

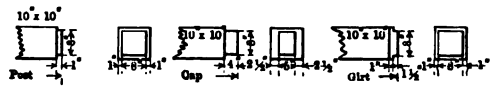


Fig 223. Syndicate Group, Butte, Mont

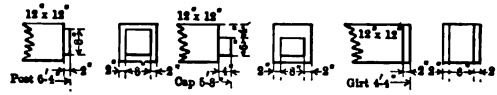


Fig 224. Eureka, Nev

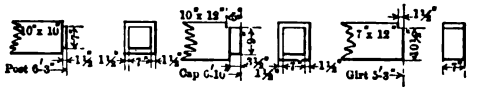


Fig 225. Leadville, Colo

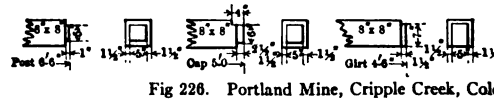


Fig 226. Portland Mine, Cripple Creek, Colo

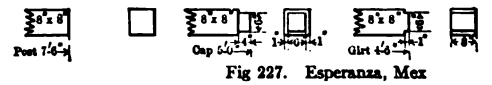
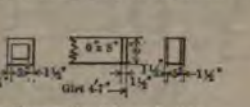


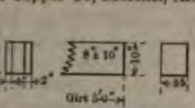
Fig 227. Esperanza, Mex

n, Colo. B. H. Dunshee regards this as the best
le, cheap, and retains full strength of timber for
Usually, a 6 by 10-in girt is strong enough, and
(Fig 221). Eureka framing for 12-in timbers is
g 222); the girt is 8 by 12. 12 by 12-in timbers

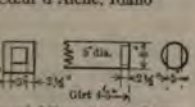
s wider supports for the horiz members; an ad-
ing the cap horn. Unsymmetrical cap and girt
ty of placing timbers.
olo, and Bingham, Utah. Bottom of post is un-
nmetrical. The only advantage is that the top
ers; hence, easy to clean out.



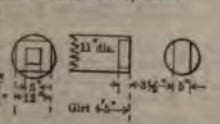
a Copper Co, Morenci, Ariz



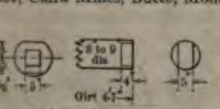
Cœur d'Alene, Idaho



vard Mine, Butte, Mont



Set, Clark Mines, Butte, Mont



gnon Mine, Butte, Mont

8-in timbers. Esperanza set is the High Ore set
ramed, members are all symmetrical, set is cheap
ht pressures. Portland set (Fig 226) has slightly
ical throughout. The use of one unframed mem-
out, as sawed timbers vary in cross-sec, unframed
s.

round and square timbers; they are planned
framing sets entirely of round timber.

nd posts and square caps and girts. It is well de-
all members are symmetrical and the set is easily
hoveling floor may be laid either cap or girtwise.
osts butting; its weak point is the slender upper
only 3-in; it affords good shoveling floors. Fig
st and girt. The bottom post horn is only 1.5 in

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ng, on the theory that it is easier and quicker to stand a post with a short ho
..t requires an unsymmetrical cap horn, and the small round girt of this set
use of a floor laid girtwise.

Round-timber sets (Art 50). Simplest joint is made by first squa
ends of timbers for a sufficient length to avoid interference of their cy
surfaces at the joint, which is then framed as with square timber. Th
the cost of squaring the whole log, an important item in hand work. Di
tages: the joint utilizes a maximum of only 60% of cross-sec of the tim
round timber weighs at least 40% more than square of equal joint stre

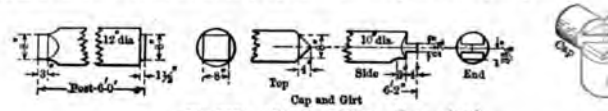

Fig 233. Chillagoe Mines, Queensland

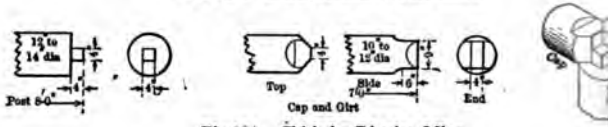

Fig 234. Chisholm District, Minn

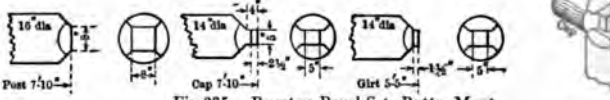

Fig 235. Brunton Bevel Set, Butte, Mont

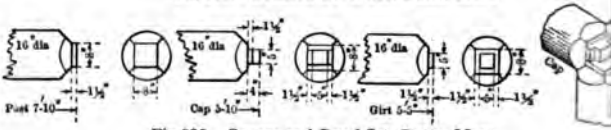

Fig 236. Square and Bevel Set, Butte, Mont

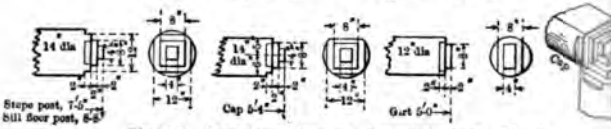

Fig 237. Step-down Framing, Anaconda, Butte, Mont

Fig 231, 232 show simple framings, open to the objections stated; also, they p
poor seat for the bottom of the post. Similar joints are used at Rossland, B C.
sets can be framed in machines of the block type.

Bevel or miter framing (Fig 233–236) brings the entire area of members in
at the joint. In general, the sets are hard to stand and block, and the bevels
distortion under pressure if the sets once get out of line. Post, cap, and girt of
set must be firmly blocked when the other members are missing. Many miter j
quire blocking on top before they can be blocked sidewise, or else simultaneous
in 2 directions. Either case increases expense and difficulty of placing. Fig 23
simple miter joint, to permit hand-framing of stunted, twisted timber; caps a
are identical (209). It has given satisfaction at Chillagoe mine, Queensland.
is a hand-framed set, for heavy vert pressure; a 1-in squeeze-space is left betw

id to prevent caps from splitting (221). It may the members missing. Fig 235 is a set designed is beveled. It requires top blocking, to prevent cked sidewise. The square and bevel joint (Fig utting round-timber sets. It is easily blocked, e to frame (222).

ning for round timbers is the STEP-DOWN SET the cutter-head saw, and costs little if any more . Full strength of timbers is utilized, the square easy blocking, and ends of posts and caps are eldom used (219). In general, better joints are nd caps are approx of equal diam. Upper side surface for flooring if laid capwise and to avoid

e Timber for Square-sets

ver freight rates, and is stronger per sq in, step-down framing (Fig 237) gives an effi- serious drawback to its use. But, round ne than square; caps must be slabbed to ing may be offset by value of slabs, if used nd-timber sets is difficult and expensive. inishing, and its price is rising. Use of ly in stopes where filling closely follows me trees of poor grade, or too small to go e square-set timbers at reduced cost (219).

In some round-timber stopes, square tim- d chutes, to cheapen work of reinforcement. sets; it provides a flat face for spiking lin- l than slabbed round timber.

er is stronger either with or across grain than erial. There are no data showing comparative ual diam (*i e*, 10 in diam and 10 in square). A as pine at Morenci, Ariz, showed equal strength at round timber deflected more and gave more ne load. A safe estimate is that round timber both compression and bending as square timber of this relation in substituting round for square eaper cost of the round.

r and Walls, Square-set Stopes

designed to give adequate foundation for

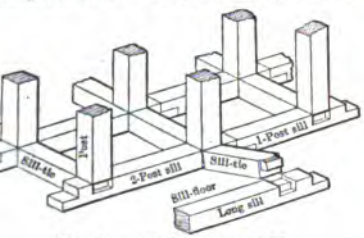

Fig 238. Sill-floor Timbering

m, consisting of long sills (stringers), mortised at d apart by sill-ties (girts, spreaders, or short sills)

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Long sills are also called 1-post, 2-post sills, etc, depending on number of pos
support. Sills are braced to stope walls by short pieces (butt-sills) cut to fit fr
ties. Joints between long sills are usually made under posts, with butt or halve
For simpler framing, see Fig 239. Length of long sills depends on character of

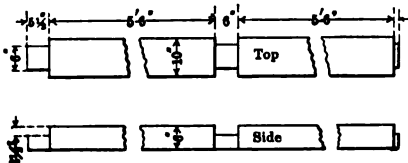


Fig 239. Sill Timbering, Homestake Mine

some ores will not stand
ported over more than 1
requiring 1-post sills. I
ground, convenience in l
determines max length
usually covers 2 or 3 set
of sills is from 5 by 10-
by 12-in or larger, depe
size of posts.

Long sills, firmly bra
good in open stopes, w
floor is in ore. Sills e
over 2 or 3 sets will brid
siderable opening, when
is subsequently mined,
easily picked up on po
stope timbering below. Framing for sill-ties (Fig 238) prevents them from drop
when this work is going on. Wedging is relied upon to hold simpler framing (
Sills distribute pressure and reduce settlement on a soft floor. In open stopes,
covered with plank or poles for shoveling; in filled stopes, a floor prevents runs
into stope below. Attempts have been made to set sills in line with correspondin
level above, but it is unnecessary, as stope timbers may swing 1 or 2 ft out of a
before the level above is reached. Size of stope and speed of work should be
so that sills will not rot before stope from below breaks through. See Bib (
replacement of rotted sills. Simple forms of sill often suffice. A 3-post sill of
timber is common for 10 by 10 posts, which have flat bottoms and are braced ca
2-in plank spiked on top of sills; light sprags brace the post and sill girtwise.

Examples. BISBEE, Ariz. Large, irregular masses of ore occur in limest
porphyry, with no well-defined foot or hanging wall; usually ground is heavy
be mined only in small sections. M. J. Elsing states (225): Sill-floor set de
nature of ground in which stope is started. If floor is waste, a flat-bottom pos
resting on a small block with 4 by 6-in spreaders between posts. If floor is o
mined from below, 10 by 10 sills covering 1 set are placed
capwise and braced apart with unframed 8 by 10 timbers
laid girtwise (known as a "cap-sill" and "tie"). "Mud-
sills" may be used instead, framed of 10 by 10 timber,
1 set long, running girtwise and braced with 4 by 6
spreaders. Cap-sill and tie are stronger, and are used
in either ore or waste around raises which must be kept
open (Fig 240). In very soft ground, even though floor
is waste, a cap-sill and tie give good foundation for sets
above, and, for supporting a floor covered with waste
filling, are easily caught up from below with stulls and
posts. In general, where floor is on ore and the ground
heavy, they are preferred to mud-sills. GOLDFIELD,
Nev. Sill-floor posts (10 by 10-in) have a flat bottom
and stand on 8 by 10 sills. 2-in spreaders are spiked to
sills between posts, but there are no braces between
sills. Lateral bracing for feet of posts is furnished by
a floor of 2 layers of 2-in plank spiked to top of sills (145). BUTTE, Mont. Amal
Copper Co often omits sills (80). Flat-bottom posts stand directly on the stope
on footblocks where the floor is low; 4 by 10 spreaders brace feet of posts cap
braces, girtwise. This practice is largely due to the time required to level
points on stope floor, for placing sills properly. In some camps, sills are bed
layer of muck for quicker placing and because the muck protects them when th
mined from below. Floor is sheathed with poles or old plank laid across the
The filling is well compacted by the time the stope comes up from below, and

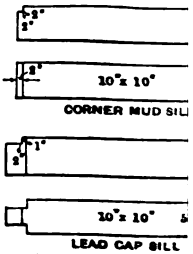


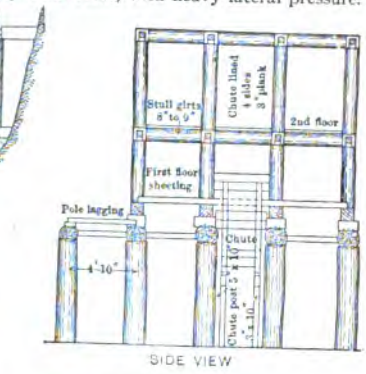
Fig 240. Sill Timb
Bisbee, Ariz

if sills were used. Sills are omitted under round-
re then difficult to aline. G. D. Moulthrop (219)
often of 2-in plank, set to grade with a carpenter's
centers of caps already in place.

longer and heavier than stope posts. In
es, the sill-floor sets carry great lateral and
traveling ways which must be kept open

ain sufficient headroom when settlement occurs, or
t (Art 52) (Fig 217 to 219 and 237). Their extra
ly, 8 to 12 in. 10 or 12-in timber is usual for sill-
10-in, respectively.
n trouble and expense of keeping tramming sets
sometimes be reduced by placing main haulage-
cuts under each stope.

ion at Stewart and Gagnon mines, Butte,
ely wide veins, with heavy lateral pressure.



Timbering, Stewart Mine

. Posts are selected timbers, 14 to 23-in diam.
ainst side pressure and postpones repairs. False
heeting on which
and caps aids in

stope floor, have
242) (80). This
timbers, unless in
hat harder to take
The level is called
tope is opened, the
Sill-floor posts are
Bedrock and stope
m, respectively.

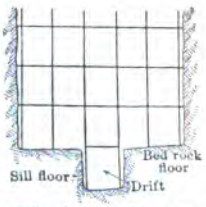


Fig 242. Leonard Mine; Vert Cross-sec through Stope

uts require special
rves and for clear-
oints a double length cap (and sill) is used, with
a truss set (Fig 255) may be built over the long
m, Utah, an arch set (Fig 243), built on the sill
(226). Its applicability depends on height a

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width of sets and size of car. Special timbering for turnouts is avoided by u tables; most engineers prefer timbering.

Timbering on walls. Standard sets are often impossible in min irregular walls of dipping orebodies. Square-sets must have firm supp footwall and follow the hanging closely enoug proper blocking of timbers.

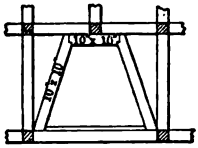


Fig 243. Arch Set at Turnout, Bingham, Utah

In firm ground, short posts, caps, or girts are use walls, with their ends in hitches or wedged agains Short posts on the footwall are GROUND POSTS (Fig of the bearing block *a*, and depth of hitch depend o of ground. CAP-SILL (butt-cap or butt-brace) is used on flat dips or soft rock, to tie the foot of the post to stope timbers and distribute pressure from the post over a larger area than that of the ground post. SHORT SETS are common on hanging-wall side. Fig 245 shows a form used by L. S. Cates at Bingham, Utah (211). Cap *a* is not framed into post *b*, but is spiked to it and supported by a piece of 2 by 8-in lagging *c*, spiked to *b*. Lateral motion of the cap is prevented by a brace of 2 pieces of 2 by 8 lagging, spiked in place and reaching to next set. In soft ground, a good but rather costly bearing for sets is secu ting into the walls far enough to allow use of full size se where the dip requires an offset in the floors. ANGLE SETS (Fig 246), or HALF-ANGLE SETS (Fig 247), are for following the hanging wall and supporting lagging. Angle braces and sets may also be used to support sets on footwall (*b*, Fig 244). Details of wall timbering are varied locally. In open stopes, lagging may be used on either wall over areas of shelly ground. In close-filled stopes, a exposed at one time is small, and lagging is rarely necessary. For use of lagg walls in the block system, see Leonard mine, Art 47.

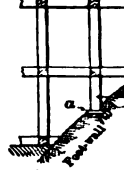


Fig 244. T against Slopin

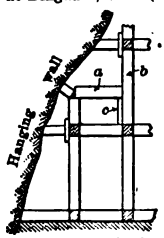


Fig 245. Timbering against Inclined Hanging Wall

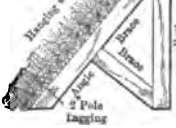


Fig 246. Angle Set (*E & M Jour*)

Fig 247. Hal (*E & M*

52. Reinforcing Square-sets

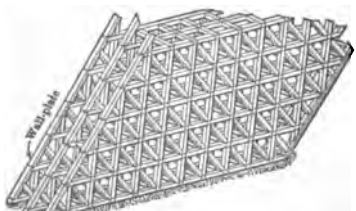


Fig 248. Complete Reinforcement, Deidesheimer System

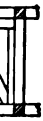
Complete reinforcemer cured in the original De system by wall-plates braces (Fig 248). Some open stopes were thus m ore. But no system of tin support heavy rock press length of time. Wall-plat plete angle-bracing have carded because of their bracing interferes with ways and chutes.

Modern practice in heavy ground regards the timbers as temporary su support being furnished by filling, kept within 2 or 3 sets of the back. T

by rapid working in stopes of small area. Even wing out of line and require reinforcement. Re- ors, where haulageways must be kept open for a from filling in stope above (Art 51). Reinforce- pes in amounts varying widely with local condi- ground, the sets serve merely to support slabs, ; in that case, no reinforcement may be needed.

(Fig 249), either round or square, have no o tighten them, but a driving fit is better.

right angles to hanging wall; practically, this is ts tending to swing out of line are effective only pressures. If used in the upper sets of open or irts to "ride" off the posts when the sets above



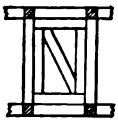


Fig 251
R-FRAME SET

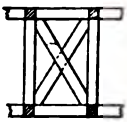


Fig 252
DIAMOND BRACE

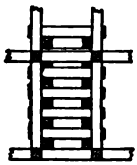


Fig 254
CRIB

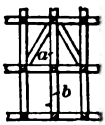


Fig 255
TRUSS-SET

Tenons on posts at the ends of an angle-brace nts begin to fail, modified braces may be used, -frame set (Fig 251). The X-frame, or diamond races are also used to take weight off a post show- good for reinforcing drift posts on sill floors, as it way. The same bracing, with post *b* omitted, is ith the post *a* omitted, it will take the weight of way. Sets thus braced are TRUSS-SETS; their use ring is uncommon, as it involves keeping 1st-floor port stope floors, where a post is removed to allow

elpers (Fig 253), are convenient for pick- rts; they are round or square, and vary in

3 or 4-piece reinforcing sets, like the N-brace , with angle braces omitted. The timbers mbers, but joints are not framed. They may l of angle-braces; like the latter, they are oad cases of swinging.

rock filling, afford strongest reinforcement.

built between the posts of sets (Fig 254). They ock. Bulkheads, of criss-crossed timbers, skin to cost more. When used in open stopes to arres'

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swinging or crushing, or to prevent runs of filling, the bulkheads or cribs ex
wall to wall and from floor to back. In close-filled stopes, trouble from sw
may be stopped by building one or more cribs 4 sets square, on top of the fille
rarely more than 2 or 3 sets high. Each layer of the crib is of 6 pieces equa
to 2 sets. At Centennial-Eureka mine, Utah, continuous cribs, 1 set wide,
at intervals of 5 to 10 sets to support a domed stope (Art 46), about 100 ft w
ft long; they were of 8 by 8 timber and required about 1 300 bd ft per set.

53. Miscellaneous Details of Square-sets

Fire protection. Disastrous fires have often occurred in square-set st
Leonard mine, Butte, fire doors are placed in crosscuts from shaft in each le
principal drifts. 1-in water pipes, connected with city water supply, are l
working stopes, a length of 0.75-in hose being attached to end of each pipe.
reducing valves are necessary in main pipe in the shaft. Timbers in stopes
are wet down at beginning of each shift. WATER is also used FOR LAYING D
of a run of filling. This use of water also improves hygienic conditions (8
ventilating currents in Leonard mine rapidly dry out timbers and filling.

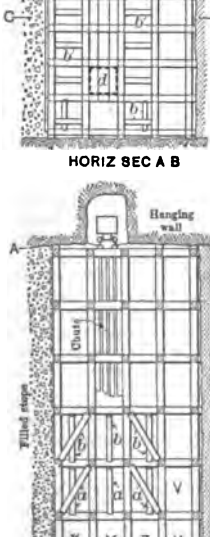

Fig 256. Recovering Timber from Square-set Stopes

Crowning floors. Trouble from settling o
not serious in low or narrow stopes, nor when
such that sets are repeatedly offset along th
In high, wide, vertical stopes, timbers settle co
and faster in middle of stope than along wa
troubles increase with height of lift. Small
are often met by SPECIAL POSTS, 1 or 2 in l
regular stope posts and kept in readiness. A
Ariz, with block-system stoping, the sill flo
block is arched or crowned to compensate
Length of posts increases gradually towards
stope, where the posts are 2 in longer than t
walls. Crowning is repeated in one of the upp
soon as the arch effect is lost. At Butte, in b
wide and 200 ft high, crowning is necessary o
or 4th floor above a height of 120 ft, the po
the outside rows being 3 in longer and thos
middle rows 8 in l
the regular posts.

Flirting posts.
are slightly distor
post may sometim
erly placed by cutt
of the bottom horn
out the correspondi
this is known as fli
It is usually done n
in cap-butting tha
butting sets, becau
of the former have
(Art 49).

Blocking, if car
materially reduces
sets. It is also i
crush under the ini
walls or back, the
the main timbers.
be blocked at joint
bending strains in t
a long block (spra
sary to reach a wa
have a headboard, so that the pressure will come onto the timbers through w
grain. Good blocking is especially necessary over the posts at back of stope,

ouble-end machine) and delivering timbers at mine yard is included; for stope ig 229; for method of mining, see Art 46. Square-set timbers: 1 705 posts, 1 449.25; 1 657 caps, @ 45¢, $745.65; 1 507 ties, @ 45¢, $678.15; 507 10-ft ties, 451.23; 180 "helpers," @ 43¢, $77.40; total, $3 401.68. Miscellaneous stope t 000 pcs 3-ft cribbing, @ 18¢, $1 080; 5 220 pcs 4-ft cribbing, @ 25¢, $1 305; prag poles, 10 ft long, @ 64¢, $608; 107 loads slab lagging, @ $3.50, $374.50; 2 edar lagging, @ 4.5¢, $112.50; 180 000 bd ft, 2-in plank lagging, @ $13, $2 340; vedges, @ 0.5¢, $137.50; total, $5 957.50. BUTTE, Mont. B. H. Dunshee states mount of auxiliary timber in a stope, compared with that in square-sets, depend n number of sets and width of stope. Chutes and manways are 25 ft apart, and gures for the bd ft required in sets 5.33 ft square are: top lagging (4-in) for 1 hute, 1 floor high, (2-in lagging, 277; 4 by 10-in lumber, 111; 2 by 6-in lumber, 1 anway, 1 floor high, 2-in lagging, 224; timber slide, 1 floor high, 53; total, 791

Table 39. Amounts of Timber Used in Square-sets

Fig No	Name of mine or set	Dimensions, c to c						Cu ft of ore per set	Bd ft per set*
		Height		Capwise		Girtwise			
		ft	in	ft	in	ft	in		
212	Copper Queen	7	10	5	0	5	0	195.8	141.7
215	Vulcan, Mich	8	0	7	5	7	0	415.3	257.0
....	Broken Hill, N S W	6	2	6	0	5	0	185.0	137.5
....	Savage, Nev	7	2	4	9	4	9	161.7	184.0
217	Cananea, Mex	7	4	5	3	5	3	202.0	133.6
218	Portland, Colo	7	3	5	8	5	8	233.0	149.3
219	Homestake, S Dak	8	3	6	0	6	0	296.9	238.5
220	Anaconda, Mont	7	6	5	0	5	0	187.5	137.5
221	Anaconda, Mont	7	6	5	0	5	0	187.5	122.5
222	Leonard, Mont	7	8	5	4	5	4	217.6	185.3
224	Eureka, Nev	7	0	5	8	5	0	198.3	196.0
....	Bingham, Utah	7	4	5	0	5	0	183.3	113.3
226	Portland, Colo	7	0	5	0	4	11	172.0	85.3
227	Esperanza, Mex	8	0	5	0	5	0	200.0	90.7
216	Goldfield Consol, Nev.	7	8	5	0	5	0	191.7	83.6
225	Leadville, Colo	7	0	6	10	5	10	279.0	157.7
228	Morenci, Ariz	7	2	5	0	5	0	179.1	84.9†
229	Cœur d'Alene, Ida	8	0	5	0	5	4	213.3	104.2†
230	Steward, Mont	7	0	5	2	4	10	174.8	93.5†
232	Gagnon, Mont	8	3	5	0	5	0	206.2	89.3†
234	Chisholm Dist, Minn	8	1	7	4	7	4	434.0	124.9†
235	Brunton Bevel	8	3	5	10	5	10	280.5	172.8†
236	Square and Bevel	8	3	5	10	5	10	280.5	200.6†
237	Step-Down	7	9	5	4	5	4	220.4	132.0†

* These figures are for a corner set (1 post, 1 cap, and 1 girt). † Bd ft in ro er is computed by QUARTER-GIRTH RULE: area of cross-sec = 1.97 × square rhich gives cross-sec of largest square timber that can be sawed from a given l

Data on framing. At ROSSLAND, B C, in 1903, round timbers averaging 16 were framed by hand. Aver work for 1 carpenter per 9 hr was 21 posts, or 21 6 caps. Cost per set was about 55¢ (wages, $3.50). Cost of same set, machine ot over 30¢ (209). At HOMESTAKE MINE, So Dak, in 1903, some 12 by 12 tim ramed by hand at 60¢ to 65¢ for a cap, girt, and post (wages, $3). Posts cost nore than caps or girts. Costs show that 1 man framed 13 or 14 posts, or 14 to r girts, per day (104). B. H. Dunshee gives following figures for BUTTE: 1 c rames by hand 10 posts, 10 caps, or 20 girts, in 8 hr from 10 by 10-in timbers. nd framing machine of the block type, with horiz and vert saws, operated by rames 300 pcs (posts, caps, or girts) per 8 hr. This is for 10 by 10 timber or rou o or 12-in diam. 2 sawyers, running a single-end block-type machine, frame aps, or girts, from 10 by 10 timber, or 200 posts, caps, or girts of round timber iam per 8 hr. Cutter-head type of framer (with 2 sawyers), used for framing th oints (Fig 237), makes 900 to 1 000 posts or caps, or 1 000 girts, per 8 hr.

Square-sets (Ibex Mine, Leadville, 1902)

	Drift posts 10 by 10-in, 6.5 ft long	Drift caps 10 by 12-in, 4 ft long	Sills, 6 by 8-in, 6 ft long	Raise cribbing, 4 by 8-in, 6 ft long	Wedges
	144	97	48	1 156	16 800
	6.75	8.75	2.5	24.75	28.75
	1.52	2.93	1.69	0.73	0.05

elays were 0.75 hr. P. B. Scotland states that a e framer, used by Ariz Copper Co, run by 2 men, re caps and girts (Fig 228) are cut in a single-end aps in 8 hr.

machine-framing on a large scale in well arranged head sawyer, $4.50 per 8 hr (229). Girts can be eceive only 1 dap on a side. Framing ordinary , as they are heavier and slower to handle. Costs harges, but not upkeep and amortization.

Butte, 2 timbermen stand and sprag 7 or 8 sets ate operation by the most expert timbermen; 2 rt 46). P. B. Scotland states, concerning mines en in placing and lagging stope sets is 2 to 3 sets per e that, if a stope is ready to be timbered, 2 miners ock down and cover the top with lagging in about or more, as loose ground must often be picked

r costs 2 to 3 times as much as on upper floors, flat holes, the face is not undercut, and shoveling mine, Butte, (Art 46), a standard crew consists of shovelers, 1 man on filling, and timbermen and This crew carries up a 25 by 50-ft stope, 100 ft ht of 200 ft, extra time being required in the 2nd ers and for crowning floors (80).

Complete square-setting is no longer practiced at

an Open Stope, Homestake Mine, So Dak

o of ces	Lin ft, or bd ft	Cost of material	Labor, sawing and framing	Total cost
421	3 650	$ 474.50	$ 96.83	$ 571.33
077	16 616	2 160.08	477.71	2 637.79
410	13 255	1 723.15	506.10	2 229.25
261	12 435	1 616.55	474.81	2 091.36
....	4 537	266.85	22.69	249.54
020	75 906	3 795.30	379.53	4 174.83
410	4 025*	64.82	30.00	94.82
352	784*	13.33	11.76	25.09
....	.....	.....	.....	327.93
....	.....	.....	.....	824.12
14	117*	1.99	3.50	5.49
....	.....	.....	.....	4 745.00†
....	.....	.....	.....	793.97
....	.....	$11 174.74	$2 057.08	$18 770.52a

a) Total cost for timbering, $0.257 per ton.

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Homestake mine (Art 69), but the table gives a good idea of cost of such w stope produced 73 000 tons of ore. Timbermen's wages, $3 and $3.50 per s figures show that the stope was about 10 sets wide, 37 long, and 6 high. timbers, 13¢ per lin ft, or $10.63 per M for 12 by 12 timber. Sets were 6 height of sill-floor sets, 9 ft; height of stope sets, 8.46 ft.

Table 42. Cost of Square-setting in a Small Mine, Chihuahua,

		Total days	Days per ton	Tons per man per day	Total cost
Labor	Shift bosses (share)	52.63	0.00304	327.8	$ 81.75
	Timbermen	662.7	0.03842	26.1	664.26
	Helpers	625.25	0.03624	27.6	464.64
	Carpenters (contract)	146	0.00846	118.1	214.63
	Total labor	1 486.58	0.08616	11.61	$1 425.28
		Quantity	Quantity per ton	Tons per unit	Total cost
Supplies	Lumber	.....	.....	.....	$1 935.10
	Candles, No 15L	9 493	0.556	1.83	78.93
	Axes	3	.....	.....	2.15
	Adzes	2	.....	.....	2.32
	Bar iron	273 lb	0.0158	63.8	10.48
	Nails	205 lb	0.0119	91.7	12.10
	Lag screws	.....	.....	.....	2.92
	Washers	.....	.....	.....	1.47
	Rope, 40-ft	39	0.0023	442	8.77
	Piles	.....	.....	.....	2.38
	Total supplies				$2 056.62
	Total cost of timbering stopes				3 481.90

R. H. Allen gives Table 42 (129). Mining is by hand drilling (see Table 25 for ore occurrence and mining method). Stope sets are of 16-ft logs, 7 to 8 small end; lagging is of 6-ft slabs; posts, 6 ft 2 in high, are placed 4 ft 8 both capwise and girtwise; stopes are kept filled with waste to within 1 fo Figures are for mining 17 253 tons, in 7 mo of 1912. Rate of exchange, 0.49 timbermen, $0.994; helpers, $0.7455; underground laborers, $0.373 to $0.6 all native labor. 16-ft poles, 6 to 9 in diam, cost $0.895 to $1.566 each; s per metric ton. See also Sec 21 for cost of labor and timber at mines using

55. Applicability and Limitations of Square-sets

In early days, the method was standard for mining nearly all large oreb U S. Increasing cost of timber, and introduction of shrinkage, filling, and cav have narrowed its field of use.

Open stopes. Experience shows that open sets do not afford support for weak ground; timbers rot, or more often crush unde caused by the extension of stoped areas. This is serious, as the large stopes may set up a slow crushing action ("creep"), which ex adjacent unmined blocks and renders their extraction difficult a The field for open square-sets is in orebodies too wide for stulls, and ground is strong, but requires prompt support to prevent slabs fa back and walls. These conditions preclude use of shrinkage stoping native is a filling method, the choice being governed by relative of these systems, as stated below and in Art 67, and by their rela

H. C. Hoover cites following cases (20): At Le Roi mine, B C, square-set 21¢ per ton; at Ivanhoe mine, W Australia, filling costs about 22¢ per ton of ber costs less than $10 per M at Le Roi; at Ivanhoe, $50 per M. Cost and

st of square-setting at Ivanhoe would be about 65¢
available for filling; cost of filling with rock from
walls, might well be 65¢ per ton of ore.

ntly justifying complete filling for square-
nent support of the surface or of overlying
; (*b*) weak ore requiring complete and im-

aving method is often cheaper than and preferable
are limited to certain types of orebody (Art 83).
port the stope back, straight filling is better than
her governed by the advantages and disadvantages
ages of different forms of square-set stope.

uare-sets have a wide field of use (Art 69).

, not covered above: (*a*) reliability; prop-
ining almost any kind of ground; (*b*) safety,
any time may be adjusted to its strength;
nded or contracted at will, irregular string-
int, dikes or waste inclusions left unmined,
any floor; (*d*) good facilities for sorting in
st be separated from waste, sulphide from
centrating ore; (*e*) good facilities for han-
the stopes; (*f*) good ventilation; (*g*) walls
ed if desired.

ting, not covered above: (*a*) high cost, due
good grade; (*b*) danger of fire, which has
cts where the method is used. Fire risk is
s (see Fire prevention, Art 53).

ts cost, square-setting is necessary and use-
and as an adjunct to other mining systems.
sts by the following devices (P. B. Scotland):
quare; (*b*) use second-class and smaller tim-
size for all kinds of ground; (*c*) purchase logs
bd ft per ton; (*d*) accept mixed shipments,
g saws and use slabs for lagging; (*f*) install
nbers on surface and underground; (*g*) frame
g freight charges; (*h*) discard sills in all but
e fact that accurate alinement of timbers is
g closely follows mining.

licing System (220, 228)

was developed at Calumet & Arizona mine, Bisbee,
heavy sulphide ore in limestone. Hanging wall is
ebodies contain little waste and rarely exceed 50
sets failed under the weight of soft, heavy ore.
ful in some oxidized orebodies of this company; a
ematite at Queen mine, Negaunee, Mich (136).

are-setting along the sides of a small block of
en mined, and hanging wall timbered. Under
ar is sliced downward by underhand stoping.

ven, preferably along one wall of deposit; from it
eads, are driven to opposite wall on 20-ft centers
s. A vertical square-setted raise to level above,
es as a timber and manway, and later as a waste
gives access to manway side of raise on first flo-

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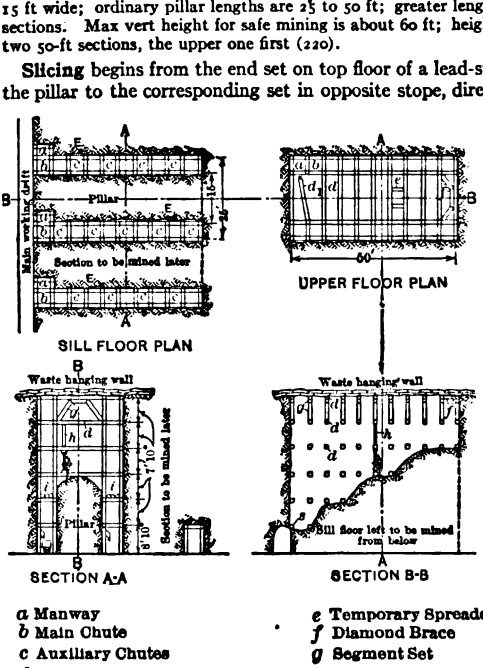
Small excavations, *E*, on sill-floor plan, are shot out at top of alternate sill-fl slice-leads, and a chute built in each; these cuts are kept small and on same jacent crosscuts, otherwise they weaken base of the pillar between 2 slice-le can not always be run into slice-leads; if not, the first set alone serves as a loading cars in main drift. Two adjacent slice-leads are next carried to hangi flat-back square-set stopes, 1 set wide, called lead-stopes; broken ore form slides to chutes; little mucking is necessary. This preliminary work cuts o 15 ft wide; ordinary pillar lengths are 2½ to 50 ft; greater lengths are mined sections. Max vert height for safe mining is about 60 ft; heights of 100 ft ar two 50-ft sections, the upper one first (220).

Slicing begins from the end set on top floor of a lead-stope, by drifti the pillar to the corresponding set in opposite stope, directly under th wall. Ore by gravity an inclined each lead- center of pi cessive drift until the wh the pillar is



a Manway
b Main Chute
c Auxiliary Chutes
d Stringer in position
d_1 Stringer being swung in position
e Temporary Spreader
f Diamond Brace
g Segment Set
h Temporary Brace
i Grizzly

Fig 257. Mitchell Slicing System

Stringers (are placed u ing wall as so is room, wi lagging and b secure back and timbers ers, 15 ft lon 10 or 8 by 10- with 2-in ten into square-s stopes, are pl d_1. A cut mu in a lead-sto one end of st swinging its place. Topm ers are supp porarily by st on the pill enough ore is allow placing between next of square-sets replaced by s

ment sets *gg*, of 10 by 10 timber (sec *AA*, Fig 257). This braces the sets in lead- transmits to them pressure from hanging. Rest of pillar is stoped underhand in slices, with hammer-drills, or piston-drills mounted on columns, which are secured *h*. Stringers, usually of 8-in timber, are placed as stope goes down. Temp spreaders *e* are put between stringers near blasts. Braces *ff* are placed as requir lateral pressure at end of stope. Most of broken ore is thrown into lead-stopes a little mucking being necessary until 1st floor is reached. Slicing is continued t leaving sill floor to be mined from below. A plank floor is then laid on top stringers *s*, the inside of the square-sets in both lead-stopes is covered with a and the stope is filled with waste through the raises to the level above. This l stopes open and ready for attacking adjacent blocks; in heavy ground, the l must be filled. Under favorable conditions, 50% of the stringers may be re filling proceeds; they are stored in the open square-sets alongside and reused ing section. No economy results from recovering timber unless it can be use Small amounts of waste may be stored on upper floors of lead-stopes, or on do

facilitated by keeping stope faces as in sec *BB*; at end of stope nearest waste raises, so that the n while mining is in progress on upper benches; he filling. Broken ore also runs more readily to n.

l are a saving in labor, timber, and powder, as vantages are the rapidity of mining and the large At Calumet & Ariz mine, 5 to 6 tons output per opes; ordinary slicing work reaches 3 or 4 times s per man-shift have been obtained. Table 43, for the Mitchell system of 20% to 30%.

Mitchell Slicing System and Square-setting

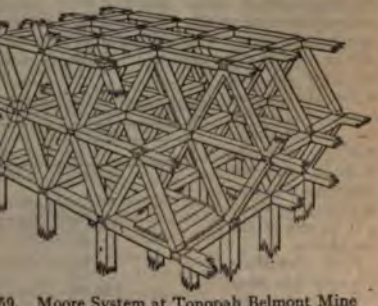
	Labor	Explosives	Timber	Candles or carbide	Air	Other supplies	Total
l...	$0.73	$0.06	$0.34	$0.01	$0.12	$0.04	$1.30
....	0.60	0.05	0.25	0.01	0.10	0.04	1.05
nd.	0.51	0.05	0.20	0.01	0.06	0.02	0.85
nd.	0.49	0.04	0.19	0.01	0.04	0.03	0.80
and	0.43	0.03	0.15	0.01	0.03	0.01	0.66

d. These costs obtain under normal conditions 3.75 for muckers. † Oxide ore in place averages in place averages 9 to 10 cu ft per ton.

locks to be sliced must be at least 20 ft wide, 25 t for one stope is about 50 ft. System requires a vall; it is not useful where lateral pressures are te, but is flexible and easily changed to regular ll.

nbering System (232)

-setting, wherein the posts are replaced by he caps. This forms a series of triangular



59. Moore System at Tonopah Belmont Mine

l *AB*. If a member *mn* fails, the force acting and *mp*; a corresponding failure in ordinary several sets.

261, showing details at Tonopah Belmont mine, e, and 5 ft 11 in girtwise. Diagonals are inclined

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65° (theoretical angle should be the dip of hanging wall, but is not practical; 6
able for most cases). Fig 259 shows framing of triangular sets and their junc
ordinary square-sets. Fig 260 shows joints *A* and *B*, Fig 259. All timbers are p

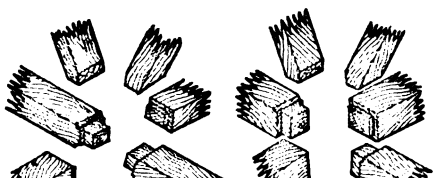

Fig 260. Details of Joints, Moore System

wise in the set; caps a
for convenience in
it is needless to ma
stronger in one direc
another. Drift sets a
by substituting 10 by
cal posts for 2 diagon
floor. Along vertical
of stopes a half-cap
on alternate floors,
at wall end by a 4 by 1
On inclined walls,
of diagonals can be c
fit, or in strong grou
caps may be set in h
Under direct vert
pressure, the stren

Moore set (of 8 by 10 and 6 by 10 timbers) is about the same as that of a squ
same dimensions with 10 by 10 cap and post and 8 by 10 girt. Under oblique s
stronger, due to the triangular panels, but, to utilize this strength, joints mu
from swinging girtwise. Moore set has no advantage over square-sets of equ

Table 44. Comparison of Corner Sets

	Dimensions of set, center to center	Bd ft per corner set	Cu ft ore per set	Bd per cu f
Moore (cap, girt, 2 diags)	7 ft 6 in by 5 ft 11 in by 8 ft	169.5	355	0.4
Square-set *	7 ft 6 in by 5 ft 11 in by 8 ft	161.1	355	0.4
Square-set †	5 ft 4 in by 5 ft 4 in by 7 ft 10 in	137.7	222	0.6

* Corresponding to Moore set. † Ordinary set, 10 by 10 post and cap, 8 by

either timber consumption or amount of timber placed in cross-frames; but t
in cross-frames is better placed to resist distortion. Advantages claimed: (
strength than square-sets for same amount of timber; (*b*) lighter timbers to hand
account of greater strength, filling is not required in some stopes, or where requir
not be kept so close to face; (*d*) flexibility, since the angles of diagonals can
(*e*) drift set construction is very strong; (*f*) chutes can be run diagonally or vertically; in latter case, one dimension of chute is limited to about 3 ft in sets of the size shown.

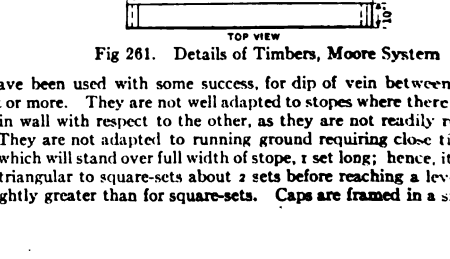

Fig 261. Details of Timbers, Moore System

F. S. Bradshaw furnishes data from Tonopah Belmont mine: Moore sets have been used with some success, for dip of vein between 75°, and width of 15 ft or more. They are not well adapted to stopes where there movement of one vein wall with respect to the other, as they are not readily r with angle braces. They are not adapted to running ground requiring close ti but rather to ground which will stand over full width of stope, 1 set long; hence, it sary to change from triangular to square-sets about 2 sets before reaching a lev Cost of framing is slightly greater than for square-sets. Caps are framed in a s

wing crosscut saw, with same number of operations by hand. Posts are framed by a swing saw, with posts. re-sets. per ton

Fig 262. Leaning Stope-set, Cal

stems

square- ng wall, a mine, was to The sys- oned in

used at ith soft At Argo- square- 30). Advantage claimed is that posts can be set n is such that square-sets can not be placed with posts superposed in the short time that the ground will hold. Stull timbering without posts will not hold; walls swell and often require lagging. Round timber is used; posts are 8 ft long, and 4 ft apart along strike; joints between posts and stulls are braced longitudinally by unframed sprags. Stope timbering starts from 2 stringers, one on each wall, blocked up from drift-set below, and separated by lagged stulls carrying filling. Swell of walls tightens timbers, so that they support filling even when drift sets are destroyed by mining the chain pillar from stope below.

Stringer sets (formerly used in Ontario mine, Utah, in wide parts of a steep vein) consist of a horiz cap (STRINGER), from wall to wall, carried by posts 5 ft apart. Caps are braced by girts opposite posts. No framing was done, 60 d spikes being used at joints. Max length of stringer, 16 ft; for wider stopes, 2 stringers were butted, with a post under the joint. These sets suited the solid quartzite walls at Ontario mine, but gave trouble when tried in weaker ground elsewhere. They require nearly as much timber as square-sets, and as much labor, when latter are framed by machine.

Fig 263 shows similar timbering at Morning mine, Coeur d'Alene, Idaho. Wall rock swells, causing side pressure great enough to broom out the ends of 24-in red fir caps, which are therefore protected by soft-wood blocking, removable for relieving pressure. Close filling is necessary; often within 1 floor of back. This system is in general use through the district for widths less than 15 ft (231).

ng

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FILLED STOPES

59. General

Definitions. FILLED STOPE, as the term is used here, is one in wh for walls and men and, at times, for back of ore is furnished by waste r sand, etc; materials so used are called FILLING or GOB. Classifi METHODS WITH CONTEMPORANEOUS FILLING, in which breaking of ore more or less closely by placing of filling; for descriptions, see followi filled square-set stopes (Art 45 et seq) may also be classed in this METHODS WITH DELAYED FILLING, in which filling is not an integral stoping method, but is introduced later to prevent eventual collapse o may be used in open, timbered, or shrinkage stopes. Examples: Me Art 35; Tonopah Mining Co, Art 37; Republic mine, Art 42; LeRo 46; Mitchell slicing system, Art 56; Homestake mine, Art 69; also and flushing, end of Art 64 and Art 93; also Art 110.

General plan. In methods (*a*) above, the orebody is excavated each practically an open or timbered stope, filled wholly or in part befo ground is attacked. Size of section mined before filling is adjusted of ore and walls; details vary with size and shape of stope and sourc methods of maintaining haulageways and of mining level-pillars are va strength of walls and ore and width of orebody. With few exceptio (*a*) are limited to overhand flat-back, stepped-face, or rill stopes (Ar

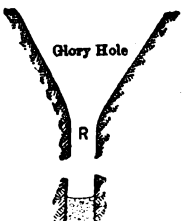

Fig 264

Source of filling. Cheapest filling is wast avoidably broken with the ore, which can be so the stope (Art 60); amounts of waste produced are usually insufficient, and must be supple other material.

Excess filling may be supplied by: (*a*) Shooting of lode; this obtains filling with little or no handling, ible if wall rock is strong; in weak rock, such excava a strip of unsafe ground alongside of ore in back of st suing, Art 61). (*b*) Driving crosscuts into walls; th duced is distributed laterally in stope in barrows o Inclined raises in walls serve same purpose, discha into the stope by gravity (Nevada Wonder mine, Art cuts and raises serve to prospect the walls, and henc able for obtaining filling where parallel stringers are a Methods (*a*) and (*b*) are used mostly in flat-back relatively small amounts of filling required in excess o sorting. (*c*) Using waste brought from surface; it may consist of mill tailin waste rock which can be broken cheaply in an open cut. Such filling, usually ground through FILLING or WASTE RAISES, is transported laterally along levels in cars (occasionally by belt conveyers) and dumped through raises into the stopes. Fig 264 shows mode of obtaining waste rock at LOS PILARES MINE, Nacozari, Mex (150). Glory-holes (Art 100) are opened in barren ground on the surface around raises *R*, which are 15 ft sq. Rock is broken with deep holes and heavy charges, and falls by gravity into the raise, at bottom of which is an offset connecting with drift *T*, and main waste raise *W*. Top of *W* is covered by a grizzly of heavy timbers 18 in apart, faced with steel, on which large boulders are broken. These raises terminate in bins, waste is distributed to stopes by electric haulage; total cost here of filling placed

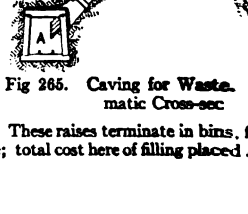

Fig 265. Caving for Waste. matic Cross-sec

ton. There are many similar plans for obtaining
n be utilized by driving drifts and raises under them.
work, thus saving or reducing cost of hoisting it.
r conditions, this gives cheap filling. Two parallel
n in country rock at a safe distance from the mine
ft wide and timbered with stulls, leave a V-shaped
tes (Art 91) are erected in drifts, say 6 or 8 ft apart.
f *S* caves and the material is drawn as needed into
e only in rock which will cave when thus undermined
stopes and reusing it in lower ones; feasible where
ebody permanently. Allowance for swell of broken
and amount of filling materjal.

e Art 60 to 66.

ped-face Overhand Stopes, Narrow Veins

lipping veins of strong ore, up to 15 or 20 ft
strong (Art 67).

n narrow veins (Art 15). If filling comes from
y for delivering it into the stopes; main raises
should be continuous through successive lifts.

nd stopes, Art 26 to 28).

shows common practice in western precious-
ike open overhand stopes (Art 36). Ore from

NE OF VEIN CROSS-SEC A B
Filled Stope, Narrow Vein

ed, and the waste used for filling other stopes.
lls or drift-sets (for choice, see Art 36).

ice, broken with light shots to avoid damage to level
on timbers; ore from first back-stope is often loaded
ned by removing 1 or 2 pieces of lagging. Before
d passages *C* (Fig 266), called CHUTES, MILLS, MILL
50 ft apart along the stope (Art 91). Work then
ing. Chutes are built up periodically, to keep their

by timbered passages, like chutes; usually, manways
ide of 2-compartment chutes. Openings *D* (Fig 266)
rly work in stepped-face stopes. This description
d in sorting to fill the stope; if this be insufficient,
in Art 59.

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Sorting in stopes is usually possible only when ore minerals are
nized, where ore and waste differ markedly in appearance, or ore occu
bands or patches. Underground sorting is not feasible where fine
required to separate ore from waste.

Sorting diminishes amount and increases grade of ore secured. This may
and operating costs for handling and milling an ore, and give more profit than
by treating entire contents of vein; it also reduces amount of outside waste
filling. Sorting may result in a profit from ore of apparently too low a grade
illustrated by following figures from a western silver mine. An 8-ft vein
shown in Fig 266; about 40% of material broken, carrying $2 per ton, was le
waste; remainder was sorted and sent to a concentrating and cyanide plan
tion, 90%; total cost of mining and treatment, about $9.90 per ton milled
had to assay $9.90 ÷ 0.9, or $11 per ton, to pay expenses; minimum aver
matter which could be broken to produce sorted ore of this value was (60 ×
$2) ÷ 100, or
Calculations
advisability
similar to tho
Resuing, Art

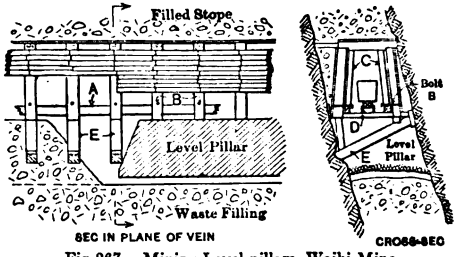


Fig 267. Mining Level-pillars, Waihi Mine

Mining
presents no
ore is strong
is timbered
support fill
above. W
that of brea
stope with
weak ore,
or cribs, resting on filling, are required to support level-pillars, and
of ground as work progresses; such ground may be broken by ho
floor of level above. If the level is required for
haulage, etc, filling from outside sources is neces-
sary to replace ore in level-pillars.

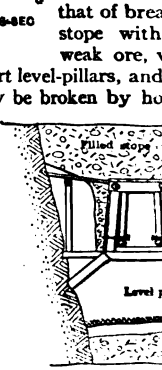


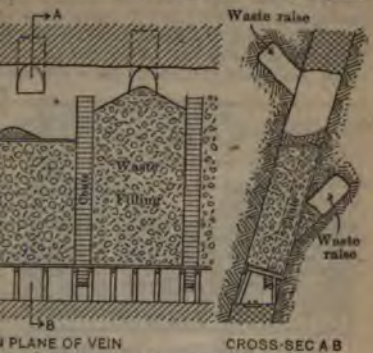
Fig 268. Mining Lev
Widths 15 to 20 ft,

Fig 267 shows method at WAIHI MINE, New Zealand, of mining pillars under timbered levels (233). Work begins by placing heavy stringers *A*, 15 to 20 ft long, on each side of track close to posts. 1.5-in bolts *B* are then driven into each post, their projecting ends resting on the stringer and supporting the posts when ground below is removed. Timbers *C* are wedged between bolts and caps; spreaders *D* are placed at each set. The underlying pillar is next broken down, each drift set laid bare being bl stulls *E*. When 3 or 4 sets have been thus picke space is filled and stringers are moved on another le ward end always resting on solid ground. This met vein widths up to 15 ft; for greater widths, stull too long to handle. Saddle-back sets (Fig 268) widths of 15 to 20 ft. Fig 269 shows method at NE MINE for supporting stull and post timbering; ot are feasible.

Fig 269. Support for Stull and Post Sets while Mining Level-pillars, Nev Wonder Mine (Fig 280)

Examples of practice given below illustrate details of filled flat-back o
stopes in narrow veins; see also Resuing, Art 61.

v; data from T. M. Smither (234). A fissure 6 ft wide. Ore is much fractured and crushed; alls in lower levels are weak and spall off in large e, level is timbered as in Fig 270, with 3-in plank



ed Flat-back Stope, Nevada Wonder Mine

ers being drilled with stope-drills. Before removles are drilled for the second, but are not charged. e chutes, the remainder being shoveled or handled d manways are built up and braced temporarily, 4 ft of back. Waste raises are driven alternately y 2 chutes; a raise serves 2 or 3 back-stopes and

from H. Rose, in 1916 (235). Orebodies are in one shoot was 3 000 ft long, 600 to 700 ft deep, r ton, chiefly in silver. Hanging wall is usually fractured, with slips parallel to veins, and requires careful support. Flat-back overhand stopes with contemporaneous filling are common when walls are too heavy to permit shrinkage stoping, but strong enough to hold until stope can be filled (Fig 271). Economic max level interval is 165 ft, due to difficulty of maintaining timbered chutes in higher lifts. Besides the drifts in the vein, counter-drifts are run in the firmer footwall ground; crosscuts, 165 ft apart, are driven from these to the vein, ween crosscuts. Drifts under stopes are not kept tes and manways are built up from ends of cross- e (Fig 271). Filling comes partly from develop-

ta Gertrudis Mine, Mex

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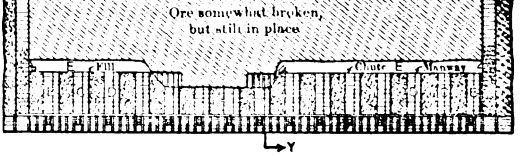
ment work; for excess filling, a central raise taps a surface quarry; this r
hard footwall, and receives or delivers waste at each level. Barrows distri
stopes, 0.67 ton being used per ton of ore extracted.

Butte, Mont; data from B. H. Dunshee (219). Weak ores are commo
filled square-set stopes (Art 45). Moderately wide veins, with strong ore s
from fault planes, are mined as in Fig 272; locally called "back-filling."
timbered with square-sets, with a raise to level above. First floor, mined
bering, has chutes 25 ft apart; top of sill-floor sets is covered with a shee
stulls, resting on timbers bl
ft above the sets. A back
started at the raise and adv
distance; miners return to
drills on broken ore and c
back-stope. This is repeate
of broken ore is 15 to 20 ft hi
stoping is begun at end of t
so formed and the broken o
into chutes. After cleanir
chute *A* (Fig 272), waste is
raise onto the sheeting; wh
longer distribute itself by
trammed from raise on a tra
filling about 6 ft below b
Filling follows in behind t

Fig 272. Back-filling at Butte, Mont

chutes being built up to level of filling track as toe of fill approaches them;
thus taken over whole length of stope. A floor of 2-in plank is then laid on t
another slice mined and filled. If ground gets heavy, square-setting can begi
on top of filling; square-sets are also used for mining level-pillars.

Filled stope in weak ore. J. E. Harding describes a method of mini
shattered ore, 300 ft long, 100 ft high, and about 20 ft wide, at MINA SAN
Mexico (236). An open square-set stope, 4 to 6 sets high, had caught fire and
stope was reopened several years later; ore had come down in huge slabs, a
portant to prevent surface subsidence. Square-setting was not deemed fea
of cost and scarcity of timber. Waste filling was obtainable cheaply fr



SEC IN PLANE OF VEIN CRO
Fig 273. Filled Flat-back Stope in Weak Ore

raise, connecting with a surface quarry. Fig 273 shows method adopted
driven on 200-ft level through the caved area, and timbered with sill-floor sq
posts 4 ft apart; little spiling was necessary. A chute and manway were st
4th set; a cribbed raise *A* was put up to the 100-ft level at each end of sto
on sill floor, at both ends of stope and both sides of drift, a room as high a
taken out across the vein; as fast as ore was removed, the back was caught
cal stulls and headboards. Filling, drawn through raises *A*, was kept clos
stowed close to back, the stulls being buried. Waste was trammed to adv
rooms through a small drift in the back directly over the sill-floor sets; car
from temporary gates built into raises *A*. When the sill-floor was mined a
cessive slices about 10 ft high were similarly mined above. In any slice
started at the end chutes *B*, and connected with waste raises *A*; filling was

s *C*; chutes *B* were then built up, waste drifts *E* l. Rooms were then opened from tops of chutes *C*, the process being repeated until slice was removed. n never exceeded 10 ft; less in bad ground. Stulls per slices were of straight-grained hard pine, 10 ft he and tapered to 6-in diam at one end; they were illed out of filling with a hydraulic jack when unis is a variant of the Crosscut method, Art 65.

also called "Stripping")

orking very narrow veins or paystreaks.

. *P* is a narrow high-grade streak, separated *AB* by gouge *C*. A flat-back overhand stope alongside of a 3; the broken xposed part of n and sent to to 50 ft apart.

Fig 274. Resuing

rebody are: (*a*) ort out as much duces more and e Art 60). (*b*) , 28), to include les of ore interng. Choice of d waste, which n stopes, and on treatment. Reher alternative.

application of f weakness, as aystreak. (*b*) n-grade ore is ts with profit.

is 10 in wide, dip 66°, diorite wall rocks, soft and art and timbered with stulls; chute interval, 50 ft. in hanging wall alongside of the vein. When a porgh, is exposed, canvas is laid on the filling and the vein is about sufficient to accommodate swell of e handling of waste is necessary (237).

ning a narrow, high-grade, gold-quartz vein; data oles, dip of vein is 85°, wall rocks contain no values, stoping can be done there without disturbing vein; ation, pumping, and general expense.

	Total
	$374.40
	37.44
	18.72
	144.00
	198.00
	$772.56
	749.00
	$ 23.56

Example A. Width of vein, 6 in; value, $48 per ton. Vein and wall rock are broken together in a stope 30 in wide; broken material contains 20% ore and 80% waste; aver value, $9.60 per ton. 5% of the waste is sorted in stope; 5% of 80 = 4, hence 96% of tonnage of ore and waste broken is sent to mill, with a value of $9.99 per ton. Mill recovery, 75%, or $7.49 per ton. Cost per 100 tons milled is shown by accompanying statement.

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Example B	Tons	Cost per ton	Total
Mining waste	500	$2.40	$1 200.00
Mining ore	100	3.60	360.00
Handling waste	110	0 48	52.80
Tramming ore	100	0.36	36.00
Hoisting waste	110	0.18	19.80
Hoisting ore	100	0.18	18.00
Milling ore	100	1.92	192.00
General	.....	.....	252.00
Total	.....	.....	$2 130.60
Value of gold from 100 tons			3 600.00
		Profit	$1 469.40

Example C	Tons	Cost per ton	Total
Mining waste	250	$2.40	$ 600.00
Mining ore	100	3.60	360.00
Handling waste	23	0.48	11.04
Handling ore	100	0.36	36 00
Hoisting ore	100	0.18	18.00
Hoisting waste	23	0.18	4.14
Milling ore	100	1.92	192.00
General	.....	.....	240.00
Total	.....	.....	$1 461.18
Value of gold from 100 tons			2 160.00
		Profit	$698.82

Example B. value of vein s ample A. Mi by resuing; a carried in foot vein broken do about 3 600 sq f some waste f stope had to b face. Mill re or $36 per ton shows cost p milled.

Example C 12-in vein; val ton. Mill rec $21.60 per t same as in Statement sh 100 tons mille

The same v a stope 30 in ore and waste gether, shows per 100 tons 5% of the w sorted in stop the waste can profit is $196.

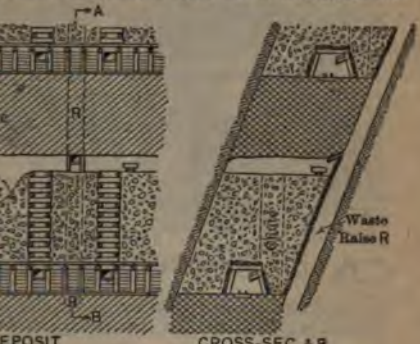
62. Filled Flat-back Stopes, Wide Orebodies

Field of use. Wide deposits with weak walls; ore should stand over back of stope or with no greater support than is afforded by p resting on filling. For use of filling methods in weak ores, see Art

General plan of work is the same as for narrow veins (Art 60). ences in detail are: Walls of wide deposits can not be used to supp protecting levels, hence haulageways are made by timbered drifts stope before beginning filling; these are called GANGWAYS in followin filled stopes (for dry-wall gangways, see Art 64). For safety, thicker must be left in wide than in narrow stopes; mining level-pillars unde stopes requires more elaborate methods than in narrow veins. Fill stopes may be obtained in part by sorting or by breaking into walls, b it comes from outside sources through raises from an upper lev Terms "back-stoping," "overhand stoping on waste," and "cut a used loosely in different districts to denote methods in filled flat-b Examples under different conditions are given below and in Art 63,

Minnesota Iron Co, Soudan, Minn (35, 153, 183, 239). Orebodies ar lenses of hematite, varying in length from 200 to 1 000 ft, in width from a pin ft, and in vertical height from 250 to 500 ft; dip, 65° to 75°. Ore is very har walls, weak and soft. Method of mining (Fig 275, 276). Overhand, flat are carried with alternate ore-breaking and filling; timbered gangways a through the filling; an occasional crib supports loose slabs in back of stope. several inclined shafts in footwall; level interval, 80 ft. Levels are opene wide crosscuts from shaft to hanging wall; a 7 by 8-ft drift is driven throu lens, followed by a breast stope (Art 30), which takes out full width of deposi of 15 to 20 ft. Gangways, following advance of breast stope, consist of 3- timber sets (Art 21), with 9-ft posts, 11-ft caps; the sets are 3 ft apart, clos held in place with sway-braces. Fig 276 shows plan of gangway. At chute

to 8-in diam, side lagging 3 to 5-in. Cribbed ladder-
anging-wall side of gangway at 50-ft intervals, and
wall side. When timbering is finished, stope is filled
there should be at least 5 ft of fill above gangway sets.
c of stope, the ore being thrown into chutes; as stope
ribbed and a new layer of filling put in, leaving a 6-ft
Successive back-stopes about 15 ft high are taken
y under the filling in stope above; level-pillars are



Stope, Minnesota Iron Co, Soudan, Minn

te for filling is admitted through 6 by 6-ft raises *R*
ng contact between rock and ore. Raises are about
cut at surface; the side exposed by stoping is tim-
pe; this allows filling to be drawn into stopes at any
ture service in lower lifts. Filling is distributed in
te gates erected in raises. For upper lifts, filling was
; the filling in exhausted upper lifts is drawn down-

ig 276. Plan of Part of Level, Minnesota Iron Co

to cave to allow use of this method, which resembles
es a pillar is supported on cribs built on top of last
ections, beginning at ends of the stope and retreating
ahead of the overlying waste, and can be recovered
ve ample warning of impending caves and men have
plicable to wide, steep-dipping orebodies, with weak
nsupported over backs of stope. Timber consumption
be reused; all ore must be moved by hand to chutes.

ria; data from P. B. Scotland (178, 152). Sulphide
ur as disseminated deposits in quartz porphyry and
shoots are irregular in outline; lengths, 450 to 1 750
ore extends to depths of 450 to 800 ft. Hard ore

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which stands well is mined in flat-back filled stopes (Fig 277); locally called
and filling. A slice, 15 to 20 ft high, is taken over whole area at bottom of

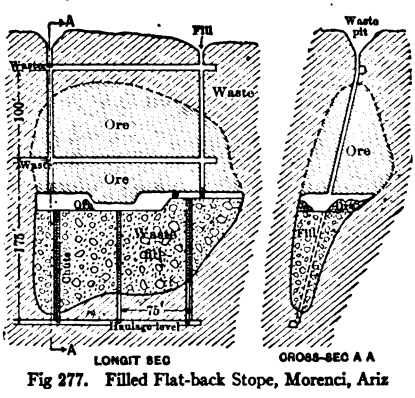


Fig 277. Filled Flat-back Stope, Morenci, Ariz

more raises are dri
or to level above f
and filling; chutes
about 75 ft apart
from tramming l
stope is filled to
back. If orebody
portant haulage l
opened 10 or 15
Back-stopes are 15
Filling is distribu
experiments with
scrapers were unsuc
cepting a few cribs
to support loose sl
timber required i
Several stopes in
were so wide (up t
pillars amounting
body were left to
and back. Danger
ported backs limits

to the firmest ground. Costs are low if filling can be obtained cheaply
quarries or glory-holes. (For other methods at Morenci, see Art 33, 46, 66,

63. Filled Flat-back Stopes, Wide Orebodies (at Broken Hil

Orebody. Lead-silver-zinc ores occur in a steeply dipping lode
fault plane in metamorphosed schists (242). Nine companies are
this lode, which is over 2 miles long and from a few ft to 350 ft wide.
from a dry sand in oxidized zone to hard compact sulphides below. W
require prompt support.

General plan of mining. For soft-ore methods, see Art 46, 65.
are mined in filled flat-back stopes; local term, "open stoping with bu

Development. Vertical shafts are usually sunk in footwall; max level i
ft, because of great wear on chutes and reduced accessibility of stopes wit
Levels follow lode where narrow; in wide portions, levels are driven in foot
lode is crosscut at intervals of 100 ft. For ventilation and handling filling, 2-
raises are put up from crosscuts to level above. Some wide parts of lode are
drifts in the ore, and permanent footwall haulage-drifts are not made unti
rock from them can be used conveniently for filling in next lower lift.

Stoping. The entire sill-floor is breast-stoped to a height of 16 ft. Timbe
are built and lagged over, chutes are started, the sill-floor is covered with
toms"), and filling is run in to a depth of 11 to 12 ft. Successive back-stope
high. Operations of stoping and filling alternate; usually a slice is complet
before one above is begun. Timber cribs on sill-floor and in back-stopes supp
See below for method of mining level-pillars and picking up stope bottoms. At PROPRIETARY mine, lode widths of 200 ft have been mined in 2 sections; a section somewhat more than 100 ft wide was stoped out and filled on footwall side, hanging-wall section being mined later in same way (240). Stoping begins at the crosscuts, the sill-floor being gradually

Fig 278. Stoping, South Mine,

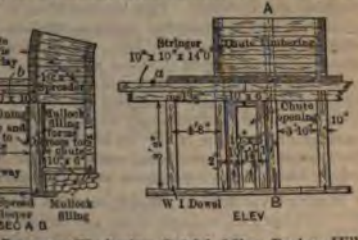
extended over whole width of lode. Some sill-floor stopes are over 100 ft w
long; max safe area depends on local conditions; open areas can often b
leaving patches of waste or low-grade ore as pillars. At SOUTH mine, a 9-f

d by a 7-ft slice (Fig 279). Advantages claimed to 12 months to make 1st cut in a large stope; t safe than a higher one. (*b*) With 16-ft backs,

Fig 279. Stoping, South Mine

rilled by 3.25-in piston drills on columns (Fig e spalling in stopes. Before blasting, 4 by 10- ng, to keep ore clean and provide a shoveling g rules enforced at SOUTH mine to render flat- ble: (*a*) 2 parties on a back-stope work away es very heavy if they approach each other. If ler the back between 2 parties, falls of ground le output from a stope. (*b*) Holes are always f breaking into walls and causing adjacent ore ited (Art 28). (*d*) Work under the low back d; falls of ground are most apt to occur there; erways under bellies as thoroughfares is forbid- back-stopes are never worked simultaneously 2 faces are far apart, and then only when a lad- sandy for the forward party; if men must travel ly to reach the ladderway, a gangway should be em protected by 2 rows of cribs, 6 ft apart. (*e*) oncentrated on bad ground to bring it down; s are supported by cribs. (*f*) Contracts for re exclude building of cribs, which are erected crews; this prevents miners from neglecting increased profit.

ıy timbering. Fig 280, 281 show different ch have proved adequate to carry weight of fill- PRIETARY mine sets (Fig 280) are spaced 4 ft Timbering at BRITISH mine (Fig 281) is designed pressures; the longitudinal sills are easy to pick by a false stringer *a*, leaving a convenient open lagging. With high lifts, extra posts are put y 10-in packing-pieces on back of original posts



Gangway Timbering, British Mine, Broken Hill

skin and lagged inside with 10 by 4-in hard- utes have been successful (Art 91).

ief points are illustrated by practice at SOUTH ed so that chutes will serve the stope above con-

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veniently, and so that winzes or raises will serve for handling filling to stopes. Proper connection should be made with main waste raises. (*c*) Ladderway offset from gangways and not be started on curves. (*d*) Two chutes and a la

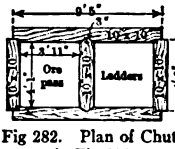

Fig 282. Plan of Chute in Fig 281

always placed near a raise, which serves as a starti stoping and for mining level-pillars. These state generally to all wide, filled flat-back stopes.

Sill-floor bottoms are of 10 by 4-in Oregon pine p 14 ft long, laid parallel to strike and close togethe floor outside of gangways. At SOUTH mine, bott on a 12-in layer of fine filling, because: (*a*) It is level the planking on a rough stope floor; the filling r ing cost, and the planks remain sounder. (*b*) Wh pillar below a bottom is mined, the layer of fines may be scraped out an inserted (Underhand square-setting, Art 45) before shooting out the ground. bottom reduces risk of runs of filling into the stope below.

Bulkheads or cribs are 5 to 7 ft square, of 10 by 10-in by 5 to 7-ft Oreg 279). Stope cribs are usually 12 ft apart, allowing use of 10 by 10-in or 10 by 6- 14 ft long, to catch up ground between; sometimes long "booms" are cant from top of a crib to support shelly ground. Cribs are not omitted in strong are spaced 15 to 20 ft apart without stringers; a special crew is kept busy o and wedging and blocking those in place. Cribs prevent small falls and g of loosening of large masses; they may rest upon planking, to distribute pre larger area of filling.

Recovery of timber. Gangway timbers and bottoms are lost. Methods ing crib timbers: (*a*) When filling approaches a crib, a new low crib is built c close to the old one; latter is then pulled down and the timbers reused. (*b*) 6-in timbers, 2 ft longer than crib timbers, are inserted in a bulkhead about 9 base at the time it is built; as a new layer of filling approaches, the projecting e timbers are supported on 4 10 by 6-in posts, and lower part of crib is removed. proceeds just ahead of the toe of the fill; the 4 posts are drawn out of the fill and chain when the next back-stope uncovers them. Method (*b*) has advantag remain in original position until fill is completed; in bad ground cribs are no

Filling. Residues from flotation plants, commonly used for filling, have p well in the stopes; at CENTRAL mine, broken rock filling ("mullock") comes fro Filling passes through raises to main level above stopes. At PROPRIETARY diam bore-holes, made by Calyx drills, have been employed for handling sand cost about 25% as much as a winze. Lateral transport to winzes serving stopes is in mine cars holding 2 500 lb; men or horses are used for tramming, depending on distance. Conveyers are used at several mines where much filling is handled; they are uneconomical if oreshoots are irregular and far apart. First layer of filling is about 12 ft deep, covering gangway timbers with 3 ft of material (Fig 279). Temporary chute gates are erected in stopes at bottom of winzes, and filling is distributed in side-tip cars, running on trestles of 2 by 1 which are subsequently withdrawn and reused. Operations of filling and layi bottoms proceed simultaneously, so that the layer of filling under bottoms may conveniently from toe of fill. In back-stopes, filling follows miners as closely the thickness of a layer being equal to height of slice taken; Fig 283 shows cedure, which leaves a 4-ft space under the back.

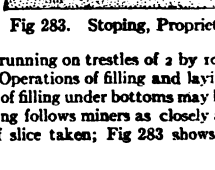

Fig 283. Stoping, Propriet

Mining level-pillars. Height to which flat-back stopes may safely be ca pends on their area, strength of ore, and on local conditions which determine wa In strong ore, entire lifts have been mined in places as flat-back stopes, the last mined in small sections, and stope bottoms above picked up on cribs. Usuall stoping is not considered safe when level-pillar above is less than 25 to 30 ft SOUTH mine, 30-ft level-pillars are mined by underhand square-setting (Art last back-stope is only high enough to allow a flat floor of square-sets to be ere of filling over entire area of stope (Fig 209, 210). These sets are blocked, an

. This leaves a level stope back and secures a uni-
the sets are a good foundation for timbers in under-
ites that
ground
worked
metimes
ts being
method
rs 40 ft
e filling
pported
k point
l above;
around

Fig 284. Mining Level-pillars, British Mine

hovel. A. J. Moore says this method presents little
-shaped piece of ore which is practically unsupported
verages 70 to 80 ft wide, with firm walls and compact
soft ground; compare with mining of level-pillars
ethods and further detail, see Bib (240, 241, 242).

Wide Orebodies (Baltic Dry-wall Method)

od, which was devised by F. W. Denton, are:
ntaining gangways and chutes through filling;
s; (*c*) devices for distributing sand filling in
ased on notes taken by author in 1910 and by
ta published by C. T. Rice in 1912 (141).

s the amygdaloidal copper deposits described in Art
tures; it is worked by Copper Range Consol Co for
c, Champion, and Trimountain mines, near Paines-
ft, averaging 24 ft; dip, 70° to 73°. Copper is in
-grade areas (170); about 25% of the ore (locally,
e sorted out as waste. In 1912, sorted ore from the
er ton. Ore is strong and stands unsupported over
ng, but usually tend to slab off, especially where cut
n make off into footwall. In all but a few places,
racter of ore and walls; the same factors, combined
use of open-stope methods (Art 38, 40).

afts, inclined at dip of lode, sunk in footwall or
ars of ore, 30 to 50 ft wide, are left on each side
e below under Stoping and filling). In shaft-
e 7 by 7 or 8 by 8 ft; elsewhere they are full
nlarge the drifts in ore to height of 14 to 16 ft.

ry-wall gangways; drawings from C. T. Rice
). Copper rock (ore) from drift and cutting-
stope is sorted in the level, waste being thrown
e. Building of gangways follows at a con-
ent distance behind the sorters; Fig 285 shows
form and dimensions.

ones for facing the walls are 14 to 30 in long; longer
s when available, or old timbers, are used as ties
een the faces. Face stones break joint, to avoid
es of weakness. Faces of walls next to drift are
ered 3 to 4 in in height of 7 ft. Rock of Baltic lode
s to break into tabular forms, convenient for facing
where necessary. Top of gangway is covered with
5-ft centers; pressure from wall-pieces is distributed
Fig 286 shows construction where lode is too narrow
lth is built along footwall, where heaviest pressure

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from filling occurs; hanging-wall ends of wall-pieces rest on a horiz timber in thickness, supported on posts under each wall-piece. Timber *A* is spike

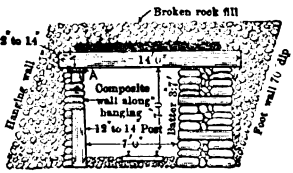

Fig 286. Gangway with One Composite Wall

a narrow stone wall is built betw space behind filled with waste. walls are adequate along hanging parts of lode and cost less than t Under wide stopes, a 4.5-ft or th quired to withstand side thrust of the minimum width of a strong 287 shows gangway construction waste is not available for walls sided cribs (pigsties) are built or filled with waste. Such walls sta when timbers rot; in these mine good and timbers last 6 to 7 years. Dry walls can be repaired by rebuildin under protection of a false set which supports ends of wall-pieces over secti

Dry-wall chutes (mills) are built at intervals of 30 to 70 ft; practice favors longer intervals and the use of cars in stopes for handling ore.

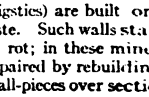

Fig 287. Cribb

Construction (Fig 288). The larger stones are used in inner walls to withstand wear of falling ore; outer wall takes most of thrust of filling; central core of loose stones allows outer wall to adjust itself somewhat to pressure, without disturbing inner wall. Walls are started from timbers, which are protected from wear by placing rocks above to overhang about 4 in. Chutes are built in sections 5 ft high; walls of a new section are started 4.5 ft thick at level of the stope filling and taper to 3 Mouth of chute is about 4.33 ft wide by 3 ft high in the clear; timber *F*, a built into the gangway wall and serves as a lip-piece for fastening chute chutes are easiest to build and maintain; where lode is narrow, some are v

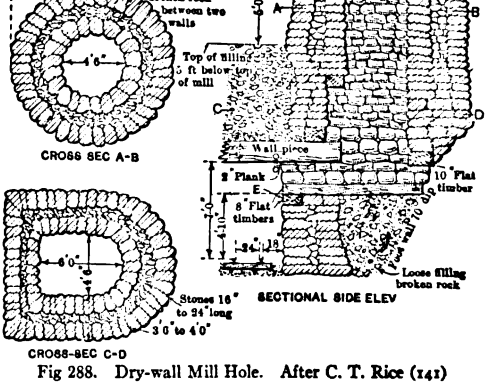

Fig 288. Dry-wall Mill Hole. After C. T. Rice (141)

feet abo are then allel to t clined ch where p

Data at BALTI are give (141). (Fig 285 men, at 30 sq f per 9 hr; largely of suita wages at two 4.5-f $2.50 pe build ab per 9 hr. of sorted in 1910. ing gang cost 4.97

bering, 3.2¢ per ton of ore milled. Parts of SUDBURY NICKEL MINES, Ont, methods similar to the above (243). A. E. Hall gives following data Dry-wall gangways are built at CREIGHTON mine, under shrinkage stope mensions are about as in Fig 286; courses of round timber, 8 to 12 in diam walls as bonds at vertical intervals of 2 ft; timbers in each course are 2 ft

ace; company furnishes stone and timber; usually
ere suitable stone is not available nearby; some-
ad of waste. A crew, averaging 4.94 wallers and
faces of which totaled 1 444 sq ft in 26 working
er day; max, 37 sq ft; minimum, 14 sq ft. Same
t. Another crew, averaging 2.59 wallers and 1.93
ller-day in 26 working days; max, 52 sq ft; mini-
for walling, 2¢ per sq ft of face.

conditions compares favorably with that of
of its greater strength and life, and conse-
ts for gangways and chutes. An incidental
less timber has to be lowered into the mine,
hoisting ore.

in detail with local conditions. Usually the
enough waste to fill spaces behind dry walls
of shortage, the space on hanging-wall side

Starting Stopes, Baltic Method; Longit Sec

work the ore through lagging into the level; waste
feasible, lagging is covered with waste before stop-
ck-stope, chutes are built up and the stope filled
). Interval between chutes must be adjusted to
unsafe wall areas are not exposed between the fill
oes not usually furnish enough waste for filling;
from above, by shooting out footwall, by driving

NE OF VEIN
ing and Filling, Baltic Method

ng (see below). Fig 290 shows second back-stope;
is thrown back and aside; in higher back-stopes,
? (Fig 292). As stope face advances, mills serve
Successive back-stopes are carried to a height of
d economy of timber lead to adoption of method
el-pillar.
le requires operation of a large number of stopes
stopes advance slowly; removal of level-pillar
tions, combined with practice of mining outward

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from shafts, often require a level-pillar to stand untouched from 3 to 8 years time ground becomes heavy, and unsuitable for flat-back stoping. It is pro Baltic method as a retreating system, to shorten time before level-pillars ar a larger level interval might be possible, as well-built stone chutes stand wea for long periods. Width and dip of lode are also related factors influencing as chutes should be vertical, if possible (Fig 292).

Sorting is done carefully by special men; rock showing even a sp is sent to chutes. Large boulders are broken and the pieces exam decide whether fine rock, requiring shoveling, shall be classed as Few fines are produced in breaking, hence loss of fines in filling is s of fine waste, placed on top of the rock fills in stopes, eases work of sh may be recovered by skimming off an inch or so of this when the sto up. (See Mining level-pillars, below.)

Breaking ground in flat-back stopes is done with breast holes (Fig 289, 290).

C. T. Rice states that, at Champion mine, max burden on a hole is 22 in; tical row of 2 flat holes and 3 water holes to break a face 9 ft high. At Trim 1 flat hole and 2 water holes break faces 6 ft high; the same round often face at Baltic mine (1141). Holes are spaced about 5 ft apart across loc charged with 5 to 7 sticks of powder; 30% dynamite is used in company w tors prefer 40%. In 1912, drilling was done with 2-man piston drills (Ex 6, placed later by 1-man drills; in 1916, most drilling was done with Jackha columns. Distance between back and top of filling determines HEIGHT OF taken; enough muck must be made to afford a place to set up drills; for t between these factors see Fig 290; if filling is low, a high pile of broken ore ma by working 2 back-stopes simultaneously, with their faces close together.

Mining level-pillars (locally, "caving pillars"). Work begins the boundary, or at a point midway between shafts, and retreats to

A rill stope (Art 36, 66) is started from bottom of the raise; cuts are inclin repose of filling, which is 40°; miners work on waste, which is dumped th

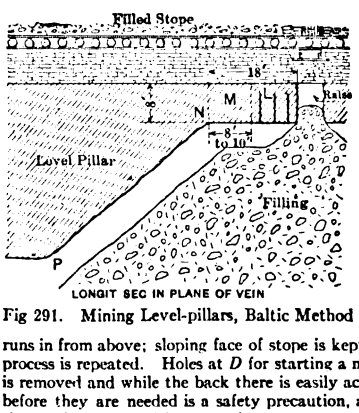

Fig 291. Mining Level-pillars, Baltic Method

fast as ore is sorted and rem is enlarged to dimensions all broken ore is then cleanc tion *L* of the 8-ft pillar rer the level is shot down from This allows dry-walls and stope above to drop; proje of the 8-ft pillar keeps op space 6 or 7 ft high alc Rest of level-pillar is mined Stope-drills are used, putt half-uppers, 6.5 to 7 ft d shift. Usual crew, 2 mac 2 sorters. After round at holes *C* are drilled. Then in *G* is sorted out and rem may take 1 or 2 weeks in cl stope. Next, round *B* is b runs in from above; sloping face of stope is kept open by the stub of pillar process is repeated. Holes at *D* for starting a new slice may be drilled befo is removed and while the back there is easily accessible. Practice of drillin before they are needed is a safety precaution, as the stub must stand for a time and the ground becomes dangerous to work under. In narrow parts strong ground, rill stopes for mining level-pillars may be carried with stepp 36). Ground is broken with breast holes, which are more efficient than u lode, but barring down takes longer, and this method is not so safe as that

Sorters work at *E*, and along face of pile of broken ore; waste rock ma sides of stope or dumped at *F*. Two chutes must be kept open, one for or

l is built close to the shaft-pillar and another
e latter, stoping is stopped and the remaining
n stub ends of pillars *B* and *C* (Fig 292) mixes
n they are shot down; about 50% of this ore
vard in mining level-pillars is replenished by
stopes at surface or on an upper level. Filling
evels; hence, at intervals, levels must be kept
s below. The pillars under these levels are

; smaller heights do not take full advantage
her, the back may become bad before the top

ltic Method (diagrammatic sections)

ork broken ore down the longer slope of the
ed low-grade ore occur over a level-pillar dur-
uch areas to tap the filling in stopes above.

nd, is hauled back to the mine in 40-ton
lling required in flat-back parts of stopes.
l at surface into a raise, roughly parallel to
n level. Sand is drawn at any desired level
ump, hopper cars, moved by electric haulage.
nnected to stopes below, at intervals of about
h which the sand is dumped. SAND-BLOWING
ute the sand laterally in the stopes (Fig 293,
mounted on a light angle-iron frame *B*, is set
Bottom of raise is closed by a sollar, to which
has a counterweighted gate *D*, which makes
k terminates at bottom in a tee *F*, connected
ompleted fill and moved or extended as neces-
n allowing the pipe line to be deflected slightly
with the air line supplying the stoping drills;
ower jet *J*. OPERATION: A tankful of sand is
essed air is turned on, forcing gate *D* tight
tank under pressure. Under combined effect
d is forced through pipe line *K*. When tank

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is empty, air is shut off and a new charge of sand drawn in. By shifting the pip end of a new layer of filling is built up across full width of stope.

Tank is operated by 1 man; it works successfully with pipe lines 100 ft long air pressure, some sand is thrown about 20 ft beyond end of a 70-ft pipe line.

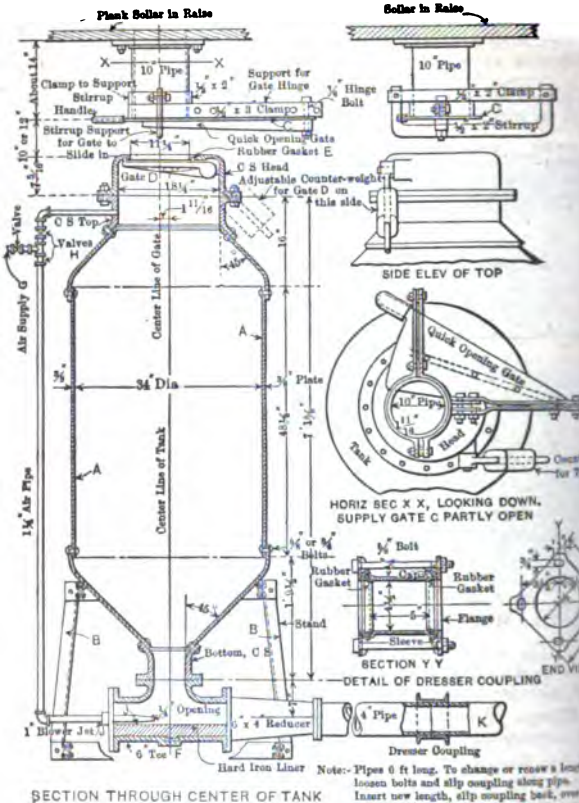

Fig 293. Sand-blowing Tank, Champion Mine, Mich (F. W. Denton)

tions on 15 runs (with a 28-cu ft tank, air pressure about 70 lb and 65 ft of pi following time results: for filling tank, 20 to 35 sec; for adjusting valves, 20 t for blowing sand, 1.3 to 1.5 min. No data on aver work over long periods are a Speed of work varies with percentage of moisture in sand and with length of pipe. pipes is 4 to 6 mo; wear is greatest with dry sand; pipes are turned frequentl tribute wear and obtain max service.

Sand-filling, flushed by water, is frequently employed in coal mining (Art 11

65. Crosscut Method

Crosscut method is used for wide veins or masses, with weak walls an ore which will stand unsupported only over small openings.

n horiz slices, in ascending order and in small before the next is begun.

ketch, Fig 294) consists of footwall drifts *D*, cross- ct drifts at intervals of 40 to 60 ft. A slice is re- crosscuts d through cuts *C* are re mined ack of *D*, om which ling cross- kept open g ore from wall drifts slice they r slices is p through l slices in simulta- slices and w.

crosscuts all cars; eled into

re contig- of 3 as in ire imme- n a cross- one (245).

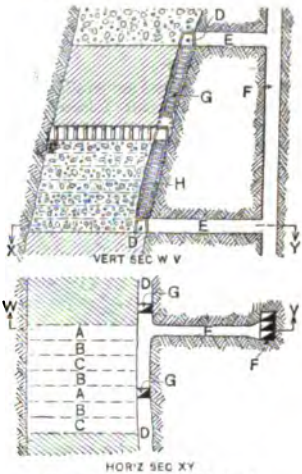

Fig 294. Crosscut Method

d 6 to 10 ft wide, or as wide as strength of ore e timbered with light sets, or stulls and head prevent runs of filling into adjacent workings.

OPRIETARY MINE, Broken Hill, N S W, where this y 10-in struts *S*, running along the sides of cross- osts; the back is carried by lagging on cross-pieces *D*; diagonals *E* are put in where required by pres- sure. Sets are about 6 ft apart; their size depends on character of ground and length of timber avail- able. About 67 % of the timber used is recovered on opening a new slice. The tapered shape of posts and the hole *H* (for inserting a piece of drill steel) aid in withdrawing them; or a chain and lever may be used (240).

Breaking ground. First openings in lowest slice of a lift are driven like drifts (Art 21); adjacent crosscuts cost less, as ore is free on at least one side. he ore is always free on the bottom. At CABEZAS t price for mining the lowest slice averaged 50¢ per

verlying filled stopes requires care. Plank or crosscuts in bottom slice, to prevent runs of below; sometimes filling becomes so compact sscuts may be spiled under loose filling.

m is usually 65 to 70 ft (246). Filling can to crosscuts, and each layer settles somewhat

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under weight of the back. Successive subsidences cause ore abo and increase difficulty and cost of mining upper slices; this effect in height of lift and weakness of ore.

At CHAPIN MINE, Mich, where this method was formerly used, settlemen a 100-ft lift are said to have occurred (247). At CABEZAS DEL PASTO MIN filling was coarse quartzite, quarried on surface, and so firm that no subside was noticed (248). At some places subsidence occurs because the top slic incompletely filled.

Several variations are possible. Haulage drifts *D* (Fig 294) need not be but may be in any part of deposit or in wall rock. Crosscuts *A*, *B*, *C*, m desired direction through deposit; large horses of waste may be left in pl shows application to an irregular massive deposit, opened by shaft *S*. Usu on a level is to outline orebody with *A*; then haulage drifts and crossc placed in best position; one arrange cuts for slicing is indicated by dott least one main waste raise *R* is d further detail and other variations, 245–249).

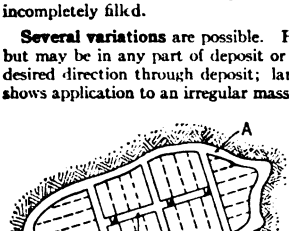

Fig 296. Crosscut Method Applied to Massive Orebody; Plan of Level

Summary. Crosscut method all complete extraction of large, weak almost any shape, without serious s ence. Cost of development is high, small level interval and frequent nece ing haulageways and waste raises in Cost of shoveling large amounts of is unfavorable to use of this method in the U S. Timber consumption will stand over back of a crosscut until it is filled; even in weaker ore, used than in top-slicing (Art 71–75) or filled square-sets (Art 46), which a methods affording complete extraction; caving methods are preferable if sur be supported. Crosscut method is sometimes advantageous for mining l bad ground over filled flat-back stopes (see Minn Iron Co, Art 62).

66. Filled Rill Stopes

General data are given in Art 36 (Fig 159, 160). Filling is ru stope through the raise from which it was opened, which enters the highest point. Filling is distributed largely by gravity. By i stope backs at the angle of repose of filling, the surface of the fi parallel to back of stope (Fig 297), and therefore convenient for sup and walls; a sloping surface of fill also aids in moving broken ore t

In narrow stopes, little or no shoveling may be required for handling fillir wide stopes, filling builds up under the raise in form of a cone and must be sho of stope; similarly, broken ore must be shoveled from sides of wide stop See Calumet & Ariz Co and Mt Lyell mine, under examples below, for mod shoveling in wide stopes. For field of use, see below; also under Compari back stopes, Art 67. Term "cut and fill" is used in several districts of U mining in filled rill stopes.

Examples of practice (for further data on breaking ground, see

Kalgoorlie, Australia (251, 257, 258, 146). Filled rill stopes are used in like deposits of gold telluride ores, occurring as lenticular shoots of quartz country rock, chiefly in amphibolite schist. Method is not applicable o than 35°; usual dips are steep. DEVELOPMENT. Ideal plan is to run level connected by winzes *W* (Fig 298) at intervals of 150 to 200 ft; winzes are sun wall to minimize handling of filling. A flat cutting-out stope, 7 ft high, is t are timbered with stulls for widths to 14 ft; for greater widths, use saddle-ba 36). Stulls are 5 ft apart, and lagged with 1-in poles covered with old filte of cyanide cases, etc, to retain fine filling. Chutes are started about 50 ft ap

arter points *M* (Fig 297). STOPING AND FILLING about 37°, and rill cut is generally adopted (Fig first opened at foot of winzes *W*. Broken ore is made by taking up 1 or 2 lagging poles, and waste above.

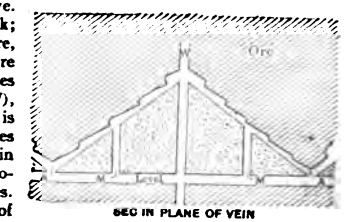

Fig 297. Starting Rill Stopes, Kalgoorlie

thick; n ore, *M* are stopes 297), ore is hutes 7-in pro- utes. ne of nning used

sture content is less than 25%; if wetter it tends used to distribute filling, as at *C* (Fig 298); then only one winze needs to reach the surface. Even with winzes on the hanging wall, some filling must always be shoveled in stopes to edge of rill. In rich ore, plank floors are laid on faces of fill to avoid loss of fines and help ore to slide to chutes; old filter cloths are sometimes laid under the floors. Floors are usually omitted, involving the removal of a few inches of filling with the ore from each slice (83). Stopes near top of lift can not be filled through original winzes *W*; auxiliary raises *R* (Fig 298) are put up to handle filling.

Superior & Boston mine, Globe, Ariz; data from R. L. Herrick in 1910 (252). In the following example, width of vein is 7 to 15 ft, averaging 9.5 ft; dip, 58°; footwall is hard and smooth; hanging wall, weak and ill-defined. Levels are 100 ft apart vertically, connected every 100 ft by raises on footwall. Raises are divided by stulls into a 4-ft chute are timbered by heavily lagged sets, 5 ft apart. between back of ore and top of lagging; when 15 ft centers. s; chutes are ndoned when like that at ct, all broken ises, where a ing. Chutes falling ore. stope-drills. oting for men etween top of pe, 5 to 7 ft. a new slice is o cross-pieces s finished and ons are taken y for relaying

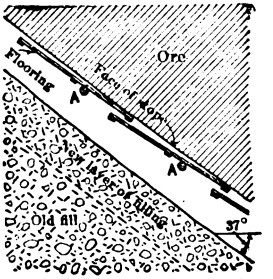

Fig 299. Sec in Plane of Vein

Access to stopes is chiefly through manway com- of access from below is not objectionable. After

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stope faces break through to level above, filling is dumped in from the drift cost is about ¼ that of square-setting, formerly employed.

Arizona Copper Co, Morenci, Ariz. P. B. Scotland (178) states that this uses rill stopes, taking 10 to 15-ft slices, in narrow veins where filling is needed is eliminated, but output from a stope is necessarily intermittent.

Calumet & Ariz Copper Co, Bisbee, Ariz; data from P. D. Wilson (2 Dickson (250). Massive sulphide replacement deposits in limestone are ingenious form of filled ril called GILMAN CUT AND This may be used only walls are strong. Orebodi and were formerly mined age stopes, which became u areas increased. GENER 300–304). Rill stopes, 4 carried one at a time acro filled; their length depen deposit, and they extend tween levels or from a leve commercial ore above. Su may be contiguous, or sep pillars, mined later in sa

Fig 300. Gilman Cut and Fill, Plan of Main Level

tinctive feature of the method is that the floor of each stope is trough-s broken ore slides without shoveling to chute gates under middle of stope. Fig 300 shows typical lateral development; level interval is 100 ft; main haulage drift *A*, from which crosscuts *B* are driven, is preferably in wall rock. Raises connect with haulageways on level above; work in raises through country rock is delayed until the waste can be used for filling. STOPING AND FILLING begin at a raise; back of crosscut is shot down and widened as at *C*, Fig 303. Crosscuts are timbered with temporary sets (Fig 301) as soon as the back has been cut out to a height of 8 or 9 ft above rails. The sets are 5 ft apart and floored with 4 by 10-in plank, except for a 12-in space over track, which is covered with 2-ft pieces of 2 by 10-in placed crosswise. A piece of 2 by 6-in plank, spiked to the flooring on each side of the central opening, holds the 2-ft pieces in position. The temporary sets reduce shoveling in early stages of the work. Though only 5 ft high, the sets admit a 30 cu ft-car, for receiving ore falling through the o which is uncovered as required by prying up the short planks. Cutting-o

Fig 301. Gilman Cut porary Crosscut

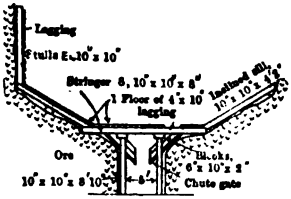

Fig 302. Gilman Cut and Fill; Stringer Sets for Crosscuts

widened to 40 ft; floor of stope o crosscut rises at an angle of about being kept arched. As space is obt porary sets are replaced by stringers at 5-ft centers, and chute gates ar side of crosscut between alternate When cutting-out stope is complet 10 to 12 ft above stringers *S* (see al Filling is then run through the raise above, until it forms a cone reaching shoveled to sides of stope, into a wed (Fig 304). Each face of the pile i 2-in plank, resting on 2 by 10-in sill filling; these floors form slides slopi repose of filling (about 40°). Rill cuts, parallel to slides, are then started raise; when completed, the slides are taken up, waste is run in and trimmed to a wedge-shaped pile, new slides are built, and another inclined slice is take All slices are 10 ft thick, which keeps the back close enough to the slides t

g 304). Slices are mined with stope-drills, from *bot-* e between face of a new fill and back of stope must be nstruction of slides. Side walls of stope, as exposed, o which 2-in lagging is spiked, to hold the filling when or on bottom holds filling when a stope comes up from ute gates, thus minimizing shoveling where stope face end of stope (*G*, Fig 304); cribbed chutes must be car- of slides. Chutes are also required where stoping is see middle of Fig 304), considerable shoveling being ugh-shaped floor soon loses its value, and a flat floor up from below. Author.) To prevent clogging by cted by a grizzly of two 8 by 8-in timbers. When raise, say 50 ft away, is driven to level above and rations; outlying portions of stopes have been filled top of orebody shoveling is necessary if stope is to ying rock is picked up by timber cribs (Fig 303). t cost of mining at Bisbee by Gilman system aver- 4¢; explosives, 4¢; timber, 7¢; candles or carbide, e is the cheapest mining done in the district (Table rative costs of square-setting and Mitchell slicing

300 Fig 304. Vert Sec *VV*, Fig 300

em costs slightly more than shrinkage stoping, but f its comparative safety. Under normal conditions, daily output of 1 stope averages 100 tons and may e Bib (216) for rill stoping at Copper Queen mine. ill stopes may be varied to suit shape and size of ng cheap and safe, timber consumption low, ground of ore and filling is minimized. The method re- acks with practically no support; it is not adapted

es are auriferous quartz veins in diorite; ore, moder- erous, thus requiring filling; shoots are 100 to 400 ; assay, about $15 gold. Rill stopes were adopted nd filling, and because the fill could be kept close to osit opened by an adit, 20 ft in footwall and 200 to rebody were 65 ft apart, with raises from alternate e rock could be obtained cheaply. Vein was breast- and ore broken back around bottoms of raises as far noving broken ore, and running in filling, alternate from first few cuts rolled to floor of stope and was opes were connected, ore was passed through chutes kept open under the filling raises, to serve the lower perintendence, for stoping, filling, and tramming was n labor, 50¢; miners, according to nationality, 60¢

R. M. Murray in 1915 (254). Orebody is a huge elliptical in horiz section; ore, strong and homo-

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geneous; conglomerate or schist footwall is strong and regular, schist
irregular and weak; general dip, 64°, flattening in depth. Orebody was w
and mining method could be planned completely in advance. GENERAL PL
(Fig 306). The stopes are large rooms, separated by 25-ft pillars in rows
apart. Sill-floor of a room is breast-stoped 12 ft high; a raise near middle
nects with a waste ("mullock") pass on level above. Sill-floor gangways

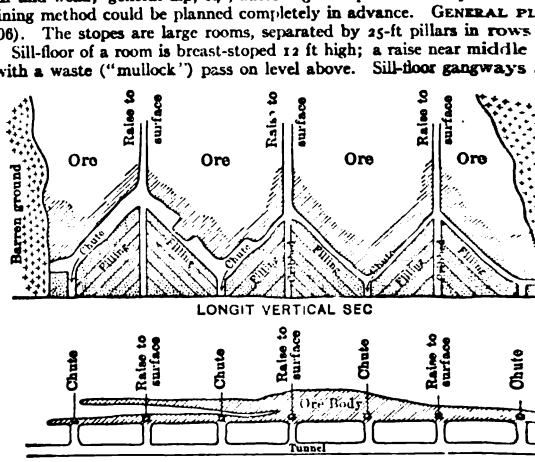

Fig 305. Rill Stopes, Zaruma, Ecuador

with square-sets, resting on 8 to 10-in round logs, 15 ft long; lagging of the s
with old sacks to keep out fines from the filling. Usually, there is 1 main ga
shaft and several cross gangways, 21 ft apart, extending to walls or pillars.
between gangways is covered with heavy lagging. As much filling as poss
through the waste raise, the bottom of which has a bell mouth to allow filli
Stoping begins at the raise, working outward and downward in all directi

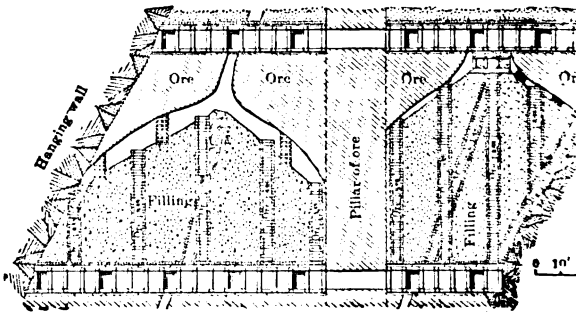

Fig 306. Rill or Domed Stope, Mt Lyell Mine, Tasmania

mouth being enlarged to a dome-shaped chamber. More waste is then run
further portions of sill-floor and gangways sets, and forming a cone below rai
to resume stoping. Finally, a dome-shaped back is formed over whole a
Cribbed ORE-PASSES, 4 by 6 ft, are built up from gangways about 15 ft apa
spacing minimizes handling. Ore is shoveled or pushed into nearest ch
made to gravitate to bottom of sloping fill, as usual in rill stopes. As the st
in height, some passes at the middle are abandoned. Prior to each filling

inst back of stope, their downhill sides being braced
es in middle of stope and next to pillars are vertical;
ip. A special chute under the waste raise is used
elow. A 3 by 3-ft manway is built at each corner
from an open cut, is led into the pass with a stream
onsistency of wet concrete; its flow is controlled by
about 75% of the waste required to fill it is run in
equired in stope. The wet filling runs to remote
little shoveling. Middle part of stope is filled to
anging wall and near pillars it is completely filled.
icking to break it, the sides of stopes are not lagged;
lars are mined. Towards end of a filling operation,
and form a smooth surface on which to break down
ll stope is ready for building up passes and running
filled and is ready for stoping. No data are avail-
pillars. The stoppage of production from a stope
y of output, but is remedied by working stopes in
e filling in the other. STOPE-BOTTOMS ARE PICKED
stope below (Fig 306). Filling will flow through
a radius of 30 ft, beyond which it must be trammed
urs in levels thus picked up, but it rarely exceeds
OST of stoping, filling and delivering ore to surface
ton; 25 tons are broken per drill shift (3-in piston
, 10.5 tons. ADVANTAGES claimed are: (*a*) dome-
; (*b*) both ore and filling are handled economically;
inspection or barring down; (*d*) small timber con-
a form of Chamber working (Art 42). For further

ata from C. P. C. Sullivan (255). A form of filled
d-zinc ore in schist; ore and hanging wall are weak
ping width, 6.5 ft; dip, about 80°. Levels, 98 ft
t raises at intervals of 98 ft. In Fig 307, triangular
by driving 3 superposed drifts *A*, *B*, and *C*, from
e full width of stope, are timbered with close-lagged

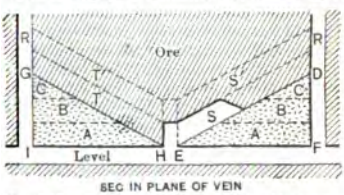

Fig 307. Rill Stope in Weak Ore, Calamon Mine, Spain

ing
red
ve.
ich
A
and
ght
off
ing
are
S is
fill-
t of
built up after each pair of slices *S*, *T*, etc, have been
n the floors of inclined slices with hoes. Filling for
ibuted largely by hand. Raises *R* are abandoned as
gates are built in them, as required, for distributing
ater; filling is waste from old workings or mill tailing.
driving and filling horiz superposed drifts between
s, Art 58). The present method reduced timber con-
tput per man-day and reduced total costs about 35%.

mary, Filled Stopes

lling methods (Art 59, 60, 66), essentially for
ls, have a wide and growing range of applica-
oo great for timber, and is the only sure means

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of permanent support of overlying surface or higher orebodies. Where dence is allowable, caving methods (Art 71) are usually cheaper, but ited to certain types of orebody (Art 83). Filling is also employed for or containing much waste that can be sorted in the stopes; waste can be support, and cost of hoisting it avoided. Where timber is costly, fille may be cheapest, and are preferred to timbered stopes even if latter are ad

Other factors governing use of filled stopes: STEEP DIP is desirable; w than 35° to 40° the cost of delivering and distributing filling increases Thick, flat orebodies, minable in rooms of considerable height, are exceptions to this statement (see Calumet & Ariz mine, Art 66). Fin can be handled cheaply on flat dips in a stream of water, but in gene device is better suited to delayed than to contemporaneous filling (s filling, Art 64; note the use of water at Mt Lyell and Calamon mines, Contemporaneous filling is best applied in STRONG ORE, where the ba stand safely, or require only temporary stulls or cribs (Broken Hill Filling is less advantageous in WEAK ORE, because excavations must b much auxiliary timber is usually required, and the close filling often n involves much shoveling. Filled square-set is the type of filled stope in U S for weak ores (Art 46); the crosscut method is an alternative (se for its limitations).

Advantages of filling: (*a*) Walls of stopes are supported permanently, wi danger of "creeps" (Art 55). Filling will settle but little if the material is st carefully placed; fine stuff like mill tailing packs well; weak rock carelessly pla settle as much as 50% under pressure. (*b*) Use of filling avoids risk of fire, alway in timbered stopes. (*c*) When properly applied, filling allows practically extraction of an orebody. (*d*) Ventilation is usually good, through the numero and raises; air currents must travel near working faces and can not short circuit, a stopes. (*e*) Waste from development can be used for filling, thus saving cost of h

Disadvantages of filling: (*a*) Cost of obtaining and handling filling, whic be compared with that of alternative methods of support. (*b*) Cost of gangw chutes. Maintenance of chutes is often difficult; its cost increases with heig and largely determines the maximum economic level interval. (*c*) Some fines when broken ore drops on top of filling. Loss is important where ore is rich a occur in brittle minerals, is greater in coarse than in fine filling, and recurs with layer of filling; hence, total loss of fines in a lift depends on height of stope sli between filling operations. (For prevention or reduction of this loss, see Ex practice, Art 60, 61, and 66; also under Floors, Art 47).

Filled flat-back *vs* filled rill stopes (20, 83, 146, 257, 258, 256). Flat-back s commonest. Factors influencing choice: ORE AND RILLING ARE HANDLED mor in rill stopes, as gravity assists their movement; flat-back requires more chutes a twice as many filling raises (20). Wide rill stopes lose much of this advantage, must be distributed to, and ore shoveled from, sides of stope. SORTING may b flat-back, but not in rill stopes. DELAYS in handling and setting up drills, in in timber and supplies, and time lost by men in reaching working places, are le back than in rill stopes, because the level surface of fill in the former facilitates m of men. Drills must be removed from a rill stope before blasting, but can ge placed in safety in a flat-back stope. SUPPORT. Stulls and cribs may be pla cheaply and efficiently in flat-back stopes; the back may be arched if desired rill stope tends to make its back self-supporting, even if orebodies contain horiz planes of weakness. Fairly firm walls are readily supported in either type walls require close filling, involving much shoveling in flat-back stopes; rill stope filled tightly by gravity. BREAKING GROUND (Art 26–28) is usually underha stopes (compare Sup & Boston mine, Art 66); it must be overhand for close fill stopes. FLEXIBILITY is greater in flat-back stopes; area of stope may be ex any horizon; prospecting openings may be run into walls where desired and produced used for filling; horses of waste or low-grade ore may be left unmined stopes, faces must advance uniformly and excess waste can not be stored. Out a rill stope usually ceases while filling; in a flat-back stope, mining may often p

TY. Principal danger in both types is from falls of utions in flat-back stopes). The sloping fill in rill dged from walls or back (or from surface of fill) near e and injure men; this risk is increased where fill is and CONTAMINATION OF ORE with waste are generally ore fines are lost when ore rolls or slides to chutes in in flat-back stopes. Rill stopes are filled at mini- nds at its angle of repose; but, as ore blasted upon ay cause much filling to roll down into the chutes, l with plank if ore is to be kept clean, or if a large

best employed in steep-dipping veins, less than strong and of uniform character, walls fairly fined. Under other conditions, the advantages weigh their higher cost for handling ore and ce of the factors discussed above varies locally.

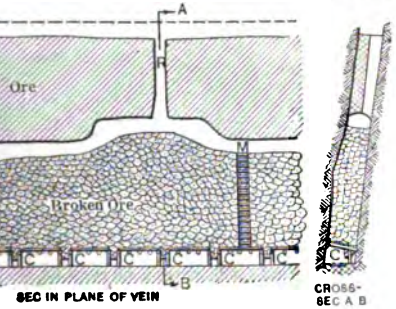
KAGE STOPES

68. General

are overhand stopes in which the broken ore completed. As rock increases in bulk when ore in a shrinkage stope must be drawn out as orking space under the back. The remaining s temporary support to walls; it is drawn when y be left empty and allowed to cave, or may be re is called SHRINKAGE STOPING WITH DELAYED -STOPING and OVERHAND STOPING ON ORE are note shrinkage stoping; other names are "lay e mining" (Swedish).

opes is in steep-dipping deposits of strong ore s of method, see Art 70.

ows practically complete extraction, though level- ed to prevent premature caves. Masses or wide



ig 308. Shrinkage Stope, Narrow Vein

rinkage stopes, leaving pillars to be mined subsequently ne, Art 69; also under Combined methods, Art 86–88).

vs a stope in a narrow vein, opened from raise (Art 36). Ore from the cutting-out stope falls

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to the drift; that from succeeding back-stopes collects on top of le
the excess due to swell being withdrawn through chute-gates *C* (Art
tilation and access to stope are afforded by raise *R* and timbered r
Procedure in wide stopes is similar (see below, and Art 69).

Chutes for shrinkage stopes are sometimes built up at regular intervals
waste-filled stopes (Art 60); they are then used for passing excess ore to the
fill of broken ore in a stope is stable, a weak back may be supported on cribs
Hill (Art 63), but this practice involves much shoveling, which is obviat
shrinkage stopes (Fig 308). Chutes may also be justified by the presence
ore, which can be sorted in stopes and kept separate for special treatment
below). Chutes are sometimes used to handle small patches of waste,
otherwise mix with the ore. Shrinkage stopes with chutes are relatively
for examples, see El Tigre mine (Art 69) and Bib (199).

Development for shrinkage stopes in narrow veins follows gener
down in Art 15. Level interval may be greater than in filled stop
because there are no chutes to maintain. Raises may be far apart,
being opened from a single raise in the middle, or from a raise
Blind oreshoots may even be mined without any raise to level above
and access being obtained through timbered manways at ends of
velopment practice in wide orebodies is largely a question of arrang
for drawing ore economically from the stopes (examples, Art 69).

Shape of stope-face. Both wide and narrow shrinkage stopes
have a flat back, under which, in drawing excess broken ore, it is eas
tain a working space of uniform height than where the back is inclin
lar. The advantages of a level working floor (Art 67) also apply her
shows a stepped-face shrinkage stope.

The shrinkage principle may also be applied in rill stopes of moderate
allow use of down holes, which alone should determine the angle of rill. St
which broken ore will roll, are a pronounced disadvantage in shrinkage stop

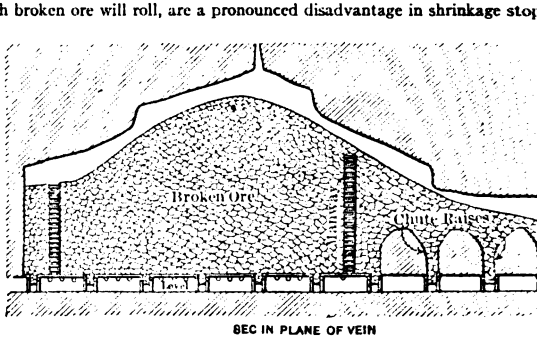

Fig 309. Shrinkage Stope, Lake View Consols Mine, Kalgoorlie, Au

from O. B. Ward (260), shows a rill stope for steep dipping oreshoots up
(for ore occurrence in this mine, see Art 66). Level interval is 200 ft; incli
back is just sufficient (say 18° to 25°) for holes at top of stope to carry wat

Breaking ground. Practice is the same as in other overhand stopes (
note Fig 127, and accompanying text, also remarks on blockholing, Art
Copper Co, Art 69).

Arrangements for protecting levels and drawing ore unde
stopes are more or less interdependent; general considerations are:

be strong, to carry weight of broken ore, part
in inclined stopes and by friction between ore
; constant drawing of ore destroys any arching
e broken ore self-supporting, and may throw
other level protection. The walls may also
the ore is being drawn (20).
s obviously depends on width of stope.

lle-back stulls, stulls and posts, drift-sets, or level-
the same factors as for open stopes (Art 36); square-
enience instead of drift-sets. In wide stopes, 4 ar-
or more timbered or dry-wall gangways, like those
be built on stope floor; for example, see Republic
Entire sill-floor may be timbered with square-sets;
chute-gates are kept open by lagging (see Cresson
Floor of stope may be opened 10 to 20 ft above the
which CHUTE RAISES are made at frequent intervals
Alaska-Treadwell mines, Art 69. (4) Floor of stope
attempt to maintain haulageways through broken
rking in numerous crosscuts, which connect edges of
by in walls of deposit or in pillars between stopes
). CHOICE OF METHOD is based on a comparison
and subsequent cost of handling broken ore; these
orebody, wages and efficiency of labor, and supply
the same problem. For variations, see Art 69.

ks (or is broken) should be considered in plan-
the levels. Large slabs clog chute-raises and
odge them; this work is annoying and costly
abs are numerous, the resulting delays soon
gates for cheap loading.

rarely serious in narrow stopes in small mines; drill
produced may be cheaply blockholed or spalled in
be broken fine, for handling in small cars or skips.
nay be drawn through simple chute-gates, the in-
y increase first cost of opening stopes. Such gates
er than 8 or 10 in.
deep holes and heavy blasts are desirable to secure
ely break fine, but
ize with structural
ockholing or spall-
; amount of labor
xceed that needed
le 29, Art 28). On
es with grizzlies in-
ed. A compromise
holing and cost of
-gates and shovel-
The latter alterna-
d slabs need to be
by 2 men; these
alanced by cost of
natives, see Alaska-
pper Co, Art 69.
ss-section of chute-
ast 3× max dimen-
he crushers should
achines is cheaper

Fig 310. Cross-sec (after Hoover)

level-pillars at top of stopes, haulageways are
e deposit (Fig 310). Stope is then opened from

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a stope-drift, as indicated by dotted lines; this device secures stron for levels, but is limited to deposits of moderate width; haulagew driven in walls on both sides of stope (Ariz Copper Co, Art 69).

(*e*) Haulageways and chute-gates should be located with refere produced on the surface of broken ore at top of stope by drawing of vertical stopes, 25 to 30 ft wide, the best location for a single row of is in center of stope; if placed along one wall, the surface of broke tends to pitch toward that wall and shoveling or staging is required stope-back on that side for drilling. In inclined stopes, ore tends t on hanging-wall side, regardless of position of drawing points; sor placed on footwall, for drawing off ore there when stope is to be en

J. A. Fulton furnishes following statement as to practice at MELONES MI has a general application (note *D*, Table 30, Art 28). Shrinkage stopes widths of 10, 25, and 40 ft, with dips of 70°, 50°, and 70° respectively. Wi the steep dip are the easiest to keep in shape, as the drawing down of ore is and more easily controlled. With a 50° dip, as ore draws down first on the it is impossible to place holes as advantageously as on 70° dip.

Chute-gates suitable for shrinkage stopes are described in Art 91. BETWEEN GATES should be small, to reduce amount of ore remain them when the stope is finally drawn, as empty stopes are often too d shovelers to enter. Gates are usually 12 to 25 ft apart; use of A-sl (Fig 320) between gates allows larger intervals. Where levels are remaining between gates may be recovered by cutting out lagging a the ore on the level (262).

Access to shrinkage stopes is as follows:

(*a*) Timbered manways are built up through broken ore; in na they are timbered with 2 rows of lagged stulls; cribbed manways wider stopes. For greater security, manways are often placed at en if located elsewhere, cribbed manways should be midway betwe gates, where they are least affected by movement of broken ore duri

At Lake View Consols mine, Kalgoorlie (Fig 309), in steep-dipping stop wide, cribbed manways 2.67 ft square have proved the most satisfactory bering. When located between drawing gates they do not collapse during st ways are abandoned about half way up the lift, raises to the level above th access; level interval is 100 to 200 ft; manways collapse when stope is final many usable timbers are recovered (260).

(*b*) Raises are driven in walls, or in pillars between stopes, and con stopes at vertical intervals of 15 to 30 ft by short drifts. This is com stopes and affords strong protection for traveling and air-ways (se Art 69; also Braden mine, Art 88).

Sorting in shrinkage stopes is not ordinarily feasible. (For exc Cobalt, El Tigre mine, and Republic mine, Art 69.)

Support of back and walls. Shrinkage methods are primarily fo the back will stand unsupported and walls will stand while stope is be (Art 70). For support of the back in shrinkage stopes with chutes, plan, above. Loose slabs in stopes without chutes (Fig 308) may b temporarily by stulls, preferably tapered and set with their big facilitate recovery from above; this is practically the only form feasible during stoping. Occasionally the back and walls of stopes width are timbered while ore is being drawn, to protect men working or to prevent contamination of ore by slabs from walls (Ariz Copper

H. H. Hodgkinson states in this connection (261): Timbering consist supported by transverse sets or frames spaced 4.5 to 8 ft apart. Fig 311 s

s required at each end of the horiz timber. Fig 312
Unsymmetrical sets of this kind are feasible where
. Fig 313 shows a rafter
oor wall at *B*. Timbers
e of back, a space of at
for placing lagging and
extra blocking. Arched
timbers and allow more
g 314 shows extension of
ering of walls; successive
as ore is drawn. (Such
inkage stopes. Author.)

Fig 311

of supporting perma-
Filling is dumped in
o level above through
layed filling operations
han contemporaneous
re open and much of
ns into place by grav-
ing is required only at
. In long stopes, the
wall left unsupported
ing may be reduced by
the ore possible from
-gate in succession,
one end of stope. As
ined, filling is dumped
pty end of stope from
; toe of fill advances
pile recedes towards
f stope (Fig 327, Art
pts made to draw ore
ver entire area of stope,
waste into the top at
ate that ore is with-
ays result in loss of
d serious contamina-

Fig 312

Fig 313

ste. See Ariz Copper Co, Art 69; Inspiration
; and Bib (262, 263).

etails of shrinkage stopes are given in Art 69, 85–88.

Practice, Shrinkage Stopes

ve silver, with arsenides of Co and Ni, occurs in
wacke, and quartzite. Width of veins, rarely over
; values often extend into wall rocks, forming 3 to
. Entry is by vertical shafts, generally following
rinkage stopes are usual; a few veins are mined in
at BUFFALO MINE is typical of shrinkage methods
mpact ore is broken with 5-ft uppers, placed fairly
lling or blockholing. Before stoping begins, 6 by
re; levels are timbered with lagged stulls after the
nlarged the drift to a height of 13 to 15 ft. Ore
d supports miners while drilling holes for a second
rt, and a cribbed manway is built at each end of
drawn during stoping, a 6-ft open space being kept
e ore on one side of vein is first blasted. The high-

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grade streak is then broken with light shots, the ore being sorted, sacked, and to smelter. The remaining low-grade ore on opposite side is then broken (co Resuing, Art 61). At some mines no sorting is done in stopes. At NIPIS levels are timbered with 3-piece sets; raises to level above are used as manway is a section through 2 levels of CONIAGAS MINE, showing method of dealing wit with so flat a pitch that ore will not run along the floor by gravity. Chutes, ra funneled tops, are put up from the level through barren material; they per of the surface of broken ore during stoping and of the final withdrawal of ore.

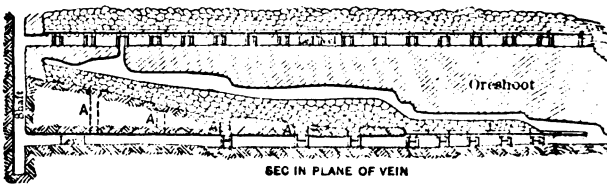

Fig 315. Shrinkage Stope, Coniagas Mine, Cobalt, Ont

El Tigre mine, Sonora, Mex. Data from R. L. Herrick in 1909 (266). ores occur in long shoots in 4 to 7-ft veins, dipping 70 to 75°; wall rocks ar veins carry 4 to 30-in streaks of high-grade smelting ore. Level interval, 100 topography is rough and all levels are drift-tunnels. Mining is done in shrink with chutes 25 ft apart; every 4th chute has a manway compartment. Leve bered with lagged stulls. In stoping, the high-grade streak is first stripped blasting down waste and mill ore alongside; it is shot down on next shift, sor canvas, and sorted out and shoveled into chutes. Enough mill ore is draw chute-gates to keep a proper working space below the back.

Kalgoorlie, W Australia (258). See Art 66 for ore occurrence. Where wa are strong enough to permit, rill shrinkage stopes (Fig 309) are used. Levels, ft apart, are timbered like filled rill stopes (Art 66); chute-gates are built int timbering every 20 ft. Economic length of stulls for levels (set as in Fig 164, 14 to 18 ft; on greater widths, saddle-back stulls have been tried and found unsa At LAKE VIEW CONSOLS MINE, in lodes up to 35 ft wide, it proved advantageou levels 12 to 13 ft wide and to widen the stope to the walls directly above them level was then protected by horiz stulls of moderate length, resting in hitches cu section of stope floor with top of drift. At GREAT FINGALL MINE, with a 2 interval, simple stulls failed on widths over 18 ft. Horiz stulls, reinforced (Art 36), were the cheapest and best means of protecting levels at widths of 1 stulls were 15 to 27 in diam, set 5 ft apart; 1 post was always placed near ea a stull, with intermediate posts as required (260). Only 1 raise in center of h mined is required for preliminary development. Slope of stope faces, 23°; all are water holes. A manway 3 ft sq, carried up at each end of stope, is cribbed round timbers halved at corners; manways are halfway between 2 chute-gates are filled after ore is drawn.

Tomboy mine, Telluride, Colo. Orebody is a gold-quartz fissure, 1 to 1 (normally 4 to 7 ft); dip, 77°; country rocks, andesite and breccias. Shrinka are similar to those in Fig 308, Art 68, but ore pillars are used to protect levels timber. Swell of broken ore is 33%; chute raises are 20 to 25 ft apart. I finally cleared of track, timber and pipes, and the pillar above shot down before below holes through (269).

Cresson mine, Cripple Cr, Colo. Data from R. L. Herrick in 1911 (151). rock is granite; gold telluride ores occur either in massive shoots in fissured zo replacement veins in well defined sheeted zones alongside of or near basic dikes. is done in flat-back shrinkage stopes, details of work being varied to suit size, sh richness of ore; swell of broken ore, 30% to 35%; for data on breaking ground Table 30, Art 28. Levels are usually opened by drifting along one wall of ore its full length; then crosscuts are driven to the opposite wall to determine ir in width and to allow sampling. Level-pillars are left in low-grade ore on width 10 ft; height of pillar = width of orebody. Lagged stulls (Art 36) are used

16 ft. Stulls and posts, spaced 6 to 8 ft apart, are
are 14 to 16-in diam, set horizontally and made by
d joint; posts are 9 ft long. This timbering is strong

Cresson Mine, Plan of 600-ft Level (Fig 317)

riz sec On each level, a breast stope (Art 30), 10 ft
oot, in which square-sets were erected, as in Fig 316,
te-gates. Open square-sets, which were inadequate
had reached some height, were strengthened by lag-
ging the sides of certain sets and then fill-
ing them with ore by shooting down the
lagging overhead; sets behind chute-gates
were left open for access in making repairs.
Level timbering and tracks were all recov-
ered after stopes were drawn, starting at
one end of stope and retreating toward
the other. Flat-back shrinkage stopes
were used. As soon as any part of the
back reached a point 8 ft below floor of
level above, it was caught up on cribs,
wedged tight and in places filled with ore.
Level-pillars were mined as in Fig 317;
work began by drilling holes in back around
a crib in middle of stope; the crib was then
removed and the holes blasted, making an
opening about 12 ft diam into the stope
above. Work then stopped for several
days to allow loosened rock to work and
drop; then the back around the opening
was attacked. Cribs were close enough
together to allow all holes to be drilled un-
der their protection. After center opening
had reached a diam of 30 ft, the remaining
pillar sagged and its wt was supported en-
tirely by the cribs; this wt and the result-
ing incipient fractures materially reduced
consumption of explosive for mining rest
of pillar. This method is stated to be safe
labor and dynamite, as compared with the drilling
ar and shooting them simultaneously. In one place
ied to a height of 320 ft. This saved cost of mining
fairly soft and required thick pillars, would have had
was drawn from this stope on several levels through
the walls.

3 C (p68). Orebody is a vertical, vein-like deposit
neous rocks and limestone; aver width, 25 ft. Ore

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is very heavy, averaging 8 to 8.5 cu ft per ton in place, hence levels mu protected. Drifts are run full width of deposit and timbered with stulls and often 30 in diam, are 5 ft apart, with 8 to 12-in round lagging. After stop timbers may be recovered, beginning at end of stope and retreating toward pillars in preference to timbering are left over drifts where ore contains mu

Center Star mine, Rossland, B C. Data from R. H. Allen, 1909 (147). and data on breaking ground, see Ex 9, Table 30, Art 28. Drifts are full and at least 10 ft high, to give room for placing first set of timbers. If ve 12 ft wide, back of level is timbered with 12 to 18-in stulls, close lagged w holes are left in lagging for erecting chute-gates. Stope is then carried up 8 and another row of lagged stulls placed, with holes left in lagging at interv wall. Above this is the ordinary flat-back shrinkage stope. Broken ore openings in upper row of stulls and spreads out in fan-shaped piles, restir and on lower stulls; each pile supplies 2 gates, which may be lightly const are free from pressure of ore. There is ample room between rows of stulls spalling or blockholing. In wider parts of vein, levels are square-setted, which allows similar facilities for handling ore (compare with Chinaman c

Morris Lloyd mine, Marquette Range, Mich (184). Shrinkage stopes mining a deposit of strong siliceous hematite, 500 ft long by 25 to 75 ft wid 85°; hanging wall gave some trouble from falling slabs. Level interval, haulage drifts were driven near footwall and timbered with 3-piece sets.

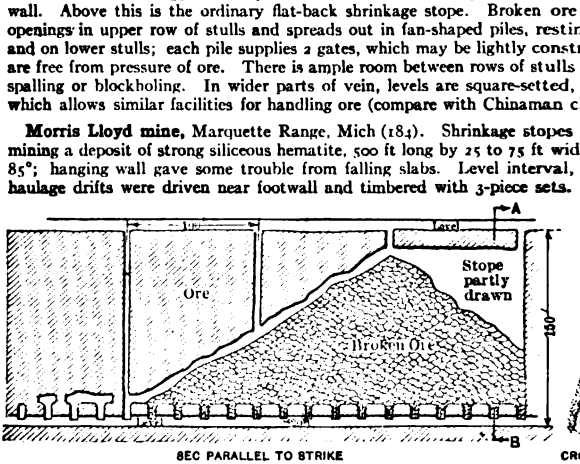


Fig 318. Shrinkage Stoping, Morris Lloyd Mine, Marquette Range

crosscut at 100-ft intervals, to determine width and grade; in wider porti drift was driven later near hanging wall. Max stope width, 40 ft; greate mined in 2 stopes parallel to walls, with a pillar between, or in a series of tra with pillars (Fig 318). Levels are connected by untimbered raises 100 ft ventilate and give access to stopes. Chute-raises, put up at 20-ft intervals 10 ft above back of level, are timbered with 3-in plank and protected with 4 injury by blasting. Tops of raises are connected by a subdrift 15 ft ab stope is then widened to full size, tops of raises are funneled out and stope as a stepped-face shrinkage stope; 30% of the ore is drawn during mining. with 2 stope-drills per 50-ft length of face; 1 laborer per 100 ft of stope b

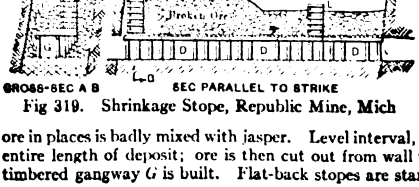


Fig 319. Shrinkage Stope, Republic Mine, Mich

sledges large lumps tion of stope, ore evenly as possible as this seems to re

Republic mine, Hard hematite depo by shrinkage stop shows method in a ft wide, 80 ft long hundred ft high; d

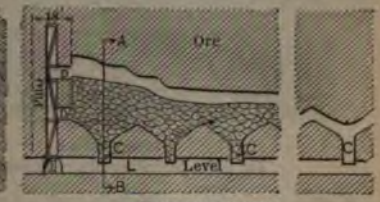
ore in places is badly mixed with jasper. Level interval, 100 to 150 ft. Drift entire length of deposit; ore is then cut out from wall to wall to a height timbered gangway *G* is built. Flat-back stopes are started, and broken ore

d manway *L* being built near each end of stope. WING PLACES *D* are left by omitting a post on either is shoveled from *D* into cars, jasper being sorted tes are advantageous where the ore is broken small re more convenient where blockholing and sorting pened by crosscuts driven from a footwall drift, as ried up to within 10 or 15 ft of level above. Level- g from one end of stope. Final drawing of ore is timbers at end of stope and retreating toward shaft.

In some cases, stopes are filled with waste before be built up through the filling. This work is slow.

oing deposits of hard hematite of uniform grade ned in shrinkage stopes. Chute-raises from main th a subdrift, leaving 10 ft of ore above the level. level above. Where the overlying stope has been pillar below are mined simultaneously, by retreat- entry (270).

Colo (267). Orebody is a steep-dipping, low-grade, 55 ft). Walls are andesite, rhyolite, tuff and breccia, s dry, does not pack in stope when broken, is strong,



SEC IN PLANE OF VEIN

hrinkage Stope, Gold Prince Mine, Colo

rom top of each chute-raise 4 raises are driven on 2 along strike; these connect with similar openings e lowest breakthrough in the pillar at end of stope. o form funnel-shaped openings over tops of chute- . flat-back shrinkage stopes are opened; about 30% holed or spalled in stopes to a max dimension of 2 ft. ansverse pillars or ribs 18 ft thick. Vertical raises level above; short drifts *D*, called breakthroughs, are driven from raises to give access and ventilation to stope. Only a part of the pillars may be recovered when stope is drawn. This is a small-scale example of the Alaska-Treadwell method (see below).

ROSS-SEC A B

, Metcalf, Ariz

Arizona Copper Co, Metcalf, Ariz (152). At KING MINE, 2 lenticular oreshoots, 500 and 700 ft long, occur in a fault fissure in granite porphyry; max width, 30 ft; walls are strong and well defined; dip, 70°. Fig 321 shows method of mining parts of these shoots. On each level, haulageways were driven in walls, the vein was crosscut at f shoot excavated as a breast stope (Art 30), the ore hen carried up as a flat-back shrinkage stope, 33% oarurts by shoveling. Access to stope was through

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lines of chute-raises; this saves shoveling and the ore thus left is ultimately c stopes in lift below. Tops of chute-raises are funneled out while stope floo The stopes are arched-back shrinkage stopes (see sec *XY*, Fig 324). Of th 33% is drawn during mining. Stopes are carried until they break into thos above, where a shelf of ore ("sheet pillar") is left around edge of stope to pro work below during the final drawing. Fig 324 is diagrammatic; details are local irregularities, as indicated by Fig 326 (an actual plan of 900-ft level at Tr on which stope outlines have been projected). For data on breaking grou Table 29, Art 28. SUMMARY. This method gives a large output per man st economical handling and requires little timber; all of which are essentials deposits. The narrow development work in intermediate drifts and chute-r sive, but is justified at this mine by the economy of handling ore through chut rifice of 20% of orebody in rib pillars for support is admissible only in low-g

Homestake mine, Lead, So Dak (104, 271, 272). Orebodies are hu mineralized zones in clay slate; dips, steep; widths, from a pinch to 500 ft. strong, altered country rock, containing quartz and sulphides; it averages ton, mostly in free gold; both walls are strong. Deposits are worked along st 1 mile, and in 1916 were depth of over 1 850 ft. Th vertical shafts in hanging c in ore; levels are 100 ft a 1 100 ft level; below that. MINING SYSTEMS used here gone a process of evoluti resulting in reduced timber Early work was done ent square-set stopes (Art 45). age stopes were carried 60 f

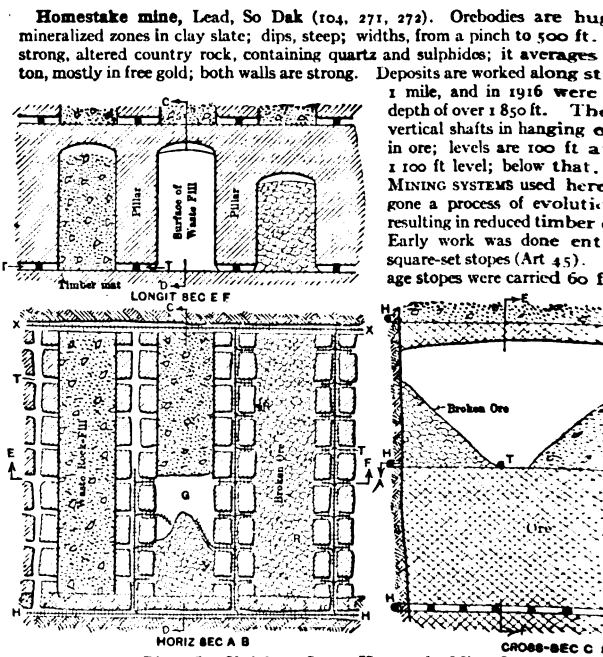

Fig 327. Shrinkage Stope, Homestake Mine, So Dak

the deposit, separated by 60-ft pillars, a pillar 20 to 30 ft thick being left floors were completely timbered with square-sets, through which haulageway open. Ore was drawn by shoveling on sill-floor at sides of haulageway sets, the stopes were filled. Pillars under levels and between stopes were then mine setting. Later, complete square-setting of sill-floors of shrinkage stopes was haulageway sets only were erected, and were found to stand without swinging tions of present methods (described below) do away with nearly all timber

B. C. Yates (104) gives following data on system at DEADWOOD-TERRA M 30 to 50 ft wide, dipping 75°. On each level a crosscut was driven throu from it a breast stope from wall to wall was carried each way to limits of ore, a ore was removed. When the shape of the orebody was ascertained, a drift parallel to the footwall and about 20 ft away; drift and stope were connected

d up as a shrinkage stope, about 40% of the ore oveling. Stoping could be done simultaneously l-pillars were left at top of stopes. Level-pillars ve were finished and drawn. The only timber ways. This general plan has been used on vein nsverse stopes are carried, separated by pillars. ls are opened by driving main haulageways *H*, sscuts are turned off at intervals of 102 ft and on center lines of pillars, which are 42 ft wide; 'breakthroughs") are run each way from cross- occasional raises are put up for ventilation. A ried over entire sill-floor; where deposit is very neat hanging wall. At ends of breakthroughs open cribs covered with sheet steel, just high s; tops of breakthroughs are enlarged at stope ope is then carried up as a flat-back shrinkage stope above, about 20 ft thick at crown. The g in breakthroughs; miners enter stope through ways *K*. BREAKING GROUND. Breast holes are as much ore as possible with each blast; block- an fine breaking from face. DRAWING STOPES. he end and is followed closely by filling (see sec area of walls left unsupported while ore is being ome useless they are removed; track is laid into ille; ground is so strong that men may work in Empty stope floors are covered with a mat of l with timber for a height of 15 or 20 ft before n an opencut or from old workings above, and cting with crosscuts in the pillars on level above, rom *X* may enter stope through the level-pillar, *CD*, Fig 327). MINING PILLARS. After stopes then the pillars between stopes; all are removed ts (Art 45). GENERAL. As a whole, the above ed method (Art 84), involving both shrinkage ed here for comparison with Alaska-Treadwell alternative methods for similar orebodies.

, Shrinkage Stopes

tage methods is limited by their rigid require- haracter of orebody and its walls. These are uiring little timber and practically no shovel- wards low costs. Under proper conditions, narrow or wide veins; as auxiliaries to other ion to large, massive orebodies (Art 84–89).

nkage stopes without chutes (ore-passes); their us if read in connection with the limitations of discussion is based largely on a summary by see Bib (83, 132, 260, 274).

lication of shrinkage methods: (*a*) Dip of 60°, for ore to settle freely; tendency of nging-wall side decreases as dip steepens. ape, otherwise loose ore will lodge on foot- s are usually too dangerous for shovelers ay be timbered during drawing (Art 68), ility of using open stulled stopes (Art 36, t to mine in shrinkage stopes, even if in ; sudden flattening of a shoot necessitates ises through unpayable ground (see Conia-

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gas mine, Fig 315, Art 69). Regular and fairly continuous orebo essential, because with shrinkage methods stoping must be carried ably in advance of output requirements. (*c*) Ore should be stron stand over back of stope, occasional slabs being supported by stu of weak ground may be broken underhand (Ariz Copper Co, Art 6 sistently weak ore can not be mined in shrinkage stopes. (*d*) Ide should be of uniform value; at least it should be payable throughou of waste can not be left unbroken, and sorting in stopes is rarely f exceptional conditions at Cobalt, and at El Tigre and Republic mi (*e*) Walls should stand without crushing or spalling off into the brok stopes are drawn. Friction between ore and walls during drawi any natural tendency of walls to slab off; the hanging wall especial strong. (*f*) Prospecting openings cannot be run in the walls of shrin hence limits of orebody should be well defined. (*g*) Physical or n peculiarities of ore may prohibit shrinkage stoping.

Some broken ores tend to "pack" in stopes, and must be blasted out; o occurs only when stopes are left undrawn for a considerable time; freq prevents packing. Packing may be due to pressure or to cementing action o on pyritic ores; it may be controlled if the size of stopes and the plan of regulated for drawing stopes promptly. At Coronado mine, Ariz (Art 69) the considerable pyrite. Broken ore when drawn was found to be highly oxi caused trouble in concentration and led to abandonment of shrinkage meth

Advantages of shrinkage stopes: (*a*) Raises and levels may be far apart, t cost of development. (*b*) No shoveling of ore is required in working spa stope; if ore can be drawn through chute-gates, there is no shoveling (se conditions justifying shoveling in drawing off ore). (*c*) Little timber is u for miners and shoveling plats are unnecessary; timber for manways is rec stope is drawn. (*d*) Ventilation is good, as air currents follow working face pared with filled stopes, shrinkage stopes sometimes wholly save cost of fil event, filling an empty stope in one operation is always cheaper than conte filling (Art 59). Shrinkage stopes also avoid loss of fines in filling, and cost and of shoveling ore to them. (*f*) In narrow veins, shrinkage stopes c stulled stopes (Art 36, 37), and are preferable to the latter for wide veins and They furnish a firm working floor for miners; timber staging may reduce causing a feeling of insecurity. (See Art 43 for comparison with sub-level st

Disadvantages of shrinkage stopes: (*a*) If chute-gates are used, fine br quired; some large pieces may be buried in broken ore, and make trouble in and gates. For alternative mode of dealing with this problem see Arran protecting levels, Art 68. (*b*) Some spalling of waste from walls into ore alw a given amount of it reduces the aver value of broken ore more in narrow t stopes. Much ore is lost if walls crush. (*c*) Ore can not be selected from di faces, for maintaining a uniform grade. (*d*) Ore left in stopes until they are up a considerable investment for labor, explosives, etc, to break it, and ad capital is necessary. At a given mine, the tonnage of broken ore eventual fairly constant maximum. Interest on money required to break this to operating charge against shrinkage methods; it represents the cost of utili ore for support. B. C. Yates states that, in 1904, over 1 000 000 tons of brok in stopes at Homestake mine (104). H. C. Hoover cites an example where tons annual production, the interest (5%) on the capital represented by amounts to 7¢ per ton treated (20). A large reserve of broken ore is adva insuring a regular output; but, only the ore in completed stopes is availa purpose. In the few Lake Superior iron mines using shrinkage stopes, feature may be availed of to save cost of stock-piling and reloading from 60 the ore mined while navigation is closed. (*e*) Shrinkage systems are not flexi once started, they are difficult to alter, owing to lack of frequent raises; this true if filling is the only alternative. Change of method may be required b dip, or in character of ore or walls in depth. (*f*) Walls of shrinkage stopes can for lower-grade ore only when stopes are filled promptly and filling is left u walls often crush when filling is drawn down for use in lower stopes.

G METHODS

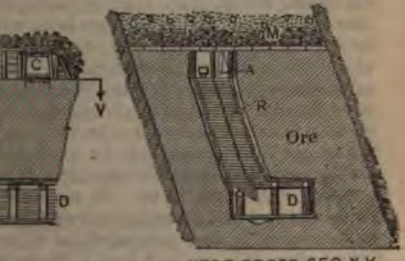
ose in which ore is first undercut and then
by weight of overlying rock, or by a combina-
custom, operations involving caving of the
ystematic and essential part of the work, are
gh practically all the ore is broken by drilling
ods, SUB-LEVEL CAVING, BLOCK-CAVING, and
ification, each having many modifications.
o Combined Methods, Art 84.

Top-slicing

n the U S was developed from the "North of
n certain English iron mines. Term top-
plied (270) to sub-level caving (Art 76); to
ses to call the former method "Top-slicing
"Top-slicing and Partial Ore-caving" (286).
y for top-slicing in some Lake Superior dis-
lled "Slicing."

asses, or thick beds of soft ore, where clean
sired and where the overlying surface need
s and conditions, see Art 75.

General plan. Ore is mined in a series of horiz floors or slices, taken in descending order, from the top of the deposit. Each floor is mined in small sections, the roof of each section being allowed or forced to cave before an adjacent one is attacked. Work on each floor retreats from limits of ore toward points of entry; all of the ore is broken by blasting. The principle is illustrated by



VERT CROSS-SEC X Y

ng, Lake Angeline Mine, Mich

ake Angeline mine, Mich, 1894; J. P. Chan-

ut 75 ft wide and overlain by glacial drift. First
ebody by driving a drift *D* about midway between

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walls; 2-compartment, cribbed raises *R* were put up to the overburden a
From top of each raise a 7 by 8-ft drift *A* is driven 50 ft each way, meeti
as shown, or a drift from the next raise. Drifts are timbered with 3-p
advanced by spiling, where necessary, as under loose overburden at top of
begins by driving 7 by 8-ft crosscuts *C* to the walls, at points half-way be
ends of drifts alongside caved ground, as shown. Crosscuts are timbe
sets of 6 to 10-in round timber, spaced 2 to 4 ft apart. Ore is shoveled
1 500 lb and trammed to chute compartment of raise. When a crossc
floor is covered with split lagging *F*, laid close together on 3 8-in roun
crosscut sets are then blasted down, allowing overburden to drop. Con
are driven and caved in succession, working back toward each raise unt
high has been removed from top of orebody. Other slices of same heigh
mined below the first. After taking several slices, the work assumes th
Fig 328. A mass of twisted timber *M*, the MAT, collects at the top, its thi
as work descends. The mat forms an artificial roof between ore and
subsidence may be controlled and it is easily picked up on timbers of th

Slicing usually begins on a new floor as soon as it will not inte
on floor above, and before the latter is finished; this maintains a
Simultaneous work on several floors is shown in Fig 342 and 344.
ings like drifts (*A*, Fig 328), driven on each floor as a prelimi
slicing, are collectively called SUB-LEVELS. These are designat
No 1 being highest on each lift. Term SLICE means a horiz layer
also any opening driven for removing blocks of ore outlined by sub
ment. Thus, crosscuts *C*, Fig 328, are slices; the exact meani
must be inferred from the context.

Modifications. All top-slicing resembles in general Fig 328, but detai
different orebodies. Variations in the timbering of slices give rise to term
PROP-SLICING, and SQUARE-SET SLICING, denoting work in which the sli
with drift-sets, stulls, and square-sets respectively. Drift-slices may b
sets wide, or square-set slices may be 2 sets wide and 2 to 4 sets high, if
Such excavations are called ROOMS, this work being sometimes termed
rooms" to distinguish it from "top-slicing by drifts" (or crosscuts), as i
Art 72–74 show details of modifications of top-slicing.

Development. ENTRY. Shafts must be located where the
affected by caving. If the position of property lines compels sinl
overlying the orebody (Art 72), pillars of ore must be left for sh
and these are not minable unless they can be reached from adjoi
On Mesabi Range, Minn, auxiliary shafts for handling timber are
over the orebody, the shaft pillars being mined last; even then,
be lost through wearing away of the outside of pillars as the
around them (Art 72; also Calumet & Ariz, Art 74). LEVEL INTE
45 summarizes practice at mines described in Art 72–74; levels
to 75 ft apart. The intervals must be smaller in wide deposits and
ore than in narrow orebodies or stronger ore. In most ores soft e
slicing, the maintenance cost for levels and raises is an appre
tending toward a small level interval; cost of hoisting slice timber
increases with height of lift. In some districts, a balance is st
these costs and that of driving and equipping haulage levels, as fo
for mechanical haulage are spaced 100 to 300 ft apart vertically,
ditions allow, only one such level is driven at bottom of deposit.
hand-tramming levels are then opened as required, 35 to 50 ft ap
they are often connected with a shaft in the country rock, throu
bers are lowered. LATERAL DEVELOPMENT. Levels may consist o
(Fig 328), but in orebodies of large horiz section a rectangular sy
and crosscuts is often driven, arranged to allow proper spacing
handling ore from slices. Layout of intermediate levels, where us

Top-slicing Details

ween Raises	Method of timbering slices	Height of slices, ft	Width of slices, ft	For details, see Art No
(a)	*Ds, P, Sq*(a)	10–33(*b*)	(*a*)	72
40	*Ds*	10	8	73
50	*Ds*	10	.....	73
30–50*	*Ds*	8–9	6–8	73
50	*Ds*	12	.....	73
25–40	*Ds*	7–15(*d*)	6–10	74
50*	*P*(*e*)	11	5	74
50	*Ds & Bk*	10	7–12	74
50	*Ds*	10–17	9*	74
(*c*)	*P*(*e*)	8–10.5	5–6	74
50*	*Sq*	8–9	5–6	74

for drift-slices, 11 to 12 ft. (*c*) See Art 74. (*d*) aid under floors and caught up on props in slice t used. *Ds* = Drift-sets. *P* = Props. *Sq* =

lt details of work in Art 72–74.

ing then determined by location of raises wide orebodies short trams can be secured m a haulage level consisting of 1 or 2 drifts so located as to strike an economic balance es and cost of development; close spacing the latter. Table 45 indicates that the this does not indicate the tramming distance with area of orebody and detailed arrange- any cases, raises provide the only means of slices, and hence must have 2 compartments Art 72). SUB-LEVELS. Where slices are 11 often driven as untimbered drifts, say 6 ft s in Fig 328. This increases speed of sub- then enlarged at entrance of each slice, just y sub-levels opened long before they are ing of the mat on the overlying slice throws kwire mine, Art 73, and Fig 342, Art 74, for on several floors, without opening sub-levels e to rest. In general, development plans

form of breast-stoping (Art 30). In high square- For examples, see Fig 333, 335, 336, 343, and

mparatively small areas of mat for only a poor grade are adequate; to cheapen costs, ible.

posts and simple framing, or none at all, are most generally strong enough; (*b*) floors laid without n the underlying slice; (*c*) in working alongside d along the solid side before the slice is caved, to e the same advantages, but require more timber. drift-sets (see Bingham, Art 74), or in high slices

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where single drift-sets would require posts of unmanageable length and drift-sets would not support the side pressure of caved ground (Art 72). Pr stulls) are employed where floors are laid on sills, the latter taking the place a drift-set. Props are also used in slicing under rock capping at top of an under peculiar conditions in Minnesota iron mines (Art 72). For light pressur (Fig 350, Art 74) form cheap and satisfactory supports for plank floors laid See Art 72–74 for details of timbering.

Floors may be of lagging (Fig 328) or plank. Plank is best, bec be laid with tight joints to keep out fine waste; it is also easier to block up from below than round lagging. Usually, one layer of 2 by 12 plank, in 10 to 16-ft lengths, makes a good floor; double layers ma in a few slices at top of orebody; on Mesabi Range, single layers of have been found adequate (Art 72).

Sills under plank floors are generally omitted. To avoid breakage, the pla directly on ore, hence sills must be sunk in the floor, as in Fig 348, Art grooves for sills is slow and costly work, and, as it is practically impossible t on successive sub-levels directly under one another, there may be trouble i sills from below. Sills allow use of props instead of drift-sets in slices, but compensating advantage, as it effects no material saving in timber. Th omission of floors on certain sub-levels is possible only under a thick mat feasible in high-grade ore. Data as to this are conflicting. It is successfu at Cananea, it increased costs in spite of saving timber (Art 74).

Handling timber. In top-slicing, as in Fig 328, all timber for slic hoisted up the ladderways of raises; small air-hoists of the stretcher-ba 15, Art 21) are commonly used for this purpose. See Art 72 and Caspia 73) for modes of arranging work so that timbers may be lowered to

Blasting down timbers in slices to force mat to cave is done by sir firing of small charges of dynamite in holes bored in the posts. Holes near the middle of posts, and are 4 to 6 in deep; in large-scale work, can be saved by having 1 or 2 men bore all holes with air-driven aug a stick of dynamite or less is used per post; it is often unnecessary the posts in a slice (see Fig 338, also Caspian mine, Art 73).

Some engineers hold that mat settles in better shape if the supporting allowed to fail by pressure than if they are blasted down; some blasting is alwa Top-slicing is possible only when the mat caves promptly and rests solidly on this condition is apt to be met when posts in slices are shot down. For har areas of mat which "hang up," see Morenci-Metcalf district, Art 74.

Recovery of timber is rarely attempted in top-slicing, largely because che used, which will not stand removal and reuse. But, possibilities of salvage a by following statement from P. B. Scotland, Gen Supt Ariz Copper Co, Morer

"In mining coal by longwall method (Art 108), 25 to 50% of the stulls support are recovered. When mat becomes thicker in our top-slice stopes, a similar sa be made. At present, the mat is often penetrated and broken by vertical st would not occur if they were removed. In longwall working, it is . . . . more to timber closely and make large recovery of stulls than to reduce the number and lose all of them."

Height of slices. (*a*) Height of drift-slices and of those where sills c above are picked up on stulls is limited by max length of post whi handled; this is determined partly by size of openings through whic must pass and partly by comparative costs of handling and erecting different lengths. Usual height of drift-slices is 10 to 12 ft (Table 4 represents an economic mean of all the factors. Art 72 gives exampl drift- and prop-slices, and reasons for their use. (*b*) In very heavy height of slice is generally reduced to avoid need for heavier posts, w more to handle and often more per bd ft than the 6 to 10-in diam comm

swinging" under heavy pressure (Bingham, fluences possible height of slices. Square-set lices 17 to 20 ft high are sometimes timbered . Lateral pressure from caved ground limits Art 72; Cumberland-Ely, Art 74; Bib (278). a thick strong mat than under a poor one. f orebody until mat is well formed, then the naximum (Cumberland-Ely and Ariz Copper may limit height of slice. At Miami (Art igher than 10 ft faster than it can be mucked ses mat to cave prematurely (285). (*f*) In e economic maximum, to reduce the number g timber.

itchell top-slicing method (Fig 351, Art 74), e ore from upper floors is handled through ore broken in top-slicing must be shoveled. shoveling plats should always be used. In raises or handled in barrows or small cars. uires raises spaced 25 to 30 ft apart through- aises; also Ariz Copper Co, Art 74).

en barrows and cars, for transporting ore to ng extreme examples:

very heavy, slices are narrow and timbers require 350 lb are used on sub-levels. 1000-lb cars were uld not make the short turns required, nor reach timbers in narrow runways. It was found that 75 ft (66).

ub-levels are straight and usually stand without systematically; trams on sub-levels may be 300 n cars at much lower cost than with barrows.

of raises provide for this, which aids in maintain- decreases as slicing approaches a level. Close ers on levels is then necessary to avoid delays. torage, while slicing on lowest sub-level of a lift

is thrown back and becomes a part of the mat; ing systematic underground sorting.

begin directly at the top of orebodies overlain re necessary, to advance sub-levels or slices. and costly, and causes unavoidable mixture which may be cheaper in spite of the ore ft below top of orebody. Pillars thus left level, and aid in holding back fine waste dur- ian mine, Art 73; also Mesabi practice where , Art 72).

ng orebodies overlain by a ROCK CAPPING; re often irregular. It is then essential that g up over large areas, which may give way flat surface under which to begin slicing is ps of orebodies are usually mined as open ng are examples of practice (see also Mesabi 52, Art 74).

nci, Ariz (Art 74), tops of highest raises are con- ich an open square-set stope, usually 10 to 20 ft

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high, is carried to capping. Stringers of ore extending into capping are fo which also weaken the back. Stope floor is covered with 2 layers of 2-i angles to each other. Deep uppers are drilled at frequent intervals in holes are also drilled in posts of square-sets; all holes are blasted simult

At Ariz Copper Co's mines, "tops of orebodies are mined by open st stopes, or sub-level caving (Art 76). A double floor of 2-in plank is laid caved either to surface or sufficiently high to make it safe to work under

At Cananea, Mex (Art 74), tops of orebodies are mined with square-s of 2-in plank laid on 5 by 10-in sills, 10 ft long. Capping sometimes sta to allow salvage of some timber. When capping hangs up, sets are no but are filled with waste to form a cushion above the mat on which larg without endangering work in the slices below (282).

In this work a continuous square-set stope over whole area of ore essary. As 1 or 2 floors of caved sets sufficiently protect the slices height are opened at different elevations in different parts of the quired by irregularities in lower surface of capping. This avoids h stopes of large area, and does not interfere with slicing. Where co ore and capping is regular, simpler methods of starting a mat ar capping is very soft, slicing begins at top of orebody, usually with d first few sub-levels to build up mat quickly. In harder ground, a br 10 ft high is carried across top of deposit, with stulls and headboa the back. After laying a floor, the timbers are shot down. The fo tions are also preliminary to other caving methods in which we ground above the workings is an essential.

Ventilation is generally poor in top-slicing unless fans are used. Sl "dead ends;" decaying timbers in mat give off heat and some CO. F artificial ventilation, see Miami, Cumberland-Ely, and Bingham, Art 74; installation of blowers caused a marked increase in efficiency of labor (28

Percentage extraction by top-slicing is high; usually only 5 to 10 ore is lost. Further details of top-slicing are given in Art 72–74.

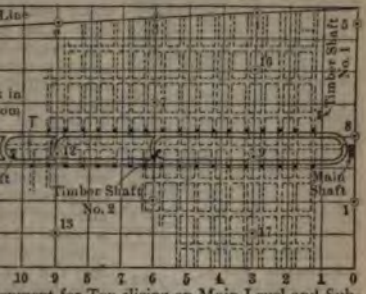
72. Top-slicing, Mesabi Range, Minn

General. Mesabi orebodies are flat, or slightly dipping, thick beds of h ing large areas. They are overlain by taconite or slate and glacial drift, or Ore varies in character, even in the same mine; usually it is readily dri augers, but is compact enough to stand untimbered in small drifts for sever of it is hard. Open-cut mining is done where possible (Art 97).

Underground mining, all by top-slicing, is confined to small or deep de would not pay to strip. Examples of typical conditions for underground mi mine, near Hibbing, is a 40-acre tract (the usual area worked by 1 mine 60 ft thick, averaging 40 ft, and underlies the whole tract; shaft is vertic estimated tonnage, 4 to 5 million tons. WHITESIDE mine, near Ruhl, has underlain by ore from 45 to 200 ft thick; depth of overburden, 50 to 125 tonnage, 5 to 6 million tons. Following notes are based on data publishe 1913 by C. E. van Barneveld (35); W. Bayliss, E. D. McNeil, and J. S L. D. Davenport (276); A. L. Gerry (277). For modes of prospecting an and information on ore occurrence, see Art 11, 12.

Development. Usual ENTRY is by a 3-compartment hoisting shaft, sun part of deposit to facilitate drainage (Art 17). Shafts generally penetra a location in country rock is preferable. From 2 to 4 timber shafts are sun geous points, the number depending on area of property; the first (Timb Fig 329) is sunk near hoisting shaft and at same time. Timber shafts are 6 9 ft in section, and contain a ladderway. A MAIN LEVEL for haulage is ope of deposit, which usually contains one or more troughs of deep ore. M follow the troughs, their best position being determined from a contour m of orebody based upon borehole data (Art 12). Drifts should be near bott yet kept in ore to avoid more costly rock excavation; to minimize hand sub-levels above, they should extend if possible throughout length of prop

ctors; in deposits 50 ft thick or more, it is good
height that it will extend a considerable distance
and later to drive another and lower main level.
sist of 2 parallel drifts, 60 to 100 ft apart, con-
(Fig 329). Double drifts give better ventilation
one drift strikes rock, it may be temporarily dis-
e other. Main drifts are timbered with 3-piece
sts and 10 or 12-ft caps, lagged on top. In thick
at vertical intervals of 50 to 60 ft. Untimbered



opment for Top-slicing on Main Level and Sub-levels, Mesabi, Minn

RIFT (*T*, Fig 329), timbered with 3-piece sets, con-
nd ladderway of main shaft. These connections
timbers are lowered in them from the surface.
n detail. Sub-level drifts are sometimes opened
r cross-drifts, say 100 ft apart, may be run to
ut up, from which the sub-level is opened. The
between length of hand tram on sub-levels and
e on main level.

the ore limits, or at property lines, and retreats toward raises. Details vary with height of ore over the sub-level and with character and shape of back. Following are variations:

e of Deposit, Mesabi

Drift-slicing. Fig 330 shows method of slicing at edge or "shore line" of deposit; 3 by 3-ft test raises *R* are first put up at 50-ft intervals, to determine height of ore over the sub-level. At a point *C*, where ore is about 7 ft high,

eight of ore; first slice *EF* is then taken. Cross-
ts, with 7-ft caps and vertical posts of a length

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to suit height of ore. Slice *EF* is advanced until ore becomes too thin to work; p
sets are usually 24 to 30 in. Other slices are driven alongside (see dotted line
shoveled or wheeled to a car in crosscut *DE*. As the rock at shore line usuall
steep angle, it does not pay to turn the trac
slice. For this kind of work, sets are best p
their caps along the sides of *DE*, instead of ac
330). Though slicing may be started direct
sub-level drift, a distinct gain results from dri
liminary crosscut, and slicing up the slope of t
rock; ore comes down grade, posts of slice-s
about equal length, and the whole layout is
ible. When ground in slices becomes heavy
bers are shot down and new slices started alo
ground to right of *DE* is thus mined and caved
shows drift-slicing alongside of caved ground
the left side of *DE*, Fig 330), after work descr
is completed. Work is begun by enlarging
the drift and erecting sets 1 and 2 on each side,
caps parallel to sides of drift. These 4 sets,
SETS, protect the entrance to the slice and are
sprags between both caps and posts. Slicing
taking out sets 3 to 9, and the track from sub
is turned into this slice as it advances. If t
at end of slice has been caved, a pillar of o
331) 1 or 2 ft thick is generally left to hold
If timbers do not take much weight, sets 1
next mined consecutively. If ground is hea
second slice are removed in reverse order; w
in set 15, as it is easier to attack than No 1
taken out next. Floor of sets 9 and 16 is co

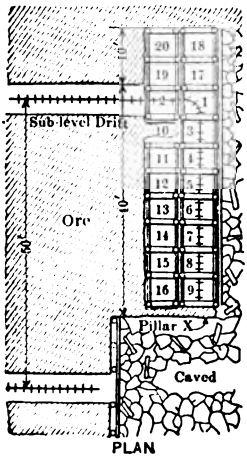


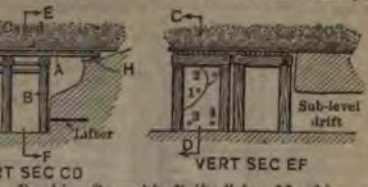
Fig 331. Drift-slicing, Mesabi

boards and the pillar *X* is mined; a little sand may run in from adjoining cave
no harm. Sets 13, 12, and 11 are mined next in the order named, and the sides
to 11 against solid ore are boarded with horiz 1-in boards spiked to posts. Sets
opposite side of sub-level drift are then mined, and finally
set 10, which thus far has served as a pillar to protect the
entrance to slices. This work forms a room 2 sets wide by
50 ft long, known as a double slice. The order in which
sets are taken varies widely. Under a strong back, or with
solid ore on both sides, rooms are mined 2 sets wide in one
operation; where caving had not completely filled the pre-
ceding room, sets 10 to 16 would be taken first, and then
sets 9 back to 3. Ground under dangerous overburden,
such as that containing quicksand, is mined in slices 1 set
wide. The reason for slicing from sub-level drift 40 ft one
way and only 10 ft the other is that each room then requires
but 1 curved section of rails. Ore from sets 17 to 20 is
shoveled to a car in the sub-level drift. When a room is
finished, it is cleaned out and its sides against solid ore are
boarded up; if there is ore below, the floor is covered with
a layer of 1-in boards; posts in center and on cave side of
room are then blasted down. Entrance of sub-level drift
into the room is also boarded up (Fig 332), leaving a door
as an exit for the miner who shoots down the timbers, and
for entrance if it is necessary to blast the side of the old
cave to make the room fill. As rooms usually fill quickly

Fig 332. Boarding
level Drift, Me

with caved material, slicing may begin alongside in a few hours. Fig 333 show
BREAKING GROUND and placing timbers. Holes 1 and 2 are blasted and squared
3 lagging poles *A*, placed across top of excavation, are supported by cap of set
hitches *H* cut at top of breast; short boards are put on top of lagging poles,
then wedged up tightly from below. The round is squared up by holes 3 and 4.
OF DRIFT-SLICES on the top sub-level depends on irregularities in the top of ore
length of posts in successive slices is changed to fit; max length of post for dri

-ft slice, is preferred. Other modes of timbering an a 12-ft slice. WIDTH of a drift-slice is always



Breaking Ground in Drift-slicing, Mesabi

pment, difficult ventilation, and delay in ore pro- oms 2 or 3 sets wide and 2 to 4 sets high; com- s are boarded up and caved as in drift-slicing. d is elastic, and useful under pitching backs or floors. Fig 330 shows typical conditions; the 26 ft high, and would be sliced from existing sub- change from drift-slicing would begin where ore ft high. The first square-set slice is taken in the ng 1 set of ore standing between it and last caved o insure that the square-sets are in line and at to sub-level drift. This slice, 2 sets high, is tim- dvances; the pillar between it and the drift-slice d, the order of removing sets depending on weight In succeeding square-set rooms, the first slice is gside the cave, the caps with the timbers on wall room. TYPICAL DETAILS: et (Fig 234, Art 49) illus- Fig 334 shows a room next or set *A*, which is firmly wide across the set, which s a chute-gate for loading n broken into with a hole up to take set *B*; set *C* is on set *A*. Other sets are back of highest set is blocked and lagged; side d next (Fig 336); its top is covered with lagging but the lagging runs at right angles to that on long dimension of chute opening will be parallel ng from *A* into *D*. Sets *E* and *F* are then mined, C. Work is resumed on sill-floor in set *G*, and slice 1 set wide has been mined along the cave *H* and those over it are taken last. Second slice ay, starting with set *A'* (Fig 334); ground in sets tc, is attacked from *D*, *G*, etc. Track is slice only; ore from ond slice is handled 7). Usual length of ding set in sub-level of connecting rooms, ; if higher, the side difficult to prevent room, all lagging is g up posts along the ts, posts next to ore are boarded with horiz 1-in the same. Holes are then drilled in some of the is boarded up and timbers shot down. Rooms

Fig 335. First Hole for *B* Set, Fig 334

Fig 337. Chute for Square-set Slicing, Mesabi

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may be blasted with or without system; Fig 338 shows a systematic me
marked by arrows are blasted; caps and posts at *A*, *B*, and *C* are expe
props and allow room to cave without disturbing timbers against solid ore.
shoot down posts next to cave, 1 or 2 center posts and a few caps, claiming i
settling and leaves timber in better sha
adjacent slice. Some timbers on the or
often fall when the cave occurs; they a
ground is timbered in taking the slice al

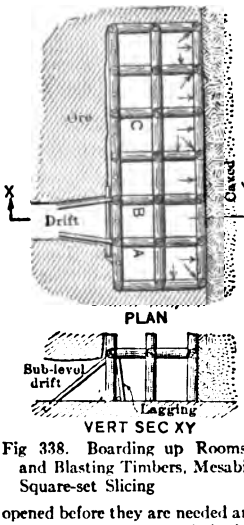

Fig 338. Boarding up Rooms and Blasting Timbers, Mesabi Square-set Slicing

Second and lower sub-levels. Whe
part of the top sub-level reaches withi
raise from main-level drift, a second sub-
by starting another drift from the raise,
the one above. Parallel drifts are dri
raise as work is finished on the top sub-le
the second sub-level will resemble the t
Fig 329; same procedure is followed in
sub-levels. Vertical distance between s
justed so that total height of ore betwe
sub-level and floor of main level is divide
to 15 ft high, preferably not exceeding
dimensions provide a height of 5 to 9
between the back of a 5 by 6-ft sub-leve
mat above; drifts usually stand untimbe
protection. Slicing on second or lower s
at property line or "shore line," and is c
raises as on the top sub-level. Slices are
drift-sets; their caps pick up the mat or
lying sub-level; flat top surface and uni
slices simplify this work. Slicing may
taneously on 2 sub-levels, provided work
is at least 50 ft ahead of that below, but
opened before they are needed are often badly crushed when the sub-level ah
and require much extra timbering.

Prop-slicing. Extensive areas occur where ore is overlain by a consider
of firm taconite; the slice at top of deposit can then be taken out with p
required; max length of prop, 20 ft. Some orebodies 5 to 18 ft thick lie be
of taconite, the roof layer being 40 to 50 ft thick, very hard and tough an
when a large area is mined out. Such ground is developed by a main haul
which drifts are run 50 ft apart to the "shore line;" the pillars are then re
slices. Drifts are timbered with 3-piece sets; slices, with 10 to 18-in props
ft apart on cave side of slice. Where height of ore exceeds 12 ft, 1 floor of
erected and 2 props are set on each cap. This form of prop-slice is an open
it gives cheap ore and leaves large rooms which are not caved unless the ro
the roof comes in large blocks when it caves.

Variations and combinations of foregoing methods, to suit local condition
ideas, have had a limited use; thus, TOP-SET SLICING is a mode of timberi
with 2 tiers of drift-sets, one above the other (278).

Handling ore. Usual tramming distances on sub-levels to the raises ar
electric or mule haulage is used on main levels; irregularities in the bedro
transfers necessary.

Extraction by these methods is high if work is done carefully, keeping
clean, trimming the back and catching all ore on sides; it is from 95% to

Cost data below were published in 1912 by C. E. van Barneveld (35)
costs, see Art 21. EQUIPMENT COST for better class Mesabi underground m
for a life of 10 to 20 yr and a monthly output of 25 000 tons or more, is fro
$150 000, exclusive of employes' dwellings; of this, the shaft and undergrou
cost about $55 000. DEVELOPMENT CREW varies from 75 to 100 men during
and surface construction work, which ordinarily take 8 to 12 months. T
to 200 men are employed during development period, depending on spee
number of working faces. From 10 000 to 30 000 ft or more of developm

sary before slicing can proceed on a large scale. roducing 20 000 to 30 000 tons per month during men, as follows: 100 to 150 contract miners on underground and 40 to 50 on surface on company a contract gang on drifting or slicing consists of Contractors do their own timbering, track-laying, ny men operate main-level haulage. Company aced underground at convenient points for the ude: 4 picks, 4 hand-augers (3.5, 4, 6, and 8 ft scrapers, 2 tamping sticks, 2 round-point, short-axe, and powder heater. Picks and augers are any, other tools being charged for; candles and any. BREAKING GROUND (Fig 333 and 336). A ith 5 6-ft holes, 2 breast holes, 2 lifters and 1 back-are charged per breast hole, 4 sticks per lifter, and -ft auger hole takes 15 min.. Whiteside mine, in

e 46. Mining Costs, Top-slicing, Eastern Mesabi Range, Minn

	Cost per ton		
	Labor	Supplies	Total
ground:			
ing	$0.366	$0.061	$0.427
bering	0.058	0.097	0.155
mming	0.037	0.007	0.044
nping	0.013	0.015	0.028
cks	0.005	0.001	0.006
tains, bosses, etc.	0.023		0.023
ft sinking and repair	0.002	0.001	0.003
loration	0.004	0.001	0.005
eral expense	0.012		0.012
Total	$0.520	$0.183	$0.703
e:			
cking and loading	$0.013	$0.004	$0.017
sting	0.005	0.007	0.012
lding repairs	0.001	0.003	0.004
hinery repairs	0.010	0.009	0.019
cellaneous	0.009		0.009
eral office expense	0.005	0.004	0.009
Total	$0.043	$0.027	$0.070
Total cost	$0.563	$0.210	$0.773

nk for boarding up rooms costs $7.50 per M bd ft, A drift-slice 50 ft long, 2 sets wide, with 10-ft posts, s 2 000 bd ft of plank for boarding and flooring, or sumption in drift and square-set slicing is 12 to 16 ice and local conditions; amount of top lagging de-ices. FRAMING TIMBER for drift-slice sets consists flat bearing surface at ends of caps; contract price tract prices for framing square-sets (Fig 234) are: horns, 12¢; cap, 7¢. Ties for track are of 6-in bottom at cost of 2.5¢ each; they are cut in two 50 to $3 per day framing timber. CONTRACT man at so much per car of ore, or per ft of advance or per set in square-set slices. Car payments are from haulage-level chutes by company trammers. 5 to 17 cu ft; production from slices varies with of rock intrusions, length of tram and number of ence, contract prices vary widely even in parts of

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slice, or per car of ore delivered to chute. Interval between haulage levels i sometimes there are only 3 slices in a lift (290). Fig 328 (Art 71) with text gives

Low Moor mines, Va. Data from C. Dixon in 1912 (291). Shallow, vei of soft brown hematite are top-sliced; irregular and sudden changes in both are common. Country rocks are soft and cave readily. Fig 341 shows wor about 12 ft wide. Entry is by shaft in footwall or by adits. A main lev 7 ft high in the clear and 6 ft wide at top; it is timbered with 3-piece se Lagging is required on top of sets and usually on sides. Cribbed, 2-compa *b* are driven on footwall at 50-ft intervals; 1 raise is holed through to surfa as possible. Sub-levels *c* are driven from raises at 12-ft vertical intervals; t ft less in dimension all around than main level. Slicing (locally, robbing) sta sub-level, when it reaches the ore limit; back and sides of sub-level are shot with 12-ft posts erected. As work retreats, floor is covered with 8-ft la cross-sills of round timber 12 ft long, spaced 4 to 6 ft apart. Sets are blast a section 50 to 75 ft long has been robbed; then robbing can begin at end sub-level; sills are picked up on 12-ft posts in sub-level below. Drilling 2.25-in piston-drills. Sub-levels are so crooked that barrows must be use ore to raises. This method, which extracts about 95% of orebody, is open tion that all sub-levels must be driven to boundaries before robbing begins, timbering is necessary for their maintenance. Workings are hot and poor

74. Top-slicing at Copper Mines

Morenci-Metcalf district, Ariz. Data from P. B. Scotland (152, 178), (279), W. L. Tovote (280), and from notes by L. Johnson in 1915. **Orebo** carrying 2 to 4% Cu, occur as disseminated deposits in porphyry and as fi granite. Top-slicing is used for shoots of soft ore, some of which are very the Humboldt orebodies are 80 by 600 ft and 200 by 700 ft in horiz section. deposits are overlain by a leached porphyry capping of variable thickne rarely exceed 300 ft in vertical dimension. **Development.** A main hau electric or mule haulage is driven near bottom of shoot; intermediate or h levels are opened above it as required at vertical intervals of 50 to 100 ft (Fig connecting with the main level are spaced to limit the tramming distance on levels to 150 ft. Tramming levels consist of a rectangular system of drifts which determine shape of deposit and allow raises to slices to be spaced ov on corners of rectangles usually 25 by 30 or 40 ft. This may be done by r intervals along crosscuts 30 ft apart, or, where the deposit is wide, parallel

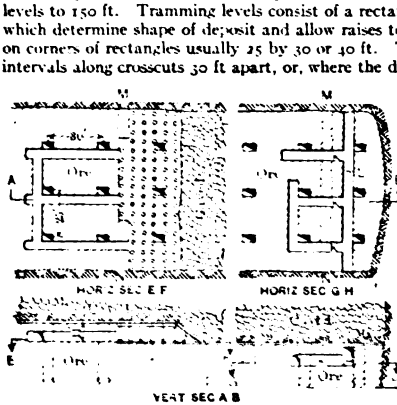

Fig 342. Top-slicing, Morenci-Metcalf District, Ariz (Diagrammatic)

driven 30 or 40 ft apa located along them. a 4 by 4 or 5 by 5-ft ch ment and a 1.5-ft ladd are spaced so that l slices may be shove chutes instead of into alined accurately to o sity for constant surve

Slicing. Irregular shoots are square-set Height of slices, 7 t nomic height, 11 ft. bered with 3-piece, u Ariz Copper Co uses and posts, both 10 ft to 9.5 in diam; Detro uses similar posts, wit cap 6 ft long; slice s 5 ft apart. Raises a shoot are connected by a crosscut *L*, timbered with slice-sets (Fig 342). necting with adjacent raises may be as in Fig 342, or else carried full heigh timbered with sets to support mat; they provide exits in case of a sudden c right side of *A* is then mined; the direction of slice-drifts here would depe

s work is completed, slicing begins at left of cross-
posit; the face is advanced by slicing 6 to 10 ft
ted lines *MN*. Section *EF* shows appearance of
Working faces on adjacent sub-levels are usually
vary widely.

of BREAKING GROUND at mines of Detroit Copper
to 6 ft deep, breaks a block of ore 11 ft high, 6 ft
s done with hammer
k are laid on several
loor is sufficient after
re 10 to 15 ft long,
s. Posts are blasted
ace being kept open
work is necessary in
oid this, Ariz Copper
hen the face has ad-
feasible, miners are
new breast is opened
viates barrow work,
n area not exceeding
lowed into the walls
he new area like that
atches of waste are
on floor. If the mat
en parallel to the old
new breast is under
lrawn. Should mat
is laid, or in case it
5 ft below; 7 or 8 ft

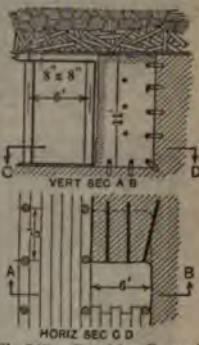

Fig 343. Breaking Ground in Slices, Detroit Copper Co, Morenci, Ariz

e overlying ore is broken with uppers, starting at
of new slice; the broken ore from each round is
he weight of mat in wide shoots is controlled by
lices are 30 to 40 ft wide; tramming levels, 55 ft

Arizona Copper Co, Morenci, Ariz

g level causes crosscuts and drifts there to crush;
er the level in a slice of double height, timbered
plan has been used by Detroit Copper Co: Ends
el are connected by a breast stope, timbered with
he back of ore remaining over the sets and below

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the mat is then blasted with long uppers; broken ore fills the sets and i allowing the mat to settle gradually on the set timbers. While one shoveled out, another is opened posts of sets in first section are and the procedure is repeated.

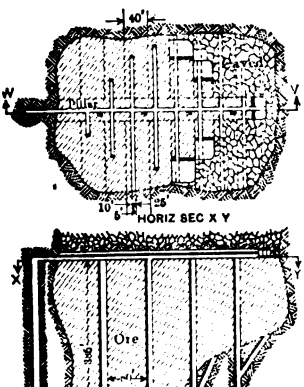

Fig 345. Panel Slicing, Arizona Copper Co (faces under attack shown thus: —)

Fig 345 shows a method devise per Co to eliminate the heavy ex and maintaining raises 25 ft a slice a central main drift pa through the oreshoot, and cross off at 40-ft intervals. The main cuts for 20 ft on each side of i with 3-piece sets, 6.5 ft high; e cuts are opened to mat above an slice-sets. Slices 40 ft wide a from the walls back toward ma a central pillar 40 ft wide whic from end of orebody after slices it are finished. Ore is shoveled trammed to raises spaced at along main drift and connectin level below. While one floor o worked, the next, 11 ft below, is This system dispenses with nee levels. Compare Fig 339.

Cost data. Ariz Copper Co. states that timber consumption including chutes, ladders and rei about 9 bd ft per ton of ore; ro costs $19 and squared Oregon fir much cheap cull lumber is use

(178). Detroit Copper Co. J. R. McLean gives following data for J 124 088 bd ft of round and 322 661 bd ft of square timber were used. details. Figures in Table 48 do not include 498 tons of ore from outs

Table 47. Timber Consumption, Top-slicing, Detroit Copp

Kind of work	Tons of ore mined	Round timber		Sq tim- ber, bd ft per ton
		Linear ft per ton	Bd ft * per ton	
Top-slicing	45 395	0.5240	2 4933	5.5022
Square-setting	1 294	0.6576	3.1236	6.3833
Development in mines	1 353	0.4205	1.9974	33 3924
Outside development	498	0.3373	1.6024	20.6104
Repairs, etc	.....	0.0146	0.0693	0.1894
Totals and averages	48 540	0.5382	2.5564	6.6473

* On basis that 1 linear ft of timber = 4.75 bd ft.

Table 48. Duty of Underground Labor, Top-slicing, Detroit C

Mine	Total output, tons	Number of men working		Outpu
		Total	Stoping	Per man
Ryerson	24 017	305	184	3.03
Yankee	8 394	182	87	1.77
Copper Mountain	12 025	169	134	2.73
Ariz Central	3 606	80	44	1 72
Totals and aver	48 042	736	449	2.54

r; 26 working days per month. Output per man-
ding outside properties, 1.85 tons. Dynamite is of

ndle Consumption, Detroit Copper Co

re d	Lb dynamite per ton	Ft fuse per ton	No caps per ton	No candles per ton
5	0.435	1.729	0.361	0.795
4	1.043	3.091	0.772	1.457
9	0.451	1.767	0.372	0.811
3	4.820	16.038	4.175	5.046
8	24.299	58.433	12.319	28.717
0	0.813	2.746	0.601	1.217

n-drill holes; latter for holes drilled by hand or by
: 49); 40% dynamite is used wherever possible;
. Blasting in
le-taped fuse.
s of ore at the
aratively dry;
rground hoist-
rs from tram-
hoists.
ng, C. deKalb
82, 283, 284).
arts of Duluth
f copper mio-
which may be
ft; ore occurs
s, 60 to 200 ft
o 900 ft thick.
rt, are opened
t 100-ft inter-
re heavy tim-
f driven long
levels are 35
varies; for a

Table 50. Distribution of Costs, Top-slicing, Detroit Copper Co

Item	Per cent of total cost
Labor	45.19
Supplies	6.11
Timber	17.70
Tool sharpening	2.25
Machine-drill reps	2.97
Ventilation	0.52
Und'g'd hoisting	0.28
Hand tramming	2.64
Mule tramming	5.45
Hoisting	5.85
Und'g'd repairs	3.51
Mine drainage	0.70
Miscellaneous	6.83
	100.00

50-ft intermediate, there may be a series of drifts following sides of 50-ft squares, with raises to level above at middle of sides of each square. Fig 346 shows a 35-ft intermediate level in Oversight mine, where the exploratory function of drifts was emphasized and a regular arrangement of raises sacrificed. Fig 347 shows a sub-level (comprising drifts 6 ft high) driven in slices over the intermediate level of Fig 346. Raises have 2 5 by 5-ft compartments, timbered as in Fig 348.

der the capping are
floor begins at end
no sub-level drift is
n from nearest raise
right angles to this,
nilar workings from
ays driven to raises
exits in solid ore in
. Ore is handled to
els it is trammed to
level below. Slices
features of work. Flooring in each slice is of 2 by
-in by 10-ft sills, spaced 5 ft centers, and set flush

Fig 347. Sub-level in Connection with Fig 346

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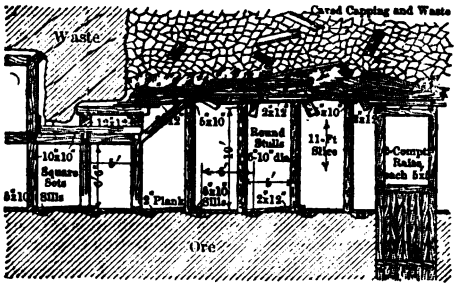
with floor of slice. Ends of floor-plank butt on center line of sill, or are lap
where sills are not used. Floor is kept close to working face, unfloored area
ing 10 ft in width. Mat is supported by vertical stulls, 5 ft apart, under sills c
Feet of stulls are on ore; if mounted on foot-blocks or on the flooring they m
latter when the mat caves. Much cheap, cull timber is used; no salvage
Mat may be allowed to cave naturally, or be shot down. In latter case, 1-in h



VERT CROSS-SEC
THROUGH SEVERAL SLICE-DRIFTS

Fig 348. Top-slicing, Cananea, Mex

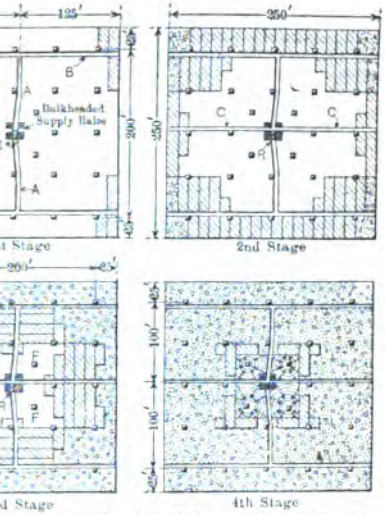
are bored by
in each post
with 0.5 stick
namite. Top
covered with
with 2-in plan
ing the mat al
faces on suc
are kept at lea
Ore often ext
walls beyond
above; such
sliced by squa
the left of Fig

Timber co
10 to 13 bd
ore; about 1
mite is used
Mar, 1909, O
produced 25
total cost of $1.90 per ton. Of this, surface and underground labor (most
cost 97¢; lumber and timber, 35¢; explosives, 18¢; steam and electric pow

Cumberland-Ely mine, Ely, Nev (281). OREBODY is a flat-lying porp
deposit (Art 11), 40 to 100 ft thick, of considerable area, and overlain by lea
about 300 ft thick. Assay, about 3.5% Cu. Ore and capping are weak; i
capping runs like sand. Top-slicing was preferred to other methods to sec
traction and clean mining. DEVELOPMENT. A vertical shaft was sunk t
orebody, where a main level was opened by driving 7 by 8-ft drifts along the
squares. Drifts are large, for mule haulage; they are timbered and cost $5
Close-cribbed 5 by 7-ft raises, with a 20-in ladderway, were driven to cappi
of each 50-ft square; raises slope 75° to prevent moist ore from packing in
cost about $5 per ft. Sub-levels are opened by drifts and crosscuts to co

Slicing. First 3 floors at top of orebody are 10 ft high; below that, 17 ft
slices are timbered with 3-piece sets; posts are 9 to 12 in diam by 8 ft long;
by 12, with a 7-ft spreader of 2-in plank, nailed on under side. Blocking and
are used to pick up mat. 17-ft slices and high backs at top of orebody are ti
2 tiers of sets, one above the other. Each slice is floored with 2 layers of 2
placed at right angles. Contiguous slices are driven until timbers show d
posts are shot down in groups of 25 to 40, 0.5 stick of 40% dynamite being u
At least 1 slice is kept open between unmined ore and the sets which are b
Tramming in slices is done in 12-cu ft cars. Slicing is contracted at 65¢ per
all men working from a raise share in the contract, and furnish their powde
and candles. Drilling is done single-hand or with augers. Each raise has
nected to a suction blower. In 1909, 400 men underground, working 2 shif
1 500 tons per day; output per man-shift, about 3.75 tons.

Miami Copper Co, Miami, Ariz. Data from E. G. Deane in 1916 (285
is a large disseminated deposit of chalcocite in schist, overlain by a leached ca
200 ft thick (Art 11). Top-slicing is employed in an area about 800 ft squa
the ore is soft but considerably harder than the siliceous capping, which bre
particles and runs like sand. These conditions, and the fact that the ore in
is richer than usual, led to the adoption of top-slicing, which is partly preparat
methods for the lower lifts (see Art 87 for the mining system generally emplo
VELOPMENT. Haulage levels are 150 ft apart vertically, with 2 sub-levels betw
intervals, which facilitate building of chutes, and, in connection with artificial



ORK ON 1 FLOOR AT DIFFERENT STAGES

Block Method of Top-slicing, Miami, Ariz

ation sets of 12 by 12 timber with 9 or 10-ft posts her protected by timber bulkheads 10 ft high; two bulkheads are 7 by 7 ft and two 7 by 11 ft in plan, as shown. Drifts *A*, *B*, and *C*, usually untimbered at first, are run in the order named. Slicing begins at ends of *B* and *C*; Fig 349 shows 4 successive stages of work in same block. New slices are started as soon as those alongside have advanced a few ft. As many men as can work to advantage are put on; ore is mined with greatest possible speed, until only 4 central pillars around the supply raise remain (see 3rd stage). Inclined raises *F* are then put up to about the middle of each pillar; crosscuts (not shown) are driven to raises *F* from drifts *A* and *C*; slicing then continues, working from outside of pillars back to bulkheads, which by this time have squeezed down to a height of 4 to 6 ft. On completing this work, the last of the stope is caved; it is found best to let ground settle several weeks before starting another floor. Slices are 10 ft high, timbered, according to the ground, either with single sets, of 2 8-ft posts and a 12-ft cap, or with double sets, of 3 8-ft posts and 2 7-ft caps, all round timber. No floor is laid if mat is thick enough to prevent runs of capping; elsewhere a 2-in floor is spiked to 2 by 10-in sills; 5 by 10 and 4 by 8

od of o

Cost per ton
$0.035
0.025
0.080
$0.140
$0.080
0.160
0.030
0.040
0.070
0.130
0.040
$0.550
$0.055
0.040
0.005
0.025
0.015
0.016
0.004
0.030
$0.880

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sills were tried, but seemed to be no stronger than 2 by 10-in, after subje and pressure of a completed slice. Bulkheads of old timber are built wi aid slice sets; the posts are shot down as soon as possible. All drillin drills, using a water spray. Ore in slices is shoveled to raises or handled i method has given good results; weight on timbers does not get beyond c time that slices must be kept open. It has been found that pressure on not intensified by caving around them. But ore in pillars is fractured, the only holes needed to break it. This fact, combined with the handli vided by raises *F*, makes pillar-ore cost less than that from outside slice of the blocks is important; without it the heat from mat is excessive ciency of labor. Air from a 60 000-cu ft fan is taken to the sub-level floor and forced through raises into the slices as desired. Costs. As sh the production per shoveler is 20 tons and per total man-shift, 10 tons, t about double those obtained in slicing on long faces without artificial ve

Bingham, Utah. Data from D. W. Jessup in 1914 (66). Top-slicing mining lenticular replacement deposits of pyrite, galena, sphalerite, a occurring wholly in limestone or along contacts with a quartzite footwall. soft; hanging wall is very heavy; lenses vary from a pinch to 100 ft about 65° and may be several hundred ft long. Some mines use square cases, top-slicing was adopted after square-set stopes had collapsed or pr because of excessive reinforcement required. Distinctive feature of to here is the use of square-sets for slices, locally called the SLASH SYSTEM. of a typical orebody is shown in Fig 95, Art 15. Entry is by tunnels or level is driven in footwall at bottom of lens, with crosscuts underneath o at intervals of 75 to 100 ft. Similar levels, 100 ft apart, are opened abov by Fig 95, the longitudinal drifts on levels No 1 and 2 were driven in o lens. Square-set, 2-compartment raises are put up between levels an they are best located in ore, but the shape of the deposits requires much Raises are kept away from heavy ground on hanging wall and are offset to break the fall of ore, prevent its packing in chutes and reduce wear c Raises in ore are about 50 ft apart; when mostly in waste, they are 100 f

Slicing. Irregular tops of lenses are mined by square-sets (Art 71). run from tops of raises to walls; slice-drifts 5 to 6 ft wide are turned of starting either at walls or midway between raises and retreating towa enough ground for working places is kept open. Both longitudinal an square-setted, with 8 by 8 timber; joint-framing is like that in Fig 212 (heavy ground, posts are 7 ft long and are set on 5-ft centers capwise a where, sets are 6 ft square and posts 8 ft long. Posts are flat-botto square horn on top; no sills are used, but feet of posts are braced apar 2-in plank. Depending on size of sets, slices are 8 or 9 ft high, leavin blocking between caps and mat. Floors and lagging are of 2-in plank often breaks floors; sets underneath such areas are lagged on top; side l along walls, and often elsewhere, to prevent mixing of ore and waste. maintained for connecting different working faces; sets here require mu

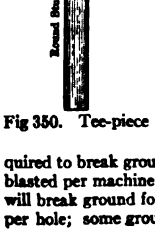
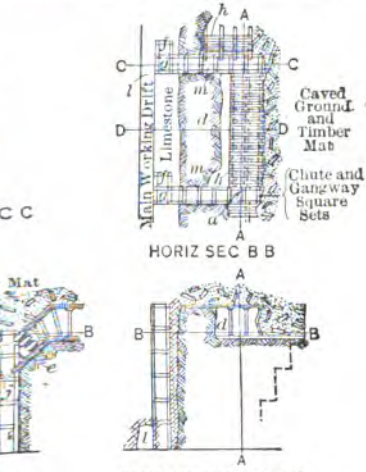


Fig 350. Tee-piece

(Art 52), for which cheap round timber is used. F burden through mat often occur; they are stoppe spiling (forepoling) through them; for detail, see Bi Art 19. T-piece sets (Fig 350) are used in irregular s widths, and instead of square-sets where mat is thi or half angle-sets (Fig 246, 247) are placed under he Records from one stope showed: 39% of total tim for posts, caps, and girts; 5% was 6 by 8-in for 38% was 2 by 12-in plank for flooring and lagging timber (minimum 6 in diam) for doubling-up posts Holes for blasting posts are drilled 5 in deep with a charged with 0.5 stick dynamite. BREAKING GR 2.25-in piston-drills are used; 6 to 12 holes, 4 to 4 quired to break ground for a lead set (Art 45). In soft ore, 1 round per s blasted per machine; some ground requires 2 rounds to make room for a will break ground for a side set. Normally, 2 to 4 sticks of 1-in, 40% d per hole; some ground takes 6 sticks. HANDLING ORE (Art 71). Fro

Horses of waste are left unmined where possible; waste floor. VENTILATION. Limestone hanging wall slakes and adding to ventilation troubles common to all top- circulation can be maintained by connecting several pressure fans have been used for isolated stopes; the vanized iron pipe connects fan with stope.

sbee, Ariz. Data from P. D. Wilson and R. H. Dickson n of top-slicing has been devised here for flat, roughly limestone and clay impregnated with enough CuO and



Mitchell Top-slicing Method

10% Cu. Some deposits are 50 ft wide, 50 ft high and rock is limestone. Ground is so heavy that cost of licing was tried and found fairly successful; present G METHOD, eliminates much of the shoveling incident PLAN (Fig 351, 352). Ore is mined in blocks 45 ft Adjacent blocks are separated from each other by de (gangway sets), started at top of deposit and ex- ks are sliced by running successive contiguous drifts , each slice-drift is driven on an upward slope of 35° a rried horizontally to center of block, where it meets a pposite side. Broken ore slides on the sloping floor to by slides (Art 47) to chutes connecting with a haulage haulage drift *l* (Fig 351) is driven near bottom of n, and preferably in wall rock, as shown. Permanent, s are put up to top of deposit at 45-ft centers along is a chute, *g* a timber and manway. From the top

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sets of chute in each raise 2 floors of gangway-sets *m* are carried across th a third is started. Besides the above work, crosscuts are sometimes drive tervals from the haulage drift, and raises put up from them on 40-ft cent

Slicing. Fig 351 shows slicing after work is well started; Fig 352 is a timbering in successive slices. Top of orebody is mined on square-sets, st.a wall and retreating toward raises; posts of sets rest on ore without sills As work progresses, a 2-in floor is laid on 4 by 10 sills *a*; posts are shot dc least 2 open sets next to Usually 1 floor of square-sets even off the stope back; so back does not cave promptly filled to form a cushion ov Ground along gangway-sets square-set slice is cut back and a floor laid there on slopi long; the ends of horiz sills to fit under sills *b*; both sill pieces of 2-in plank placed joint. A second slice may b of block, as soon as the cave comes to rest. Work begi gangway-set; a slice-drift, 8 6 ft wide, is driven on a 45° sl to lay 11-ft angle sills *b*; t carried horizontally to the c Sills of floor above are picke

Fig 352. Mitchell Top-slicing. Diagram of Timbering (letters refer to Fig 351)

6 to 8 in diam. This slice is completed by driving contiguous drifts; usual space is kept open next to the working face; stulls in third row back from begin to crush as the face is being drilled (Sec *DD*, Fig 351); stulls are s necessary. This procedure is repeated for lower slices, which vary in heigh of angle-sills used (Fig 352). In each slice, a 2-in floor is laid on 4 by 10 sills Max feasible height of slice, 10.5 ft, is not attempted until the fourth slice the mat is under perfect control. High slices reduce shoveling, as the slides in them can be made longer. Bulkheads *e* take the weight of mat off the gangway-sets, the posts of which are cut below bulkheads at *k*. Ordinarily, bulkheads built in second slice and in every third slice below will prevent gangway-sets from crushing; in other slices, sets are braced with stulls (Fig 352). Enough partly crushed timber comes from the mat to build bulkheads. BREAKING GROUND. Holes, made by 2.25-in piston-drills on columns, are usually located as in Fig 352. Weight of mat aids breaking, and cut-holes and lifters generally suffice. It is rarely necessary to set up drills in inclined parts of the slice-drifts; this ground is broken by drills mounted in gangway-sets and at top of the slide. HANDLING ORE. From 67 to 80% of the ore broken in a slice falls on sloping floors, or is thrown there by blastin gangway-sets; the rest is shoveled. As slicing proceeds downward, the gangway-sets are extended, but they cannot be opened far in advance of slic weight on them is considerable even when bulkheads are used. Grizzlies gangway sets at foot of the slice-slides; below these, 2-in plank slides *i* are la Fig 351). For the upper slices, slides *i* connect with chute compartment *g* chute-gates are erected in sill-floor gangway-sets for loading cars (section *A A* ore extends below the haulageway, raises are put up from a lower level and

Table 52. Comparative Cos and Ordinary Top-slicing Calumet & Ariz Co, E

	Cos	
	Mitche metho	
Labor	$0 54	
Explosives	0 04	
Timber	0 21	
Candles or carbide	0 01	
Air	0 10	
Other supplies	0 03	
Total	$0 93	

* Costs refer to mining in heav

Footnotes of Table 43, Art 56, a

sorting may be done in slices, the waste being thrown
licing is not adapted to careful sorting. LIMITATIONS
er forms of top-slicing (Art 75); max economic height
ut 100 ft, max economic length, 80 to 100 ft; longer
In latter case, a vertical face of ore should not be
pe; in the first block taken, each slice should be started
slice above (see dotted lines, section *DD*, Fig 351);
eak off the end wall as the mat descends alongside; a
jacent block. The method is adapted to heavy uni-
its chief advantage. As work in adjacent blocks is
ocks must be mined simultaneously. COSTS of this
ary top-slicing, are given in Table 52 (220). See Art
nd other methods at same mine. OUTPUT per man-
lucing about 125 tons per day.

mmary of Top-slicing

ially a method for mining deposits with weak
be used under many different conditions (Art
is in orebodies of large horiz section, overlain
It is well adapted to easily broken, heavy
g and filling if mined overhand (132, 286).

ms of top-slicing are: (*a*) safety; chief danger
(*b*) clean mining; (*c*) high extraction (90 to
g during mining; this is especially important
its.

mon to all caving methods (Art 79, 83). Also,
ventilation is poor, and, in ordinary top-slicing,

application are: (*a*) Subsidence of overlying
able property nor make mining dangerous; see
conditions. (*b*) The capping, if of rock, must
the surface or sufficiently to cover the mat with
enough to protect the workings below. (*c*)
small amounts of waste may be left on floors,
e. Large horses of waste may be left unmined,
must be used to form a mat below the horses,
rt 71). (*d*) Ore should break easily, since the
of narrow work increases rapidly with hardness
sts required by hard ground dislodge timbers
(*e*) Boundaries of orebody should be regular;
p-slicing, but increase cost, as they usually
Art 74). (*f*) Top-slicing is best adapted to
osition of which allow stopes with vertical or
le arises in vein-like deposits dipping less than
ist be caved and the mat extended under it on
ase with the strength of hanging and flatness of
-slicing in a bed 36 ft thick, dipping 9° to 12°.

g has been substituted for filled square-sets (Art 45)
o support the surface or ground containing unworked
following comparative advantages (see also Art 55):
heavy for economical square-set work. (*b*) In very
uire much reinforcement, it saves timber; elsewhere
ber than square-sets. But, cheap grades of timber
er framing, and its erection in stopes takes less time
d) Rich fines are recovered in slice below; some fines

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are always lost in filled square-set stopes (Art 47). (*e*) More unskilled l employed; more shovelers and fewer timbermen are needed. (*f*) Stoping m porarily, with a lower cost for renewing or reinforcing timbers.

Relative disadvantages of top-slicing, besides the general disadvantages s are: (*a*) it is less flexible; (*b*) ground is broken by breast stoping, which more than overhand work; (*c*) a much greater footage of drifts, raises, etc. is a given block of ore, and this delays production from any area; (*d*) mining i top of orebody; stopes cannot be opened on lower levels; hence, top-slicing of small area may not produce the desired output; (*e*) cost of bringing tim stopes is usually higher in top-slicing than in square-setting (Art 47). Sub- is the usual alternative caving method (for comparison, see Art 79).

76. Sub-level Caving

General. Sub-level caving, a logical development of top-slicing (Art 71 used on the Lake Superior Iron Ranges, but rarely elsewhere in the U S. terms are: SUB-DRIFT CAVING, SUB-LEVEL SLICING, SUB-SLICING, SUBBING, S CAVING and TOP-SLICING WITH PARTIAL ORE-CAVING (see Art 71).

Suitable orebodies are wide deposits of moderately soft or mode ore, overlain by ground which will cave readily. For limitations an ments of the method, see Art 79.

General plan. As shown by Fig 353, the method resembles top-slic a timber mat forms over the workings, and ore is mined in horiz slice descending order. MAIN LEVELS consist of drifts *D* and crosscuts *C*, planned to facilitate haulage and for proper spacing of raises *R*. SUB-LEVELS ("subs"), opened from raises as needed, are usually 15 to 18 ft apart vertically, and consist of timbered drifts *S* and crosscuts *T*, cutting the ore into pillars (Fig 360, 363, Art 77). This development work leaves pillars *P*, 7 to 9 ft thick, between the back of one sub-level and floor of that above.

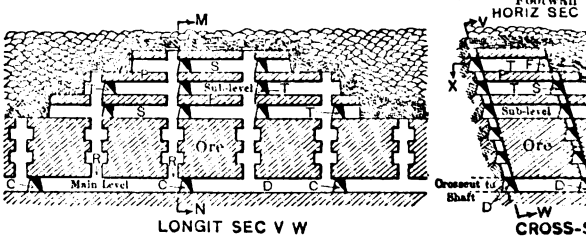

Fig 353. Sub-level Caving in Wide Vein (diagrammatic)

Mining begins on the highest sub-level, the end pillars being attac work retreats toward some central point, as in the longit sec *VW*, Work on individual pillars may begin at the hanging wall and retrea footwall (cross-sec *MN*), or may retreat from both hanging and foot t middle of vein, or may retreat in a direction parallel to the strike. In a slice-drift or slice *FF* is driven from crosscut *T*, next to caved ground way across adjacent pillars; it is timbered with drift-sets and floo lagging or plank. The overlying ore is caved into the slice-drift in smal from points *F* (horiz sec *XY*), and retreats to crosscut *T*. Ore is sho barrows or small cars and trammed to raises. Contiguous slices are su driven and caved until the footwall is reached. The caving of ore o

-358 show details of different modes of proce-
e not recovered; as the work descends, a mat
accumulates over the workings. Gob is con-
heavy pressure which breaks the ore above
of and after being well formed it hangs over
r the safe removal of caved ore. Slicing and
usly on several sub-levels, by keeping working
ce of one another (Fig 353). Main levels are
hrough raises to the next lower main level.

be adapted to orebodies of various shapes and sizes
modifying the layout of subs and main levels, the
of slicing and stripping (see below; also Art 77, 78).

nder Entry, Art 71). INTERVAL BETWEEN
, depends upon same factors as in top-slicing
of these factors is different because: (*a*) ores
ally harder than those worked by top-slicing;
ore is less in sub-level caving, less timber has
cost per ton of ore for hoisting timber to the
greater level intervals are allowable in sub-
ain haulageways may be increased by employ-
slicing.

vels were 200 ft apart vertically, because of cost of
fficult drainage conditions (153). At some places,
apart; at others, raises were offset at 50-ft intervals
cessive wear in chutes (Fig 364. See also Fig 368.
is mine favors mechanical haulage. Oblique cross-
(arrows show movement of empty and loaded cars).
only the crosscuts *A* and *B* contain trolley wires.
is that some diamond-shaped pillars are formed on

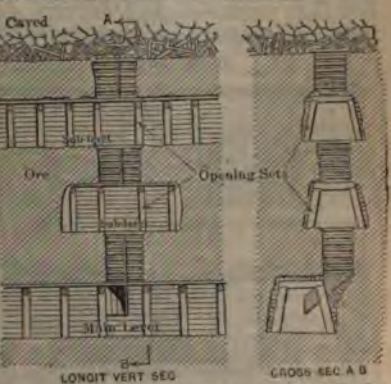

Fig 354. Timbering for Raises, Sub-level Caving

suitable for slicing and stripping, or into pillars
natically for this purpose by subordinate drifts
ally close-cribbed: drift-sets ("opening-sets"),
ng opposite each sub-level as the raise goes up;
as needed. Chute and ladderway compartments
See Montreal mine (Art 77) for precautions to

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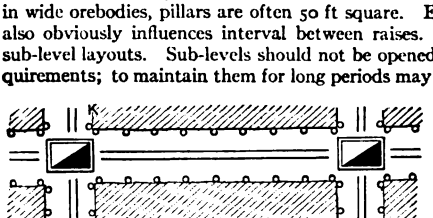
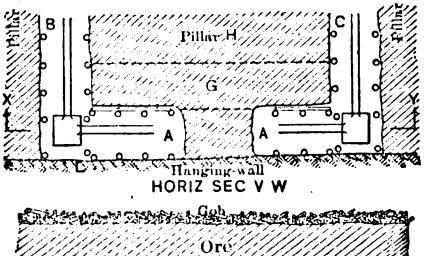
prevent men from falling down raises. Raises are ordinarily about
see Art 77 for arrangements in orebodies of different sizes. Factors g
raise interval are like those for top-slicing (Art 71); see also below
levels. Practice tends to reduce the distance between raises, in orde
the amount of ore which can be shoveled directly to them (270).

Sub-levels. (*a*) Plan of main levels governs the location of raise
the position of the principal sub-level openings; (*b*) to facilitate slici
drifts and crosscuts should intersect at right angles, the pillars thus f
rectangular; (*c*) max length of slice-drift for economical driving and
about 50 ft; hence pillars on sub-levels should have one side not exc
in wide orebodies, pillars are often 50 ft square. Economic length
also obviously influences interval between raises. See Art 77, 78
sub-level layouts. Sub-levels should not be opened far in advance
quirements; to maintain them for long periods may require extensive
timbering. Ne
can generally b
quickly, as ra
numerous point
but, when the o
level developme
to maintain a
put, the subs ar
as they are re
raises (76, 270)
a sub-level are
crushed by pre
by mining on th
especially whe
adjacent subs
over one anothe
being staggered
INTERVAL BET
LEVELS is 12 t
given orebody
promise betwee
ing factors: (
interval increas
portion of ore
caving and he
the aver cost
ore for the wh
(*b*) small inte
closer control,
ing, and higher
(*c*) maximum p
val is a function
of the ore and
the gob. Depe
relative weight





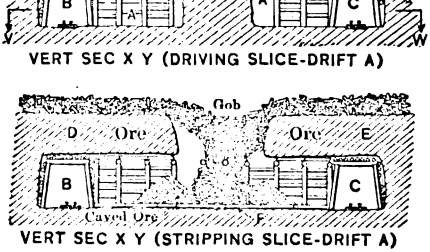


Fig 355. Slicing and Stripping Details

tors, the normal interval of 15 to 18 ft between subs is increased wh
and decreased if necessary.

Methods of slicing and stripping pillars formed by sub-level d
vary with character of ore, pressure from caved ground, and thickness a
of mat; local custom also influences the choice (270, 76, 293, 153).

r GROUND which, in narrow excavations, can be
ce-drift *A* is driven across pillar *H*, and timbered
loor *F* of lagging or old timber is laid. Stripping
s at center of pillar and allowing the ore to run

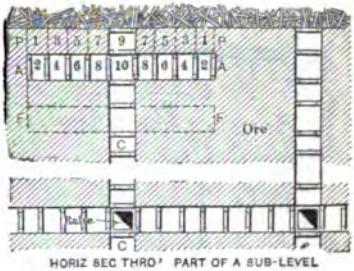


Fig 356. Slicing and Stripping Details

nd caved, slice-drifts in pillar *H* could be driven
; stripping would then begin at inner end of slice.
ce-sets holds back the gob while slice-drifts are
ws a method permitting CLOSE CONTROL of both
eral and vertical pressure. A slice-drift *AA* is
ven as shown, leaving a 6 or 8-ft pillar of ore *PP*
t to gob. Work of removing pillar and stripping
slice begins at *A* and proceeds as follows: cross-
s 1 are driven through pillar *PP* and ore over 1 is
ved; ore over 2 is then caved, and crosscuts 3 are
ven and stripped; then ore over 4 is caved and
on. The back of sub-level crosscut *C*, at 10, is
ipped last; the next slice-drift is driven at *FF* and
ocess is repeated. On the GOGEBIC RANGE, Mich,
lars only 3 ft thick are sometimes left between
ce-drifts; they are drawn without system when
drift is stripped (297).

Fig 357 shows work in STRONGER GROUND under a
ong mat. Drift *A* is a sub-level at end of ore-
dy; slice-drifts *B* and *C* are driven in turn along-
e of it. By the time slice *B* is completed, or
ssibly not until slice *C* is started, timbers in *A*
gin to crush. If ore overlying *A* does not come
wn, timbers at far end of *A* are shot out, and cav-
g is begun there by blasting down the back with
ht charges. The ideal condition is to have caving
rt at end of *A* and progress steadily back to cross-
t *D*; to accomplish this, some slice-sets may be
nforced and others shot down; similarly, some
rts of the back may require blasting and others
t. Ore caving into slice *A* is loaded by men who
rk mostly under protection of timbers in *B*.
oken timbers are thrown back and become part
the mat; slightly damaged timber may be used
reinforcing weak sets. Vert sec *LM*, from B. W.
going on in slice *B*. A thick mat will often hang
illiams states that, on the Gogebic Range, Mich,
(270). Obviously, details may be greatly varied.
be mined out systematically, drifts are driven to
as it will run. Some of the ore which can not thus
b-level below; such work is necessarily irregular.

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Sometimes work is planned so that the weight of gob will crush part of th level pillars. Fig 358 shows this method at CHANDLER mine, Ely, Minn (s description of orebody). Stripping on any sub-level retreats from the ends o Drifts *D* cut ore on the sub-levels into pillars say 10 ft square. Weight ground above breaks down ore in the back, and splinters the pillars so that mined with pick and bar, without blasting. Where necessary, long props boards support the rarily, as at *C*. I contiguous slice-drif as in Fig 357 (287).

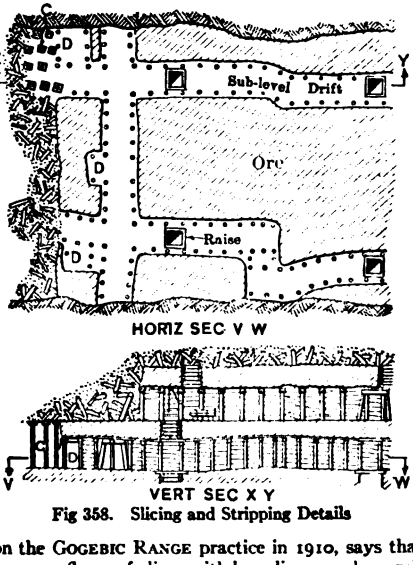

Fig 358. Slicing and Stripping Details

Breaking grou drifts is similar to slicing (Art 71), there is no free fac slicers like those in driven as ordinary 21). Normally, f 60% of the ore p sub-level caving is pressure of the c burden, without us sives.

Timbering. C is used for same re top-slicing. Simp 3-piece sets are ne employed for slices 53 and examples in When slicing und mat, FLOORS are oft P. S. Williams, c on the GOGEBIC RANGE practice in 1910, says that there is a growin to cover floors of slices with boards, even where gob is compact enough good extraction without them. Use of floor-boards sets a definite lin to work to in stripping and helps to prevent mixing waste with ore (27 LING TIMBER. Much timber used in slicing must be hoisted up raises slicing. Where sub-levels are developed in advance of mining require bers for sub-level drifts and crosscuts can be lowered from main l When a pillar of ore is left opposite a hoisting shaft in the footwall, be maintained in the pillar through which timber for both development is lowered to sub-levels(270). SALVAGE OF TIMBER is not feasible.

Handling ore. All ore broken must be shoveled. Barrows, or c 0.5 to 1.5 ton, are used on sub-levels. Some direct shoveling to raises (see Art 71 for comparison). Small amounts of waste can be sorted o in the slices; floors should be laid where this is done, otherwise, the s will appear when slice below is stripped. Extensive sorting is not fe

Mat is started at top of orebody by one of the methods described in Art Fig 361, Art 77, and accompanying text.

Ventilation problems in sub-level caving are like those in top-slicing. mining, powder smoke, and the heat and gases from decaying timber in gob, mu from the working places. Doors and brattices (Sec 14, Art 6) are used to con air currents; working places which can not be thus ventilated have motor-driv fans. Sub-levels are kept connected with main level above as long as possibl tate handling timber; these connections and those with the timber-raises in

full discussion of ventilating problems in Lake

the gob, especially in working near the top of duced by using floor boards; in any case, it has been formed (20). Pillars which crush ly recovered; such caving may cause further illars. Different authorities estimate that 5 el caving; no accurate data are available.

on Lake Superior Iron Ranges

Art 11; also Sec 2, Art 18 and accompanying Bib. e (Art 72). From standpoint of mining methods, on on other ranges, may be considered as wide, sses may be placed in the same category. Bot- of ore, generally pitching at small angles. Few dies are available. Ore varies from soft, earthy airing heavy blasting; hard ores which are much rocks of some orebodies are strong and hard; of METHODS. Sub-level caving is widely used for al development and timbering practice in different g, where conditions require or allow them, see 84 *et seq.*

Details, Lake Superior Iron Ranges

	Interval in ft between			Size of raises, ft
	Main levels	Sub-levels	Raises (*b*)	
	100(*a*)	15–16	50	4 × 8–5 × 8
	75–100	16–20	35–50	4 × 7–5 × 8
	200	15	50	5 × 8
	20–75(*b*)	20–12.5(*b*)	50	.....

	Sub-level sets			
s	Posts		Caps	
Diam	Length	Diam	Length	Diam
10–12 in	8 ft	8–10 in	8 ft	8–10 in
12–18 in	7 ft	8–12 in	7 ft	8–12 in
.....	6 ft	.....	6 ft	.....

der workings, to 200 ft in some recent practice. elow.

ebodies vary in width from a pinch to 300 ft or hs of over 2 400 ft. In soft orebodies, all drilling ed varies from 5 to 14 in per min; 5-ft holes are s taking 1 to 2 hr. Development work in country in levels, drift-sets are 4.5 to 6 ft apart, and are sets are 4 to 5 ft apart, also close-lagged. Raises y of 6-in round timber; opening sets (Fig 354), placed in ends of raises opposite sub-levels.

ata from B. W. Vallat in 1911 (76). Entry is by l to footwall dipping 68°. Fig 359 shows usual , drifts and crosscuts are driven on sides of 100-ft variations. Vertical raises are 50 ft apart along ft apart, beginning 18 ft above main level, where eways. Sub-levels are opened by connecting the 00-ft pillars so formed are subdivided into 50-ft area (Fig 360).

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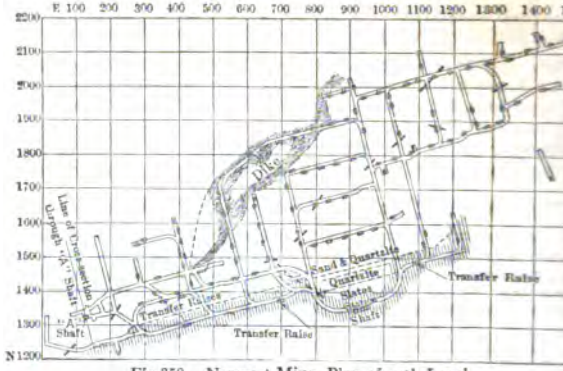

Fig 359. Newport Mine, Plan of 17th Level

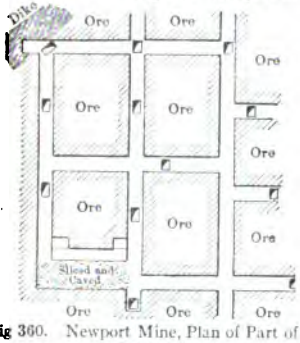

Fig 360. Newport Mine, Plan of Part of a Sub-level

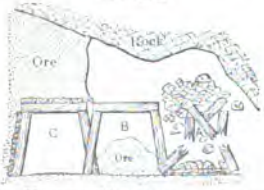

Fig 361. Newport Mine, Slicing and Stripping under Capping

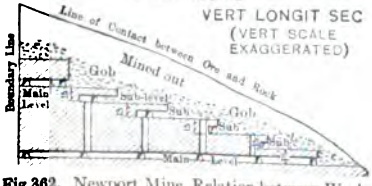

Fig 362. Newport Mine. Relation between Work on Sub-levels

Fig 363. Montreal Mine, S

sections 300 to 400 ft long by full width of deposit.
op of orebody pitches eastward. The first work on
; an area 300 ft long is subdivided into 50-ft pillars
and *B* (Fig 361) are driven in succession across a
king under protection of the timbers in *B* drill short
ean without disturbing the capping; the sets are left
then covered with old lagging, blocking, etc, which,
which the unsupported capping keeps shelling off.
s taken down, and so on over the whole area of sub-
o the west on same sub-level and 300 ft long to the
r sub-level are developed (Fig 362). Slicing and
357. On sub-levels, ore which can not be shoveled
buggies") on 8-lb rails, with turn-sheets at inter-
mine produced 1 074 800 tons in 307 days, using
0049 cord of lagging per ton.

G. E. Des Rochers in 1913 (292). Level interval,
(Fig 363). Where deposit is less than 50 ft wide,
, with crosscuts to walls every 50 ft; on greater
from *D*. Raises are 50 ft apart, as shown; those
sub, so that they will connect through at the side
falling into raises, the chute compartment is covered
the ladderway has a trap-door at each sub-level.
ing footwall raises, and crosscuts at 50-ft intervals to
ceeds 50 ft, a 2nd drift cuts the ore into pillars 50 ft
level starts at the hanging and retreats to footwall.
g and stripping. 1-ton cars are used on sub-levels.

worked on same general lines, with some variations
ain levels consist of parallel drifts 35 ft apart, with
part; sub-level drifts and crosscuts divide ore into
ams on sub-levels, allows more ore to be shoveled
of curves and crossings on main-level tracks. In
in Fig 356. Hardwood 2 by 4-in stringers are often

om H. H. Stoek, in 1909 (293). Entry is by vertical
ably the former. Table 53 gives details. Main
g footwall or near middle of deposit. Main-level

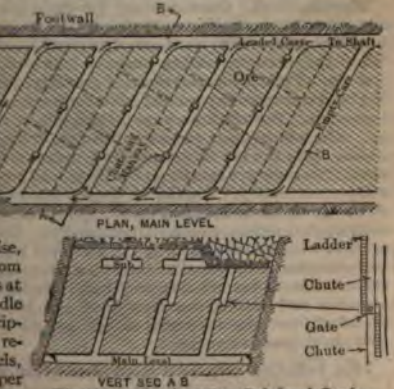

Fig 364. Chapin Mine, Sub-level Caving

Mich. Data from A. H. Fay, in 1909 (294). Ore-
2 000 ft long, and 20 to 140 ft wide; dip, about 70°.

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Walls near ore are soft; ore requires machine-drills for driving developmen (Table 53). Fig 364 shows main level development (see remarks on oblique 76). Raises, about 50 ft apart and incline deposit, are put up from crosscuts; they are have a chute-gate every 50 ft vertically. The is thus broken, so that men need not descend 50 ft during blasting in stopes. A break in partment is desirable because of large level i also allows one grade of ore to be dumped part of raise, while another grade is being below, thus avoiding delays. Sub-levels are the first being 18 ft above main level. Ore i sections, rarely more than 50 ft square, around (see dotted lines in the plan, Fig 364). Work and stripping any block is shown in Fig 365; indicate order in which slice-drifts are drive show direction of work. Ore over slices 1 and as slice 2 retreats; back of slices 3 and 4 is cav 4 retreat. Cars are 0.5-ton capacity. Where mat is thick, no floors are laid in Output from 900 to 950 men is 3 500 tons per day, or 3.7 to 3.9 tons per man-s

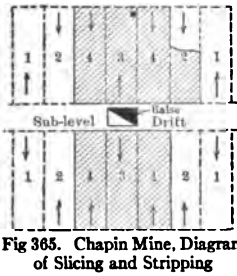


Fig 365. Chapin Mine, Diagram of Slicing and Stripping

Chandler mine, Vermilion Range, Minn (153, 183, 287). Orebody occurs dipping trough (Fig 92, Art 15) capped by jasper. Ore is hard hematite, though and broken, and machine-drills are often needed in driving main levels. Fig mode of entry and position of levels and subs (see also Table 53). Down to main levels were 50 to 75 ft apart; sub-levels, 12.5 to 15 ft apart. Ore in the trough was about 70 ft thick; main levels usually comprised 2 drifts, near the connected at intervals by crosscuts. Raises were spaced 50 ft apart along dri levels were opened, sliced and stripped, as described in text accompanying Fig 3 Sets in sub- and main-levels were 3 to 4 ft apart; top lagging was used in all dr side lagging was needed. No floors were laid in slices. Barrows or small cars were used on sub-levels. Below the 8th level, shaft stations were cut at 20- intervals; intermediate levels were of same size as main levels (Fig 92). Ore o termediate level was cut into pillars by drifts and crosscuts, and mined as on up This saved many raises and allowed greater use of cars instead of barrows.

78. Sub-level Caving in Utah and Arizona

Mercur, Utah. OREBODIES are low-grade gold and silver-bearing replacement deposits in tough, hard limestone, underlying sheets of porphyry which follow the limestone bedding planes on both strike and dip. There are several sheets of porphyry, and in places corresponding parallel orebodies overlying each other. Deposits are 4 to 70 ft thick; dip, 0° to 30°. Ore is usually soft, sometimes like clay; drilling is done single-hand or with augers; 30% dynamite is used. The porphyry hanging wall is blocky and generally caves readily. Practice of CONSOL MERCUR GOLD MINES CO is described below; this company suspended operations in 1913. DEVELOPMENT. Entry is by a compound shaft (Art 17), the lower part being vertical; at each level a crosscut is run through the orebodies, with a drift along each body. Inclines are often sunk in ore from main levels and drifts turned off from them alternately to right and left at 50 ft apart, measured on dip.

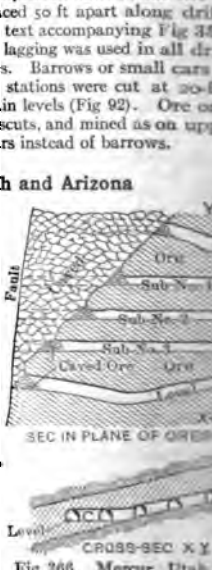
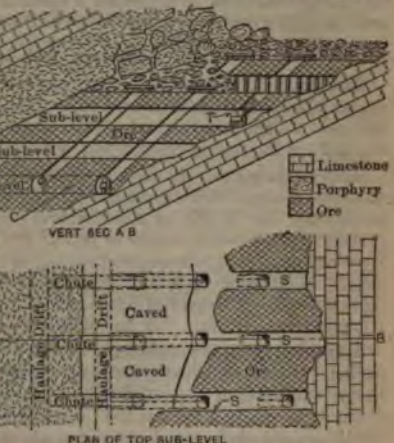


Fig 366. Mercur, Utah. level Caving

Thin beds (less than 20 ft thick; description by G. H. Dern, Bib, 295). Fig 366 shows part of an oreshoot cut off on one side by a fault. Crosscut *C* is driven to footwall and follows the footwall to level above. Sub-levels, 4 by 6-ft, driven on footwall to at 15-ft intervals, are timbered with 3-piece sets and lagged. An area thus

months, so that pillars between subs will crush and eriod, sub-levels are kept open by easing timbers fault on sub No 1, by pulling out a set of timber; as and trammed to raise *R*, or to special chutes provided required, to bring down ore and to blockhole boul- miner retreats one set and repeats operation. When 12 ft, work starts in same way in sub No 2, and so on s occur, the upper one is mined first. Advantages ber and powder consumption, safety, high extraction, deposits over 15 to 20 ft thick.

layers or slices parallel to hanging wall, by methods e is taken first, followed by lower slices as ground



g 367. Mercur, Utah. Sub-level Caving

development consists of driving crosscuts *S* from s. Mining begins on the highest sub-level; ends of crosscuts next to hanging wall are widened out until two or more crosscuts are connected, the roof of excavation being temporarily supported by stulls. To start caving, a few holes are drilled in the back next to the hanging wall. Pillars between crosscuts are worked back toward footwall by driving successive slice-drifts, 6 or 7 ft high, across them; slices are timbered with 3-piece sets. Ore is handled to chutes in barrows. Shoveling along face of cave is stopped when much waste appears, and a new section of the back is allowed or forced to cave, a safe working place being kept

ub-level Caving

s all done by hand, no heavy blasts are fired, and light ot started in any area until the sub-level above has ning ore in blocks, caving may be carried on simulta-

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neously on several sub-levels at different points along the strike. Thickness of ore on each sub-level is 4 to 7 ft; or from 30 to 50% of the orebody is broken by caving; remainder is obtained in drifts and slices. Results of work indicated that a 12-ft between sub-levels might be better; a 4-ft back could be kept under better control and higher extraction than a 7-ft back. It is difficult to extract all the ore along the especially where the dip is flat. Where this method was used, ore was fairly firm, but ng wall was weak. Timber and powder consumption were low; labor cost, high. tract prices for mining, 60¢ to $1.40 per 1.25-ton main-level car; day's pay miners per 8 hr. Daily output, 700–800 tons; output per man-shift underground, about

Arizona Copper Co, Morenci, Ariz. Fig 368 (P. B. Scotland, 1915) shows a sometimes employed in the upper part of an orebody preparatory to top-slicing. evels are 20 ft apart vertically; intermediate hand-tramming levels, about 75 ft. Interval between raises, 20 ft; ground is mined in blocks 40 ft wide. No mat grade of ore recovered is seriously decreased by mixture with waste (178).

79. Summary, Sub-level Caving

General. Sub-level caving is adapted to large-scale work in large, weak bodies (see Suitable orebodies, Art 76). It is safe, and when properly cond will yield a high extraction and produce clean ore. Disadvantages are in g the same as for top-slicing (Art 75); see comparison below.

Requirements (*a*), (*b*), (*d*) and (*f*) for top-slicing (Art 75) apply direc sub-level caving; other essentials for the successful application of sub caving are: (*a*) Weight of caved ground is necessary on top of working orebody must be wide enough to permit free descent of gob. (*b*) Waste, large amounts, prohibits sub-level caving, as it can not be stowed undergr Waste horses must be broken unless large enough to divide the deposit sections which can be worked independently. Some grading of iron ore is po (Chapin mine, Art 77). (*c*) Boundaries of orebodies should be fairly regul

Sub-level caving *vs* top-slicing (Art 75). Relative ADVANTAGES of sub caving: (*a*) Cost of breaking ground is lower. (*b*) Less timber is required. Larger daily output is possible from a given area. (*d*) Natural ventilati generally better, and there is less timber to be hoisted to working places; conditions and plan of work determine whether sub-level caving has any iority in these respects. Relative DISADVANTAGES of sub-level caving: Percentage extraction is slightly less, and there is always a greater chan losing ore. (*b*) Mining is not so clean; more waste is mixed with ore. Less sorting is possible in slices; grading of ore is more difficult. (*d*) Cavi overlying ground is not under such close control.

Factors of relative cost, percentage extraction, clean mining and co predominate in making a choice. Top-slicing is better in very soft ground is used where its higher extraction produces ore rich enough to offset its hi cost, and where admixture of waste would lower the metal content of ore ciently to impair its market value. Under other conditions, sub-level cavi preferable. Block-caving is a possible alternative for sub-level caving (Art

80. Block-caving

General. The orebody is divided into large blocks, which are then under one at a time, and allowed to fall. They are caved in descending order, from top of the orebody. Under proper conditions, the ore crushes by its own we and that of the overburden into pieces of suitable size for handling. The sett and crushing require time, often several months, during which mining and velopment proceed in some other part of the mine. This method develo from sub-level caving (Art 76) by increasing the height of ore caved in one op tion. Some pillar-caving methods (Art 85–88) are varieties of block-cav Following paragraphs presuppose a knowledge of details obtainable from Art

ins, thick beds, or massive deposits of homo-
ich will cave readily. The ore must be such
locks are developed and undercut, and will
requirements and limitations, see Art 82.
in the Lake Superior iron districts and the

ere are four distinct forms:
e first, and is the type form. The blocks are
handled by shoveling in drifts driven through
al variations, for different conditions of ore

ites (see Tobin mine and Detroit Copper Co,
on a sub-level, and the caved ore is drawn
ramming level.
hutes. Blocks are undercut on a sub-level,
below by branched raises (see Inspiration,
nines, Art 81).
locked out by main and sub-levels. Each
s around its sides by irregular open stopes,
ore finally cave. The rest of the block is
l below (see Pioneer mine and Utah Copper
ack-caving into chutes") is sometimes classi-
53).
tively "block-caving into chutes." Other
ses (see Mowry and Commercial mines, Art
nt forms, see Art 82.

to the form of block-caving used; typical
en in Art 81. Remarks on modes of entry,
, for top-slicing and sub-level caving, in Art
aving.

) Large blocks are best, as they increase the
ng and by blasting. Blocks of large horiz

ble 54. Size of Blocks in Block-caving
(Details in Art 81)

Mine	Width, ft	Length, ft	Height, ft
e mine	200 (*a*)	200–250	100–125
inee Range (*b*)	250	250	100
nine (*c*)	100	200	125
pper Co	100	100	60
ion mine	(*e*)	(*e*)	60–70
Copper Co	25	(*d*)	20–35
mine	65	180	150
rcial mine	50	50	30–70

dth of blocks equals width of orebody. (*b*) 1 mine.
st block caved up to 1912. (*d*) This dimension
dth of orebody, which may reach 150 ft or more.
escription, Art 81.

of block which can be caved in any mine is
based on experience under similar conditions;
lock-caving used, and the strength and char-

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Timber mats for separating capping from caved ore are rarely feasible in They may be formed as described in Art 71, but can not be built up nor repa ing begins (Detroit Copper Co, Art 81).

Extraction varies from 70% or less, with careless work under conditions, to 90% with careful work under favorable conditions; data are available. Extraction is affected by: (*a*) size of block; (ore, walls, and capping; (*c*) mode of drawing caved ore; (*d*) spacin

Following experiments at Inspiration mine (Art 81) indicate the relatior these factors (see also Art 82). Tests were made in 1913–14 to determine t extraction to be expected in drawing ore through chutes. A 20 by 30-in bc had glass sides and a wooden bottom in which 0.5-in holes, representing chut on the corners of 1.25-in squares. On a scale of 1 in to 5 ft, this box repres of ore, 150 by 100 ft by 125 ft high; the bottom holes represented chutes 2 6.25 ft apart. By plugging some holes, the interval between chutes cou The box was filled to a given height with crushed ore, upon which was pla crushed capping. The ore was drawn through the chutes, and the percent and dilution with waste were determined by weighing and assaying the ma Dark red, barren rock represented capping, the color of the ore being ligh the definite line between ore and capping could be watched through the drawing proceeded. Experiments were made with the box filled to depths 35 and 70 ft of ore, overlain respectively by 35 and 55 ft of capping. Chut 6.25, 8.87 and 12.5 ft were used and the size of particles of ore and cappin In some tests, drawing was done uniformly and in small quantities at a ti chute; in others, chutes were drawn direct to capping, one at a time.

Conclusions from the above experiments: (*a*) Extraction is increased by i height of ore column and by decreasing distance between chutes. (*b*) Ex clean ore, before capping appears at the chutes, is improved by uniform dra chutes and by decreasing the amount drawn at one time from each chute; not give the highest total extraction for a final product containing 1% Cu. tion increases with coarseness of the particles of ore and capping; fines dec tion, the lowest result being obtained when both ore and capping were fi (*d*) When a uniform limit of grade is allowed at the end of chute drawing, t increases with the % Cu in the ore. (*e*) Assuming that when the ore cav the same volume as in place, the total extraction increases with the amount development and undercutting.

In deciding what extraction could be expected at Inspiration mine in cav average ore 35 to 70 ft high, the results of tests 1 and 2, Table 55, were co servative, provided the chutes were closely watched and no ore below the limit was drawn. Results of tests 3 and 4 were judged obtainable when the the caved ore is strong enough to allow closer spacing of chutes. These result that the capping will break up like ore and not into a fine sand. For details, s

Table 55. Ore-drawing Tests, Inspiration Copper Co, Miami, Ari

Test No	Distance between chute centers, ft	Depth of ore, ft	Aver grade of product, % Cu (*a*)	Per cent of original ore extracted	Apparent extraction (*b*)	Aver co of pr
						% ore
1	12.5	70 (*c*)	1.751	84.90	87.34	97.20
2	12.5	35 (*d*)	1.738	72.87	75.54	96.46
3	8.87	35 (*d*)	1.742	81.15	83.93	96.69
4	8.87	70 (*c*)	1.770	89.55	91.12	98.28

Note. — These tests were calculated for ore assaying 1.80% Cu; drawing when chute product (ore and capping) assayed 1% Cu. Particles of ore and in tests were screened to sizes representing lumps 4 to 12 in diam. (*a*) include ore won by development, which amounts to 25% of total tonnage in high and 15% in blocks 70 ft high. (*b*) Total tons of ore and capping per 100 ore. (*c*) Overlain by 55 ft of capping. (*d*) Overlain by 35 ft of capping.

Block-caving Practice

ch (999, 153). A lens of hard siliceous hematite, lipping 76° to 90°, is overlain by hard, horizon-

. Block-caving
t of the deposit;
h and full width

in haulage-drift
it 20 ft from the
wall at the ends
ediate and equi-
. The hanging-
intervals along
ft of the level
ir tops. From

Development for Block-caving, Pewabic Mine

ft alongside crosscuts *T*; elsewhere the block is
nall as is consistent with safety. None of the
ork requires timbering. Pillars are drilled with
us holes and blasted out in sections. The block
then free at top and bottom, and is practically
orted at its ends. A block takes several weeks
7 ft, after which the caved mass continues to
and crush itself; in 6 or 8 months, 80% of the
pass a 3-in ring.

the ore is sufficiently crushed, timbered cross-
(Fig 371) are driven by spiling to the hanging
m the stub ends of crosscuts *C*; drifts *D* are
off from *M*, every 25 ft. Short crosscuts may
driven at 25-ft intervals from nearest drift to
wall. Plats are laid at ends of drifts, ore is
to run in and is shoveled into cars. When
ppears, a set or two of timber is blasted down
wing resumed. Work retreats from the ends of

Fig 371. Extraction Drifts

open. Caved
ed by driving
y between *D*;
wn last.
locks is done
Lake naviga-
caved blocks
es cost of this
per ton.

mainly in de-
mode of isolat-
by practice in
e Menominee
(300).
n the footwall.
re by 100 ft high. The ore under a block is cut
scuts, 7 ft wide by 8 ft high. A narrow overhand

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stope is carried to the level above along both foot and hanging walls. Pillars are then removed in slices 8 ft wide, parallel to the strike. This work starts in the pillars along the hanging wall and retreats to footwall; the back of each slice is allowed to cave before starting another. Caved ore is extracted as at the Pewabic mine.

This variation would be used where ore does not separate readily from the walls, or where the walls are irregular; more isolating stopes might be necessary at ends of blocks. Both open and shrinkage stopes have been used for isolating blocks; the former are preferred in the Lake Superior districts (286). This mode of undercutting blocks allows the use of timber for temporary support of ore too soft to stand when undercut, as in Fig 370.

Mowry mine, Ariz. Data from R. B. Brinsmade in 1907 (153). Argentiferous lead carbonate ores occur in irregular steep-dipping pipes or shoots, on a granite-limestone contact. The ore is mostly soft and crumbly, with much clay, Fe_2O_3 and MnO_2. Fig 372 shows method of mining a shoot about 65 by 180 ft in horiz section. A vertical shaft is sunk at each end of the shoot; at vertical intervals of 150 ft, the shafts are connected by drifts *D*, from which crosscuts *C*, 25 ft apart, are driven to the walls. A square-set stope (Art 45), 2 sets high, is then opened over the entire area of the deposit. Tops of 1st-floor sets are lagged, numerous chute-gates are built in the sill-floor sets, and the sides of the 1st-floor sets over the chutes are lagged with 2-in plank. The lagging over the chute-sets is then removed and the overlying ore caves into them. By drawing from all chutes uniformly, t to settle vertically, and distortion of the sets is avoided; some sets always re

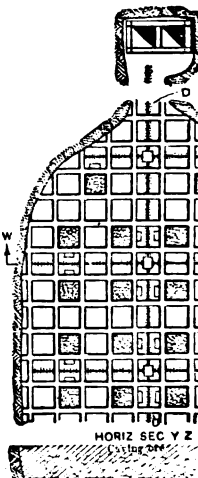

Fig 372. Block-caving
Mowry Mine (

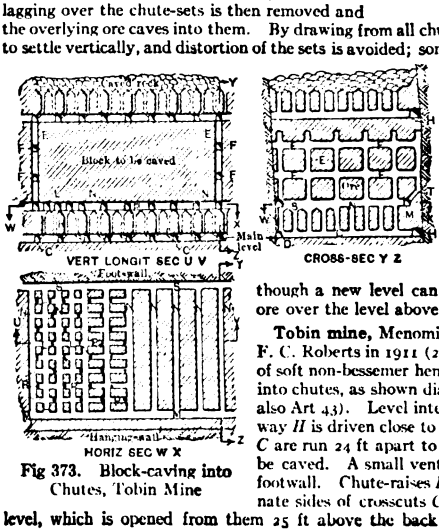

Fig 373. Block-caving into Chutes, Tobin Mine

ment (Art 5 which clog chut occasional area ore, too hard t moved by carry up to the soft or of the ore is cave simplest form o into chutes. It field of use, un tions of ore occu ing is confined though a new level can be prepared wh ore over the level above.

Tobin mine, Menominee Range, Mich F. C. Roberts in 1911 (208). A large, ve of soft non-bessemer hematite is mined b into chutes, as shown diagrammatically b also Art 43). Level interval, 125 ft. A way *H* is driven close to the hanging wall *C* are run 24 ft apart to the footwall und be caved. A small ventilating drift *D* is footwall. Chute-raises *R* (Art 68) are put nate sides of crosscuts *C* at intervals of 1 level, which is opened from them 25 ft above the back of the main lev

ut 15 ft from the hanging wall, the crosscuts *N*, *C*, and a drift *S*, along the footwall. Raises *T* wall, and opposite each crosscut; they cause the h them, thereby leaving an added thickness of *H* (cross-sec *YZ*). The tops of chute-raises are on the sub-level into pillars 10 by 16 ft; in strong ts *L* (horiz sec *WX*). The ground at the ends of rosscuts *N*. Crosscuts *F* are driven 25 and 50 ft umber of raises *E* varies according to the ground. numerous holes are then drilled in the pillars and ore above to drop. Through the chute-raise gates possible over whole area of the block. Caved ore st be blasted in chutes. Pillars between sub- and w. Up to 1912, the largest block caved was 100 ber is required. Main haulage-drifts are timbered to 15 in diam; caps, crosscuts, 8-ft posts r are used (301).

Fig 374. Modified Drift Timbering, Block-caving

ata from F. W. Sperr his general method, d haulageways were asting in chutes may s on main levels and topes at Center Star drive sub-drifts be- the ore is undercut, e-raises and allowing of blasting. (Com- i, Art 87.) Fig 374 its, recently designed e maintenance costs. considerable settlement without destroying the chute-gates and the posts of the upper sets, the latter may settle without affecting the clearance between the chute lip and the top of the car. The very low maintenance cost of this timbering is said to effect a substantial saving, although its first cost is higher than that of ordinary drift-sets.

Weaker ore may be mined by block-caving with this form of timbering than with the Tobin method, but its application is limited to ore which will stand unsupported over areas sufficient to allow erection of the sets. Chutes may be close together over the whole area of the block, to permit uniform drawing of caved ore. Undercutting in soft ore is completed a few feet above the timber by barring through holes made for that purpose. After caving begins, ore must be drawn fast enough to keep excessive weight off the timbers.

Detroit Copper Co, Morenci, Ariz. Data from W. L. Tovote in 1910 (280). For ore occurrence, see Art 74. DEVELOPMENT. Main haulageways and intermediate hand-tramming levels are opened as for top-slicing (Art 74). Two-compartment raises, put up to the capping from the highest intermediate level, are 4 by 6 or 5 by 7 ft in section, and are at the corners of 30-ft squares or 25 by 30-ft rectangles. Sub-levels *S*, Fig 375, 0 and crosscuts *C*. The vertical interval between arity with which the ore breaks; it varies from 20 the orebody on the highest sub-level. Fig 376

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shows the successive operations in any area. Drifts and crosscuts, d
pillars formed by sub-level development, subdivide each pillar into 4
sides of which are cut away as much as is safe. Ore in the sub-level f
around the raises to a funnel shape as at *F*, Fig 375. Numerous holes a

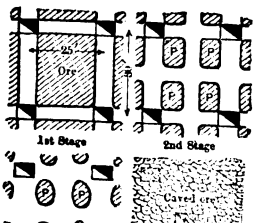


Fig 376. Stages of Work in Undercutting Sub-level Pillars, Morenci

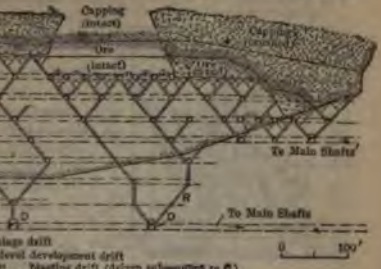
P, and 8 to 10-ft holes, 2 to 5 ft ap
all drifts. Timbers are removed fr
lying a section of a sub-level thus
holes fired simultaneously. CAVE
through the original raises, and thro
R, Fig 375, called "jigger-chutes";
untimbered, and the round that bre
through to caved ore is arranged to
Some ore is shoveled on the sub-level
edge of the caved area. Drawing
chute runs mostly waste. Occasiona
the chutes and must be blasted. G
shows block-caving on a sub-level 25
capping. Raises *T* would be unneces
between ore and capping could be d
wise, but they are useful in weakeni
caving. At times (279) the top of a
sliced (Art 74), until a heavy timber m
lished. To secure the advantage of a
ore from waste, a large area must be
level before any ore is drawn off, and drawing is done uniformly from a
then the mat sooner or later breaks up and timbers from it often clog th

Commercial mine, Bingham, Utah. Data from R. B. Brins-
made in 1907 (153). Orebodies are large lenses of low-grade
ore, formed by replacement of beds of limestone; they are 200
to 500 ft long and 30 to 70 ft thick. Practice here indicates the
possibilities of block-caving in thick, flat-dipping beds. An
inclined shaft follows the dip of the limestone; level interval,
100 ft. Ore is caved in 50-ft blocks (Fig 377). Inclined raises *R*
driven on the footwall are connected by the sub-level drifts *A*
and *B*; an intermediate sub-drift *D* and raise *S* are then driven.
Three raises *T*, from drift *D* to the hanging wall, are connected
by drift *E*; inclined raises *F* and drift *G* are next driven under
the hanging (see dotted lines in sec *UV*). All ore thus far
broken is handled through chute-gates in sub-level *A*, at the
foot of raises *R* and *S*. The pillars under the hanging wall,
formed by drifts *E* and *G* and raises *F*, are cut away (see sec
UV); footwall pillars between drifts (*B*, *D*) and raises (*R*, *S*)
are reduced to a similar shape. Holes are drilled in all pillars,
also in the backs of *B* and *D*, and in the raises *R* and *T*. The
holes in the hanging-wall pillars are fired first, then all others,
the block of ore breaking off along a vertical plane through raises
T. Caved blocks are allowed to stand and crush, and are then
drawn through gates in sub-level *A*. Successive 50-ft blocks
are similarly caved; work on each sub-level retreats from the
ends to the middle of the orebody. Caving is not begun on
any sub-level until the one above it is worked out.

Fig 377. Comm
Bingham

Inspiration mine, Miami, Ariz. Data from G. R. Lehman
in 1916 (262). Orebody is a massive disseminated chalcocite
deposit in schist and porphyry, overlain by 50 to 350 ft of leached
capping and containing about 100 000 000 tons of low-grade ore.
Fig 378 shows a cross-sec of the deposit, and the general plan of work; th
developed from that of the Ohio Copper Co (see below). DEVELOPMENT.
levels (4th and 6th levels, Fig 378) have been opened from 2 vertical shaf
100 ft apart in country rock. Fig 379, 380 show stages of lateral developm
levels consist of drifts *D*, 100 or 200 ft apart, 7.5 ft high, 9 ft wide at rail and
cap, inside of timbers. Every 25 ft along drifts *D* the back is broken out

top of the main sets. Chute-gates are built in raises *R*, which are in planes at right angles to



Branched Chutes, Inspiration Mine

on whether sub-levels are to be 30 or 35 ft apart opened from raises as shown. Raises are about

st sub-level (sub ping, consists of *E*, 150 ft apart. apart, those on blasting drifts *F*, ing the ore into us short branch- e main raises to raises are spaced f 12.5-ft squares ole area of the

UNDERCUTTING g are done by the pillars be- vel drifts; work n a cross-drift *E*, dary of the sec- nined. Seven 8 es, drilled as in stitute a round; one with 1-man , Art 21; rounds blasted singly. roken is drawn e finger-raises. round, the ore to cave as soon ercut; in hard ing does not start until undercutting has pro- siderably. DRAWING ORE (see also Table 55, imple gates (Art 91) are built in each finger- and 4 or 5 ft below the sub-level, so they will not when blasting undercutting holes. Grizzlies enings are placed at the foot of the raises on the ct below the caved ore. Men ("chute tappers") rs in drawing ore; 1 operates the gates, the large lumps on the grizzly. Boulders too large gates are blasted. GENERAL. By repeating hest sub-level may be caved and drawn off over ent and undercutting may then begin on any

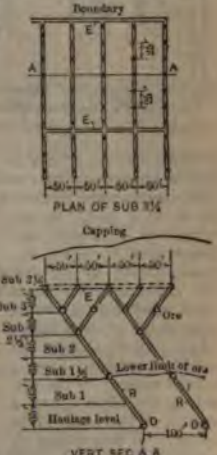


Fig 379. Development, Inspiration Mine

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lower sub-level. Obviously the system is flexible and a block of any he
the conditions of ore occurrence may be caved.

The Inspiration Co began large-scale development late in 1913, regular
tions in Aug. 1915, and the daily output
was 16 700 tons. Cost of mining at t
60¢ per ton, including 20¢ for develo
fixed charges. The percentage of ore
extraction can not be determined a
some portion of the orebody is lotto
tions, between the 1st undercut and
estimated to contain 1 886 450 tons,
extraction of 102.4% and a copp
86.52%; a higher copper recovery is
the second undercut.

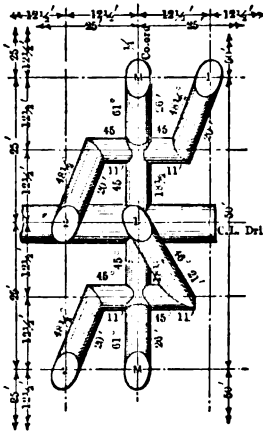

Fig 381. Diagram of Finger-raises, Showing Connections with Main Raises *M* and Sub-level Drift

Ohio Copper Co, Bingham, Utal
400 ft wide, 450 ft long, 1 300 ft deep
50°. It is a low-grade disseminated
tered quartzite; the ore stands well
ings, but caves readily. For data
ground, see Ex 3, Table 29, Art 28.
of mining is like that at Inspiration m
Fig 382, from C. G. Bamberger in 19
the original development. The shaft
meh, tools, etc. Ore is handled by g
3 main chutes in the footwall, delive
the tunnel, whence ore is hauled to
electric locomotives. ESSENTIAL FE
mining system (Fig 383), given by F. S
in 1915: (*a*) Interval between sub-le
earlier work it was 30 ft. (*b*) All raise
at angles between 42° and 50°, on which the flow of ore is easily controlled.
level is blocked out into rectangular pillars by 6 by 7-ft drifts and cross
50-ft centers respectively. (*d*) Where possible, blocks 100 ft square and
caved as a unit. (*e*) Arrangement of branch and finger-raises is shown diagrammatically by Fig 383. All raises under a 100-ft block start from a central point on the sub-level below, where a grizzly *G*, with 1-ft openings, is built. From 4 branch raises *B*, to the corners of the block above, a system of finger-raises is so opened that their collars are approximately on the corners of 10-ft squares over the entire area of the sub-level at the base of the block to be caved. Raises are untimbered and 4 ft square, with funneled collars; 6 or 7 raises are driven at a time, and it takes about 2 months to complete the 150 customary for a 100-ft block. (*f*)

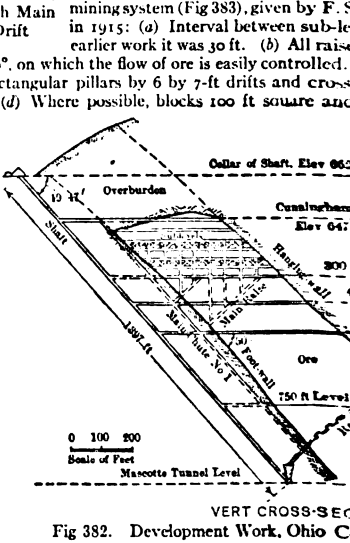

Fig 382. Development Work, Ohio C

Holes for blasting sub-level pillars (indicated in Fig 383) are drilled with
on columns; 3 10-ft back-holes and 4 8-ft side holes are drilled per set-up;
set-ups, 9 ft; 8 sticks of 1.25-in, 40% dynamite are charged per hole, 36
being used per 100-ft block; all holes in a block are fired together. (*g*) 5
in a block is removed from drifts and crosscuts, and about 15% from raise

for the block to settle and crush itself. (*h*) Caved ates, placed as close as possible to the sub-level on keep the bsolutely ay 6 ft in f ore over ng a 10% loss from the sub- sub-level will not round of es ending n; where start the ccasional of about he other ntrol the ad trains , a gate over 50 aided by amount ender re- drawing. lders are) A ton- results of 00 tons; criterion

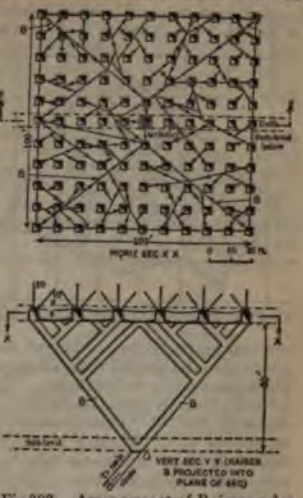

Fig 383. Arrangement of Raises under a Block 100 ft square, Ohio Copper Co

of early work here were unfavorable for high extrac- a block 100 ft square by 60 ft high (50 000 tons) is:

	Total cost	Tons ore	Cost per ton
3.25 per ft	$2 275	2 450	$0.93
1.00 per ft	6 000	8 000	0.75
.....	.....	39 550	.....
.....	$8 275	50 000	$0.22

er ton: Superintendence, 8.2; office, 1.1; engineer- , 2.5; carpenter shop, 1.8; miners and shovelers, powder men and roustabouts, 1.1; loading trains,

Nev. Data from P. B. McDonald in 1916 (306). UTH MINE. Orebody is a slightly dipping lens of soft porphyry, several hundred ft long and wide. reatest depth, 500 ft. Above the 300-ft level a) was used, but the ground is so soft that it was age-drifts. Below 300 ft, there are inclined raises . Beneath a block to be caved, branch and finger- rners of 12.5-ft squares. Main raises slope 50° to timbered with heavy square timber. Care is taken apping along a horiz plane or one with a regular ive loss of ore by mixture with overlying waste. nor on practice in undercutting pillars.

286, 303, 304). The OREBODY, an extension of the -shaped mass of hematite, occupying the sides and ng, and overlain by a jasper capping. It contains ole, cave readily and break into fine particles. It DEVELOPMENT. There are 2 shafts in wall rock on erval is 100 ft; sub-levels 33.3 ft apart are opened

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between main levels. Main-level development (Fig 384) is as systematic permit; pillars vary in length and breadth from 50 to 75 ft. Levels are t

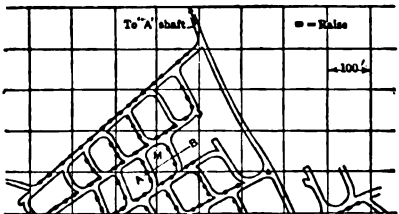


Fig 384. Plan of 12th Level, Pioneer Mine, Ely, Minn

piece sets, of 12 to ber with 8-ft posts caps. Vertical raises, 25-ft apart level openings, a with 6 to 10-in rou portant raises are 4 those for small qua ft sq. Inclined ra where necessary (se levels (Fig 385), op by drifts and cross those on main level bered with 3-piece posts and 8-ft cap gins at the ends of the main or sub-l the caved ground.

crosscuts are first driven on the sub under attack (Fig 385), forming pill square, these being small enough to be partially crushed by weight of the o

The mode of caving varies; in places it is quite irregular; usual routine is shown by Fig 386. From a raise nearest the caved ground, a short drift *D* is driven and heavily timbered. A pear-shaped stope *C* (a "chance") is then opened with short holes and light shots; it is enlarged in steps (see numbers 1, 2, and 3), until it becomes unsafe to enter and the adjacent ore caves into it. At first, ore is shoveled to raise *R*; later, caved ore runs to it with little shoveling. When mat appears at its entrance, the "chance" is boarded up or allowed to cave. Usually 3 "chances" are opened around a raise; 2 of them are shown at *C* and *E*. Fig 386; the third is started from angles to *C* and *E*, and, depending on the location, extends to the wall or

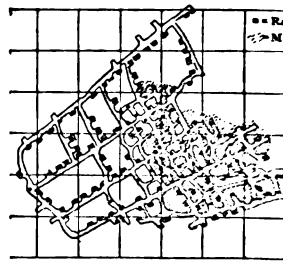


Fig 385. 1st Sub-level above 12th Pioneer Mine

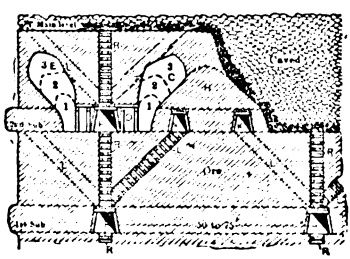
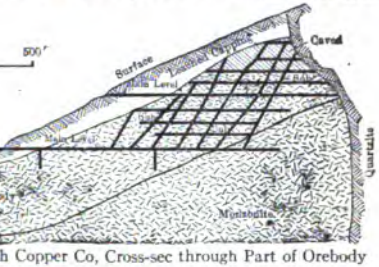


Fig 386. Diagrammatic Vert Sec on *AB*, Fig 384, Pioneer Mine

from previous work aroun from *F*. When all the "c a raise have run out, the along drift *F*, and start ca raise. After completing ca around the periphery of a 384), a "hog-back" of ore mains. *H* is caved into ra sub-level next below), whic and crosscuts *K* on sub-le ground and aid in caving ceeds simultaneously on se the faces being at least 50 sequence of operations as (35): While the 11th-level drawn, the 12th level and developed, and drawing ha subs. The 13th level is being developed and its 1st sub is started. The 14t by a drift for drainage, and the shafts are being sunk to 15th level. The no put is 720 000 tons; the block of ore between 11th and 12th levels contained tons. During 1 month (26 working days), roughly 55 000 tons were pro

hift. Output per shift per man underground, 8 to 6 5.5 tons. HANDLING ORE. On sub-levels, ore which ises is handled in barrows. Main levels are equipped is method is adapted to large deposits of moderately r and well-defined boundaries; it is unsuited to very ore is unavoidably lost by mixing with the capping. tilation; (c) cheap mining, since timber and powder openings are not proportionately numerous.

A large, low-grade deposit of disseminated chal- also Art 97). Caving was employed in the earlier ak up the capping for the steam shovels without he pillars were weakened until they caved, and the he chutes. This plan was modified later by putting f ore. Finally a branched-raise caving method was er Co; it also resembled the Pioneer method. Fig ain levels were opened from tunnels 200 ft apart



h Copper Co, Cross-sec through Part of Orebody

ntrolled. Numerous branch and sub-branch raises m main raises between subs. The collars of branch f 15-ft squares. A block of ore was caved by enlarg- finally mining the remaining pillars; for further de- e (114) estimates that about 7% of the ore would be 9 sub-levels, or a total thickness of 310 ft; in caving r 20% (these losses appear to be underestimated). l 1 500 to 1 600 tons per day, with 225 men under- development, showing a production of 7 tons per

mary. Block-caving

olicable, gives a lower mining cost than any other y for large-scale work. In its different forms (Art of various shape and to ores of various strength, but s. In unsuitable deposits, or where carelessly con- at of any other method; systematic work, careful sential to success.

common to all caving methods (Art 83), are: . Vein-like deposits should dip over 65°, and hich, for ideal conditions, the ore should part walls by opening isolating stopes, the cost of e deposit is very wide. Block-caving may be sufficiently thick, preferably at least 70 or 80 ft. ine (Art 81) is applicable to inclined beds 30 ep enough for ore to run on it. The physical break up under block-crushing can not yet be adapted to moderately soft or moderately

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hard, but not to very soft or tough ores. Homogeneous ore of grade is necessary, as no underground sorting can be done. The cave when the underlying ore is dropped; weight of overburden aid in crushing the ore. The most favorable capping breaks int soft, brittle capping decreases extraction. A capping differing from ore is desirable, for ready detection at the chutes at the operations. (See below, Pewabic method *vs* block-caving into ch

Advantages of block-caving: (*a*) Safety. (*b*) Cheap min little drilling, blasting and timbering are done per ton of ore; amo ment is relatively small. (*c*) Production is centralized, allowing vision. After caving begins, a large and easily varied output is one level. (*d*) Natural ventilation is good, compared to that i methods. DISADVANTAGES: (*a*) Preparing the blocks for caving and considerable outlay. (*b*) Extraction is often low, and the danger of losing large amounts of ore. See also below.

Pewabic method *vs* block-caving into chutes. ADVANTAG method (Art 81): simplicity and relatively small amount of de ton of ore caved. DISADVANTAGES: cost and trouble of driving a timbered drifts through caved ground, and necessity for shoveli Block-caving into chutes obviates shoveling, but the developmen plicated, chutes often clog, and maintenance cost of drifts on tra frequently high. For modes of minimizing these difficulties, see of the Tobin method (Art 81). In making a choice between block-caving, the strength and character of ore and capping are fac important than the above. The Pewabic method is limited to fa because, under each block, there must be an open stope, large eno to settle and crush. Block-caving into chutes is adaptable to m Thus, at Inspiration mine, drifts are the largest excavations whic open while a block is being undercut. Broken ore from pillar through the chutes, to provide space for settlement and crushin fications of Pewabic method (Art 81).

Some ore is lost in all block-caving, by mixing with capping when the ca This loss is greatest under a non-coherent or brittle capping, which wh fine and sifts into the ore; it can be minimized by uniform drawing over t block. Use of chutes allows fairly uniform drawing, impossible in the which is therefore applicable only to deposits overlain by strong rock, bre caved. This was the condition at the Pewabic mine, where the cappin zontally-bedded sandstone (208). In general, the Pewabic method ha limited field of use than block-caving into chutes.

Vertical *vs* inclined or branched chute-raises. Relative inclined chute-raises: (*a*) They are easier to construct, and if pro are less apt to clog. (*b*) For a given spacing they require less drif cutting on tramming-levels and subs where ore is undercut (con 375 with Fig 380, 383, Art 81). Conversely, for a given first cos velopment, they may be spaced closer than vertical chutes, wh for uniform drawing over the whole area of a block (see Table are applicable in softer ground, since with vertical chutes the practically unsupported, while the tops of raises are funneled ou sub-level pillars are cut away. (*d*) They collect the ore as draw tively few points on the sub-level below the caved block. Grizz stalled at such points and large lumps spalled before entering the haulageways (Fig 383). With vertical raises, the ore must be trammed from many different points. (*e*) Gates can be operated i

ab where the ore is undercut; hence, when draw-
wn down only a short distance into the chutes.

ates, but the author does not know whether they have
cult, and they would always have to be placed below
tom of chute-raises (Fig 373, 375, Art 81), a consid-
ed with capping at the end of drawing operations.
is limited to orebodies of large horiz area; one dis-
381) are complicated and must be set by survey to
undercutting floor.

rt 79). Relative advantages of block-caving:
utput from a given area, requires less develop-
tilation is better, and, by caving into chutes,
l caving is possible in softer ore and smaller ore-
l, and yields cleaner ore, with a higher percent-
ften outweigh the advantages of block-caving.

ing Methods (See also Art 75, 79, 82)

o use of caving methods: (*a*) large-scale work;
as wide veins, thick beds, or masses, usually
om glacial drift to firm rock; (*c*) ore which is
ghly fractured; (*d*) deposits of cheap minerals
ore or contamination with waste is less serious

, sub-level caving and block-caving are stated
requirements common to all caving methods:
d "follow down," as the ore beneath is removed.
p of the workings is useless unless that weight
n. The capping need not always cave to the
iently to protect the workings and furnish the
ing which hangs up and then drops suddenly
angerous; both the workings and the men may
idence must not injure valuable property nor
ous. This precludes caving under structures

Table 56. Comparison of Caving Methods

From the standpoint of:	Usual order of merit		
	1	2	3
Cheap mining costs....	BC	SC	TS
Clean mining.....	TS	SC	BC
Percentage extraction..	TS	SC	BC
Close grading of ore...	TS	SC	BC
Flexibility.....	TS	SC	BC
Chance of losing ore...	TS	SC	BC
% of ore won by caving	BC	SC	TS
Timber consumption..	BC	SC	TS
Natural ventilation...	BC	SC	TS
Control of caving	TS	SC	BC
Large output from given area.....	BC	SC	TS

BC = Block-caving. SC = Sub-level caving.
TS = Top-slicing.

, in addition to the summary in Table 56, that
y soft, heavy ores; sub-level and block-cavir

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to stronger ores. A weak, friable capping favors top-slicing; str which breaks large favors block-caving. Sub-level caving alway position intermediate between top-slicing and block-caving. Wh exists, it usually lies either between top-slicing and sub-level caving sub-level and some form of block-caving; for detailed comparisons, s

COMBINED METHODS

84. General

Term "Combined method" designates here a group of mining volving a systematic use or combination of two or more of the meth in Art 29–83. Such methods vary widely and are difficult to classify They are employed chiefly in mining large bodies of soft or hard or they are for large-scale work and aim at a high extraction.

Combinations used generally involve subdivision of the orebody of alternating pillars and stopes; the stopes are usually shrinkage various methods for mining pillars (see Table 57). Combinations

Table 57. Combined Methods

Mine	Method of mining	
	In stopes	In pillars
Ray Consol Copper Co..	Shrinkage	Block-caving
Miami Copper Co.....	Shrinkage	Shrinkage stoping and sub-level caving
Braden Copper Co.....	Shrinkage	Block-caving
Duluth, Cananea.....	Shrinkage	Block-caving & top-slicin
N J Zinc Co.....	Shrinkage (*a*)	Top-slicing
Homestake, S D.....	Shrinkage (*a*)	Filled square-sets

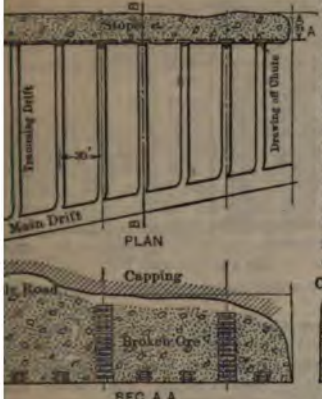
(*a*) With delayed filling, Art 68.

stopes and block-caving are also known as "Shrinkage Stope and I or more simply as "Pillar-caving" systems. The South African d use combined shrinkage stoping and sub-level caving (Art 89), dif in principle from pillar-caving.

85. Practice at Boston Mine, Utah Copper Co

General. SHRINKAGE STOPES AND PILLAR-CAVING were used for by the Boston Consol Co, but were abandoned in 1914 after the been acquired by the Utah Copper Co, which it adjoins. This tended its opencut (Art 97) into the Boston ground and ceased mining there. The present Ray mine method (Art 86) was deve from the Boston method; its evolution illustrates many essential mining of large, low-grade orebodies by shrinkage stopes and pillar-c are two variations, based wholly on differences in mode of handling TRAMMING SYSTEM, in which stopes are opened directly over the d levels and the ore is handled there in small cars to chutes conn haulage level below; (*b*) SUB-LEVEL OR MOTOR-HAULAGE SYSTEM, in begins from a sub-level above a motor-haulage level and the ore i to the latter through chute-raises; the sub-level is a stoping bas BODY is a large, low-grade, disseminated copper deposit, in monzo overlain by a leached capping. It was opened by two adit level vertically, the upper reaching a max depth of 200 ft below the Following data were published by L. A. Bl

ramming system. In the early stages, the plan shown in Fig 388 was tried. the center line of the stope nearest the ore boundary was widened to 20 ft. as carried up a shrinkage stope, access to which was through "pole roads" (Art e ends of alternate tramming drifts. At first, excess ore was shoveled at the ends of the drifts; later, chute-gates were erected in each drift to eliminate shoveling. DISADVANTAGES: (*a*) max output was limited to that which could be produced from one stope; (*b*) the broken ore in the stope drew down on one side, making it difficult for machinemen to work; (*c*) as cars had to be loaded at the end instead of the side, they slopped over, and much shoveling was required to clean the tracks; (*d*) the widening of the stope floor on the level wasted time and money, as the space so formed was filled with unrecoverable ore; (*e*) final extraction was low, as ore in rear of stope could not be recovered.



8. Boston Mine, Hand-tramming System; Chutes at One Side of Stope

Fig 389 shows a later plan. Chute-gates, 30 ft apart on each side of the stope, made it possible to draw the swell of the broken ore more

nd to secure higher final extraction; also several stopes could be worked simul-. But, with the chute-gates so arranged, the miners had to work at times in e of the stope, where the back was continually sloughing off, and in starting a e ore had to be shoveled from the middle to the sides. Ventilation was poor, the gave trouble, and the other disadvan-he plan in Fig 388 persisted. More-large openings caused by placing the from the tramming drifts opposite each kened the bases of the pillars, making to maintain the tramming drifts.) shows the final stage of the hand- system, before underground mining The development work was halved by amming drifts in alternate pillars only; were staggered, reducing the size of their entrance to a minimum. Two were erected in cut at the stope provide gates stope instead of des. The stope e above the level stead of a series apexing at the This important ent did away ensive drifting ing on the stope

Fig 389. Boston Mine, Hand-tramming System; Chutes on Both Sides of Stope

made it possible to recover all the broken ore. *Manway* raises were put up in pillars at 60-ft intervals along *tramming drifts, small crosscuts* from them fur-ccess to stopes. *This mode of entry was later replaced by* manway pillar drifts *because the pillars cracked and the raises became unsafe.* Manway drifts were *d ground, but in presence of seams and faults they caved;* also the drifts

crosscuts on the level require much retimbering. The arrangement of chutes in t required a tram uneconomically long for hand work. The detail of caving pilla drawing ore was not worked out; added drifts have been needed on the levels to extract the pil

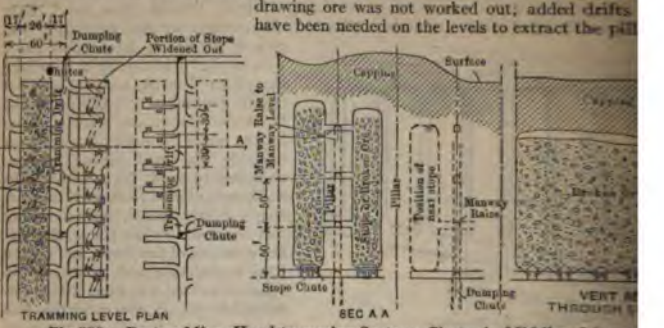
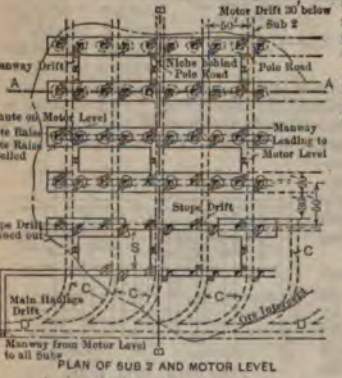
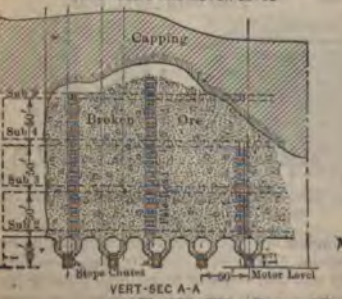


Fig 390. Boston Mine, Hand-tramming System; Chutes in Middle of Stope

Sub-level or motor-haulage system (Fig 391, 392). Stopes are at right angles drifts on the motor level and are opened 30 ft above it by widening the stope d which connect the tops of the chute-raises; the latter are spaced 25 ft apart along lines of stopes. The manway entra stopes gave trouble in this as in the tramming system. At first, raises w up from the motor drifts 100 ft apart center lines of alternate pillars, with throughs at 50-ft vertical intervals pillars usually faulted or sloughed bel



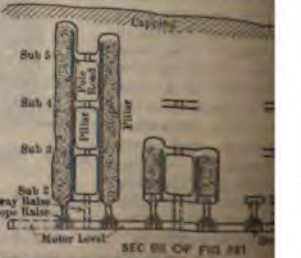
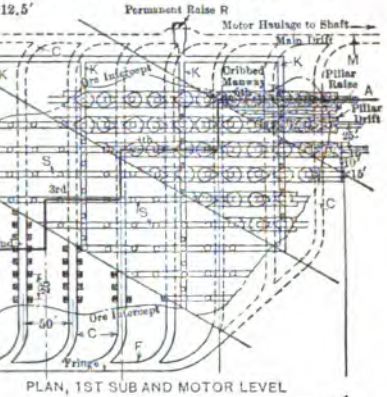


Fig 391, 392. Boston Mine, Motor-haulage System

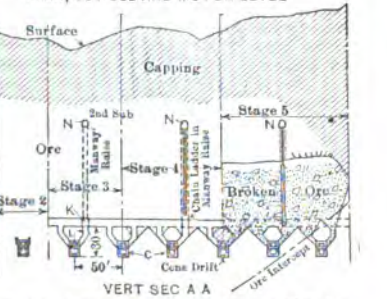
off these entries. Fig 391, 392 show the next plan pened by driving manway drifts through alternate 100-ft intervals along drifts, and a pole road up the ay raises, from motor level to 1st sub were located as were connected to a raise outside of orebody. This tries to stopes, but were costly; where pillars cracked, nd were expensive to maintain. The pole roads also ing them could not be easily handled in Sub 2, and at xpensive blocking and timbering to keep them open.

ce at Ray Mine, Ariz

deposit in schist and porphyry, covering 205 acres. and 2 500 ft at the ends, narrowing to a few ft in the 70 ft, averaging 121 ft over the whole area. Gross ging 2.19% Cu. A capping of leached schist aver- often fades into PROTORE, material which by further nmercial ore (Fig 397-8). Orebody is rolling, but re many small fractures, the ore stands fairly well e. See Art 11, 12 for methods of exploration.



PLAN, 1ST SUB AND MOTOR LEVEL



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ig 393. Ray Motor-haulage System (showing 5 stages of work)

-ft centers, leaving 10-ft pillars. Only 2 sub-levels otor level, and one usually 100 ft higher, near top

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of orebody, for a manway and ventilation. The sub-level manway drifts angles to the stope center lines, instead of along the pillars. Manways in the pillars by raises from the lower to the upper sub-level at 100-ft i the stope. Operations comprise 8 stages: STAGE 1: Drifting on motor l drift (*M*, F ft in the c along one c body. Fr motor drift are turned radius curv a "fringe the opposi deposit. A outside the compart raise *R*, th raise," is p it sub-leve STAGE 2: on motor the ore lim timbered

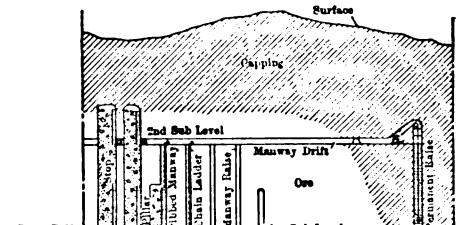

Fig 394. Ray Motor-haulage System

surmounted by pony-sets (Fig 395); angle braces and filler blocks are ground. Chute-gates (Art 91) are built in the pony sets every 12.5 ft a 2 gates being placed opposite each other in each set. Alternate pairs o chutes"). 25 ft apart, serve for drawing ore from stopes; the others ("pillar way between the stope chutes, are for drawing pillar ore. Outside of the orebody, there are no pony sets in drifts *C*; the drift sets have 12 by 14 caps and 10 by 12 collar braces. STAGE 3: Manway drifts, stope drifts and chute-raises. While driving drifts *C*, the 1st sub-level, 30 ft above, is opened. Manway drifts *K*, Fig 393, are 5 by 7 ft and 100 ft apart; they are 12.5 ft to one side of corresponding drifts *C* below, for connecting with raises *L* (Sec *AA*, Fig 393). Stope drifts *S* are driven 25 ft apart over the entire orebody, and are directly over the stope chutes on the motor level. Chute-raises about 6 ft diam are then put up from the stope chutes, on an

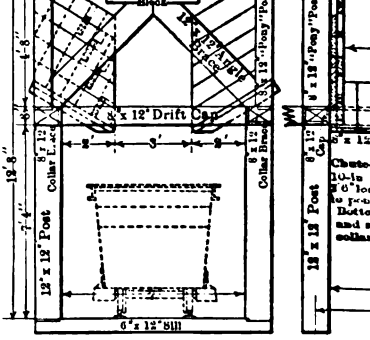

Fig 395. Ray Motor-haulage System, Drift S and Chute-gates

incline for 10 ft and then vertically to the stope drifts. On the 2nd sub drifts *N* are driven directly over drifts *K*. STAGE 4: Manway raises, bel raises, widening stope drifts, building manway sets. Manway raises are *N* at 25-ft intervals along drifts *K*; as their centers are 7.5 ft from centers they are bisected by the stope wall; chain ladders are hung in these raises. chute-raises are funneled while driving manway raises; this is done with sto in uppers. Stopes are started by drilling a line of holes slanting into the s drift; these holes and those drilled in belling the chute-raises widen the

e drifts *K*, directly under the manway raises, as soon y consist of drift-sets, on 8 by 8-in stringers projecting manways are built up from these sets through the STAGE 5: Mining stopes to the capping. For mode lso EX 5, Table 31, Art 28. The broken ore swells s are abandoned when the stope is halfway to the 2nd rough the manway raises. In hard ground, stopes are ore, 10 to 15 ft wide. STAGE 6: Before drawing the y one of the following methods. Method 1 (Fig 396), re 15 to 20 ft wide, leaves narrow pillars easily under- pillar chutes (2nd stage), start in the pillar nearest nnecting the raises along a pillar, they are widened, l. This is repeated in successive pillars, retreating (Fig 396) is used in soft ground, where pillars are wide en along center lines of pillars, 22 ft above the motor

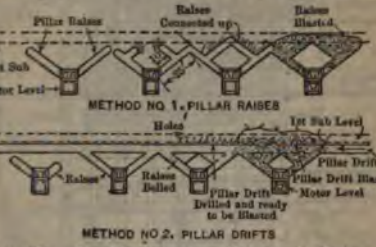

Fig 396. Ray Motor-haulage (6th stage), Undermining Pillars

ing breaks to the surface along a nearly vertical plane where haulage drifts are to be abandoned, the chutes er than the others, so that the capping assumes a slight ne the drifts take weight, all the ore beyond is drawn, ng. Only a few cars of ore are drawn at a time, to ing, with minimum loss and dilution of ore. When empty cars through the fringe drift into the back ends re trammed to the chute. A chute blaster or loader ushes it toward the main drift while another spots an rade in favor of the load. When an 8 or 12-car train the main drift, returning the empties to the back end ficiency results when only 2 trammers and 1 loader are mmers, 3 loaders, 2 machinemen (for blasting boulders nan (for repairing damage to chute-gates by blasting), of 150 tons per 8-hr man-shift. A boss, a timberman hecker are employed in each reserve-drawing section. ther detail.) STAGE 8: Cone-drifting. After all chutes , the ore below the sub-level in the pillars between the mbered "cone-drifts," parallel to drifts *C*, are driven on pillars (Sec *AA*, Fig 393). Chutes are built opposite ifts. A small shrinkage stope directly over the drift is r of the 1st sub-level. Ore is trammed to a winze at otor level. When all the cone-drift chutes are drawn s are mined by slicing. Thus, all ore above the motor- d.

97, 398). DEVELOPMENT on tramming levels comprises connected with fringe drifts *F*, near edge of the ore. using angle braces only in heavy ground. Caps are with 2 by 8-in, placed as indicated. Stope chutes are A permanent raise *R* is put up as in the motor-haulage

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system, and a sub-level opened 75 to 100 ft above the tramming level. The s
sists of manway or ventilation drifts, 75 ft apart and directly over every
drift *D*; manway drifts are turned off from a drift overlying one of the drifts
raises, with chain ladders, are put up from *D* to the manway drifts, as in the
and in Fig 398. STOPES, 15 ft wide and 25 ft apart, run at right angles
started from the stope chutes by driving inclined chute-raises intersecting
center line (Fig 397). Tops of these raises are widened and the stope carrie
motor-haulage system; 3 by 3-ft cribbed manways through broken ore direc

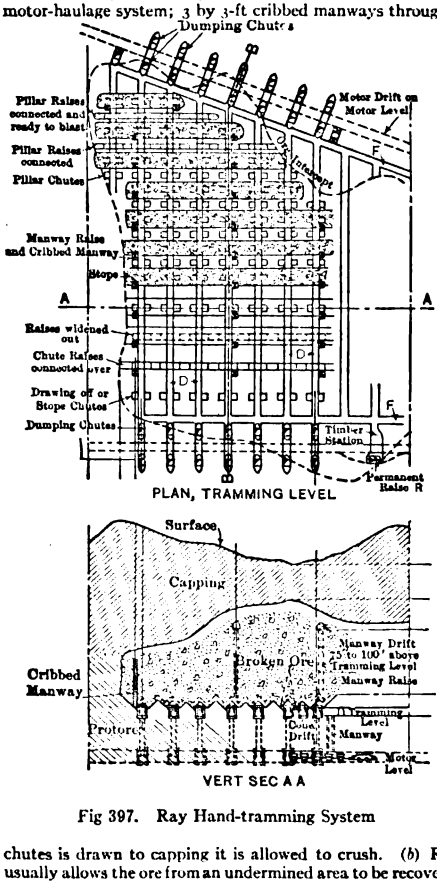

Fig 397. Ray Hand-tramming System

manway raises g
stopes. UNDERM
LARS (Fig 400) b
pleting all the sto
Pillar chutes are
D, Fig 397, betw
chutes. The rai
are driven fro
chutes, widened
the remaining or
(about 80% of th
ready to be draw
packs in cones
backs between tra
it is recovered by
as in the motor
tem, after the ch
drawn to cappin
CHUTES, connecti
tor-haulage level
ends of each dri
DRAWING ORE
drawing in the
system, because
drifts *D* is too sm
weight off them
culty is met as f
large blocks are
over their entir
pressure is brou
tramming drifts
pairs are excessi
"receding metho
is adopted. The
thest from the pe
are caved first, a
farthest from t
raise are drawn
soon as the first
run capping, the
undermined and
no area greater t
wide is crushing a
as fast as the f

chutes is drawn to capping it is allowed to crush. (*b*) Rapid drawing is a
usually allows the ore from an undermined area to be recovered without timbe
secure fast work and low costs, the number of trammers and muckers mus
proportioned to the non-producers, as timbermen and chute-blasters. Ther
many trammers as can work without interference. Chutes at each end of
drifts facilitate rapid handling; other chutes in the middle of each drift wea
and make them crush. In one section, with chutes as in Fig 398, 8 tramme
tons aver per 8-hr shift. (*c*) Drawing operations are systematized. Owin
number of chutes being drawn at one time, a continuous record must be k
know constantly the tons remaining and the assay. The car checker no

e, so that he can place his men advantageously and ar. The boss is responsible for a steady output, the e ore drawn. (*d*) Broken timbers in drift sets are ork from each end of a damaged section; neglected

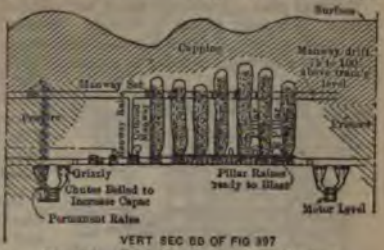

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Fig 398. Ray Hand-tramming System

rs. Large tonnages are produced per man; as tematically, the men soon become proficient. y small; the amount of raising and drifting other methods in similar orebodies. The systems are flexible as to output; practically any number of men may be used during active operations and a corresponding tonnage produced.

LONGIT SEC

stem, Drift-sets

Motor haulage is applicable only in orebodies of large area and of a height exceeding 100 ft; otherwise the cost of driving and timbering the stope laterals and of equipping and maintaining the haulage levels is not justified, and HAND TRAMMING is more economical. ADVANTAGE OF MOTOR HAULAGE: it is suited to large outputs, especially during reserve drawing, when the output is limulage and hoisting plants. ADVANTAGES OF HAND bodies: (*a*) lower initial development cost; (*b*) less , as chutes can be built in every set in both the r haulage, the pillar between the sub and motor

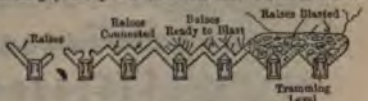

Fig 400. Ray Hand-tramming System, Undermining Pillars

t the end of 1915). This is a drawback of all shrink- compare with disadvantage *d*, Art 70).

l report of Ray mine for 1915 shows net cost roduced by the above systems; this includes and fixed charges. Development yielded 4% stoping, and 90% from reserve drawing. No le.

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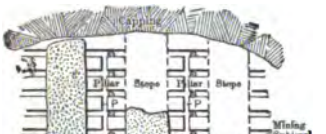
87. Practice at Miami Copper Mine, Ariz

Orebody (see Art 74). The mining system described below was applied relatively hard ground containing 2 300 000 tons.

General plan. Shrinkage stopes and pillar-caving were used; stopes, 5 200 to 500 ft long; pillars, 50 ft wide. Originally the pillars were to be r level stoping (Art 43), leaving broken ore in the stope, but it was difficult to holes, and a shrinkage stoping and sub-level caving method, similar to that (Art 89), was finally adopted. C simultaneously from both shrink stopes, to make the capping This is called locally the WID PILLAR SYSTEM. Following da lished by D. B. Scott in 1916 (



CROSS-SEC THRO STOPES AND PILLARS

Fig 401. Miami Method

Development comprises a ha drawing-off level, the sub-level and raises for handling ore and stopes (Fig 401). Added sub- was necessary to determine the the orebody; its general outline churn drilling (Art 11). HAULA 50 ft below the stopes. By drift long axes of both stopes and pillars (Fig 402), trains of cars could be loa entire stope without uncoupling; this allowed the rapid drawing of excess or during stoping. Also with chutes 25 ft apart, this plan requires a minimu development. 10 by 10-in timber was used in haulage drifts, with 9-ft posts and 8-ft caps; sets, 6.25 ft apart (Fig 445, Art 91). DRAWING-OFF LEVEL, a distinctive feature of this method, was 25 ft above the haulage level; it consisted of drifts parallel to those on the haulage level, with a fringe drift at each end for entry and ventilation (Fig 403). Drifts were on center lines of stopes to secure a symmetrical arrangement of raises for drawing ore (Fig 401). For convenience in putting up vertical development raises in the pillars the pillar drifts were 5 ft off the pillar axes; ment was later modified so that symmetrical drawing raises were possible also The drawing-off level drifts were timbered with 10 by 10-in sets 5 ft apart, usi

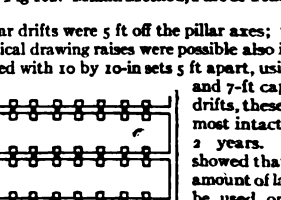


Fig 402. Miami Method, Part of Tra

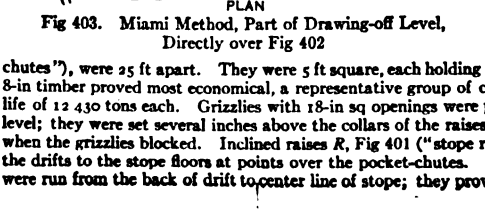


Fig 403. Miami Method, Part of Drawing-off Level, Directly over Fig 402

and 7-ft ca drifts, these most intact 2 years. showed tha amount of l be used o that it sho (2-in) and s in apart. ing at chute 406.) RAI raises *C*, Fig chutes"), were 25 ft apart. They were 5 ft square, each holding 15 tons; for c 8-in timber proved most economical, a representative group of chutes showin life of 12 430 tons each. Grizzlies with 18-in sq openings were placed on the level; they were set several inches above the collars of the raises, to permit e when the grizzlies blocked. Inclined raises *R*, Fig 401 ("stope raises"), were the drifts to the stope floors at points over the pocket-chutes. Originally, v were run from the back of drift to center line of stope; they proved difficult a

were 6 ft sq at the bottom and funneled at the top. 50 ft apart along center lines of pillars. During

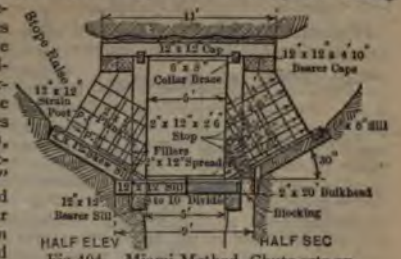
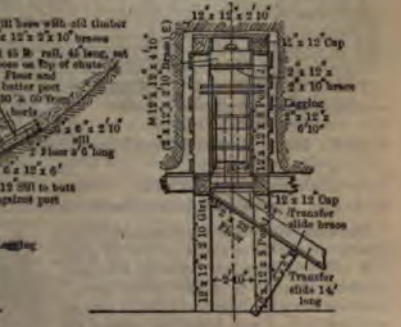


Fig 404. Miami Method, Chute-sets on Drawing-off Level

he pillar. The pony sets, of 12 by 12 timbers, were r for both drawing and repairs than 6-ft sets. SUB-LEVELS were opened in the pillars from raises P, at vertical intervals of 25 ft (Fig 401). Drifts were driven on the long axes of the pillars to the ore limits, with crosscuts 50 ft apart to the stope boundaries. This plan was selected as best, after trying several others. As originally planned, the Miami method called for continuation of the crosscuts across the stopes, but they were abandoned on finding that they could not be economically used as points of attack. DEVELOPMENT RATIO. Including all openings in stopes and pillars, 45 tons of ore were developed per ft of drift and crosscut, and 200 tons per ft of raise.

unneling the raises to a top diam of 15 to 20 ft; their le stope (Fig 405). Funneling was done by uppers



Sets, Chute-gates and Transfer Slides in Pillars

wn holes around the collar were unsuccessfully tried. e next squared up, and stubs between funnels blasted th the pillar crosscuts on the 1st sub-level, which 407, 408 show mode of breaking ground; for further Drilling began along the stope sides, as in Fig 127,

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Art 28. The swell of broken ore was 39%; enough was drawn to keep a under the back. Max height of stopes, 125 ft; in most cases, ore was bro ping, completed stopes being left full.

Pillar-mining began at the edge of the ore on the top sub, and progr and along the pillar (Fig 409), to provide a safe exit and insure even settle

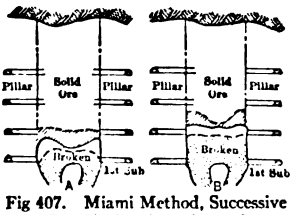
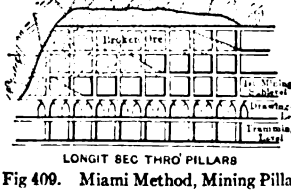


Fig 407. Miami Method, Successive Steps in Breaking Ground

(compare Kimberley method, Art took weight when the capping ov lodged, they had to be mined qu or 4 sub-levels were attacked sim working faces on adjacent subs b apart. In mining pillars, crosscu halfway between the original one-s drilling faces. After crosscuttir vanced on the top sub, the end undercut by uppers drilled outwar of the central drift and the end cro shoveling was done for setting broken ore, and long holes were Usually, no ore was broken on the highest sub nearer than 10 ft to the ca working faces sloping about 45° (Fig 409) were opened successively at the in lower subs. Fig 410 shows routine of drilling in retreating along a sub. The first round from the drift and crosscuts broke to about 12 ft above the sub; the 2nd round, for which long bits could be used, increased this height to 20 ft. Holes were often drilled to the broken ore above, but the remaining 5-ft arch usually caved. Attempts were made to drill 5-ft holes in the floor of the sub, to assist subsequent caving of this arch, but the ground "raveled" so badly that the collars of these holes had to be cased with 3-in pipe to prevent fitchering. These holes broke poorly and were given up. Excess ore was shoveled to raises *P*, Fig 401, and to intermediate raises in the pillars.

Fig 408. Horiz Sc Showing

raise was abandoned and covered by broken ore, lagged stulls were placed

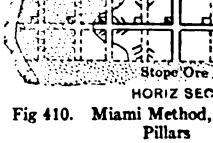


LONGIT SEC THRO' PILLARS

Fig 409. Miami Method, Mining Pillars

below the sub-level. This is imp mining; if raises were not cover would "pipe" down through th proper breaking. As each raise mining on the next lower sub, a s form was built, and the stulls ab out. Mining on the lowest sub the pillars, was subject to modifi by the pony raises, which halv between the regular 25-ft draw raises were funneled in retreatin of the pillar. Funneling was n than 25 ft ahead of pillar breal undercut the entire width of pillar. To protect drillers in case broken ore in the raise, machines for drilling the back were set on 10-in stringers acros

Drawing broken ore systematically began on completion of about 70% of all stope and pillar-mining; no drawing was permitted within 100 ft of any active mining. Drawing of a stope next to a pillar in process of being broken was tried, and caused the pillar to crush and sway. The weight on the drawing-off level during the drawing period varied greatly at different points; at first, some ground was very heavy, with apparently no weight in other places. Probable causes: (*a*) Combined height of capping and ore in northern part of sec 465 ft; in southern part, 350 ft. In the northern section, some drifts re retimbering; in the southern, only 20% needed repair. (*b*) In one small se



HORIZ SEC

Fig 410. Miami Method, Pillars

m; drawing-off drifts below unbroken blocks were
distribution of the weight. In a large area, where
breaking most complete, about 70% of the timber
ng after 3 years. (c) Weight on drawing-off drifts
e had progressed over a considerable area, probably
weight.
k little weight. The drawing-off level is advanta-
e level, sections of which were intact after 5 years,
vel drifts. The max repair cost in any drawing-off
drawn on it; the aver was much lower. Drawing
e surface between broken ore and capping dipped
ere reached first at western boundary and retreated
use even settlement of the capping. The ore ran
chutes in a plane at 90° to the direction of retreat,
pillar. In some stopes, the ore packed tightly in
tween stoping and drawing. To start such chutes,
from the drawing-off level halfway between the
ng were driven on each side of the drift to broken
e raises were funneled just before they broke into
loosened the broken ore above and usually that in
thod had the disadvantage that the pony sets took
. To start the flow of ore in packed chutes without
r the drawing raises was temporarily supported by
driven in solid ground, parallel to the long axis of
awing-off level. These drifts connected a series of
f the drift, vert and horiz holes were drilled across
tween the regular raises and left an arch of broken
wn the stope ore. No accidents occurred in work-

aver rate of drawing about 400 000 tons of ore into
-shift; monthly averages, 94 to 142 tons. These
yed in drawing. Under very favorable conditions
exceeded 200 tons. Aver cost of drawing this ore
labor and explosives. Where chutes hung up, the
ton.

nnage in a block of 1 700 000 tons. Detailed data
about 10% greater than from pillars; some pillar
he stope-chutes and vice versa. Extraction and
hardness of the ore; soft ores were more apt to
was greatly packed, drawing removed a cylindrical
ding to the capping. This tendency of chutes to
ing from the sides is greatest in soft ores which are
y contaminate large tonnages of ore with capping.
r unit of area tend to a higher extraction than fewer
-drawing tests at Inspiration mine, Art 80.)

r finishing drawing, some broken ore remained in
laries of stopes and pillars. Ore above the drawing
their backs, retreating from the ends of the drifts.
d, the output per man-day averaging 40 tons. To
ermediate drifts were driven between the original
-raises at 200-ft intervals were put up to these drifts
The drifts were supported by 12 by 12-in sets,
in each set. A shrinkage stope 20 ft wide was then
round was met or caving occurred. The ore was
e max tram being 100 ft.

ous Examples of Practice

Chile. Data from S. B. Gordy in 1916 and from
DIES, carrying chalcopyrite, bornite, etc, are large,
around a volcanic neck of tuff, about 4 000 ft diam.

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Some ore occurs in the tuff itself. The walls are often indefinite; on the sid
the tuff, values often fade gradually and stope limits are commercial. The
body is about 1 500 ft long; max width, 280 ft. The lenses dip about 70°, an
generally stands and breaks well. Pope Yeatman, in a report in 1911 (310)
the ore then developed averaged 2.7% Cu.

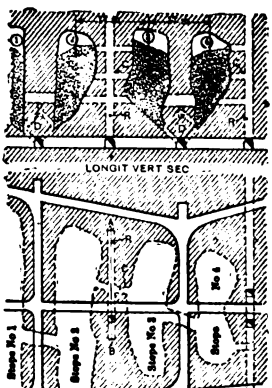
General plan. Wide parts of orebodies are mined by shrinkage stopes and p
The broken pillar ore and that left in the stopes is drawn as uniformly as possib

PLAN

Fig 411. Braden Mine, Development, Fortuna No 3 Adit

portion
in shri
along
reachin
to wall.
MENT.
adits
Fortun
vertical
275 to
later, a
terval o
was fou
for this
mediat
opened.
larger i
upper
lars fail

stopes could be completed. Adits followed the contact between tuff and an
when reached was outlined by crosscuts about 30 m apart (Fig 411), addi
being driven to facilitate haulage. Zigzag raises (Fig 412) were
put up between levels at points *Z*, Fig 411. All openings were
located for use in connection with stoping. Fig 414 shows a
simpler and more recent
layout, on an intermedi-
ate level.



PLAN OF LEVEL (WITH STOPE AND PILLAR LINES)

Fig 413. Braden Mine, Original Stoping System, Fortuna No 3 Adit

Shrinkage stoping. For data on breaking ground, see Table 26 and text, Art 28. The first stopes were opened as in Fig 413. Center lines of pillars and stopes were about 17 m apart; max stope width, 9 m; minimum width, in weak ground, 4 m. Crosscuts *C* (*cruzados*) from raises *R* gave access to stopes. Pillars were mined from drifts *D*. The arrangement of chute-raises in Fig 413 was uns
drawing depressed the surface of the br
the chute side, making it difficult to rea
for drilling. Also, for normal condition
too great an interval. Fig 414 shows a
these difficulties. Stopes about 7 m wid
by 5-m pillars, are found best for aver
bad ground, stopes are narrowed and pill
without changing the level workings. R
8 m c to c, slope 45° and are funneled

Fig 412. Br
Pillar

floor. Entry to stopes and ventilation are by zigzag raises and *crusados*; th
nect with the level above, and being near opposite ends of alternate pillars ea

ng space is kept open under the back. Stopes are above, where a final blast fills the stope.

shrinkage pillar ends g up small ground by vith heavy hen driven s undercut No data are f drawing. to secure h the over- mpled reg- rcial grade t above it, l sampling

from R. L. (282, 284). ng; width, nd is over- The ore break well RAL PLAN. of the ore- horses (Art e no horses ined block square-sets level No ichest in the mine.

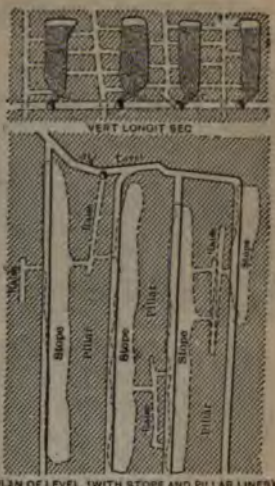


Fig 414. Braden Mine, Modified Stoping System, Fortuna 3¼ Level

Below level No 2, top-slicing are employed in the larger blocks (Fig 416). On he block, the timbers are removed and the stope left open. 5 by 7-ft drifts and crosscuts, 45 ft apart, are driven on level No 2, underneath the block, dividing the ore into 40-ft square pillars. Raises *R* (chute and ladderway), timbered with square-sets 2 sets high, are put up from drifts and crosscuts; one raise is carried to level No 1 for ventilation. From *R*, 16 ft above level No 2, a sub-level is opened by drifts and crosscuts directly over those on the level. These openings, 10 to 15 ft wide, serve as floors for shrinkage stopes of the same width, carried up to level No 1. Chutes *C*, over raises *R*, are for entry and for handling the excess ore. Chutes are lined with 3-in plank set on edge, with ends beveled at 45°. For data on breaking ground,

quence of Operations

opes cut the block into 6 pillars *P*, 25 to 30 ft sq pported on the sides by the broken ore. The tops

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of raises *R* are funneled (Fig 416) and the ore is drawn. The chute lining with the ore; end pieces and dividers and about half the side pieces are rec

Pillar-caving. The ore often contains small fractures, causing pillars *P* fall into the stopes during drawing. If it be known that the pillars will no are undercut before drawing. Dri cuts on the sub-level floor divide th pillar into small units, which are bl fore drawing begins. Broken pilla through inclined raises to the sub-le the openings on level No 2. If a been undercut and does not break drawing, a drift is run below it on l raise *S* is put up (Fig 416). The may then be blasted by holes dril raise, or in drifts and crosscuts d

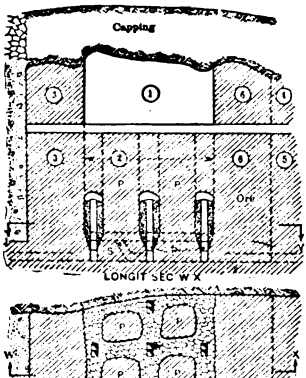

Fig 416. Duluth Mine, Pillar-caving in Block No 2, Fig 415

Mining other blocks, as prop Mine block No 2 (Fig 415) and then fill with waste to about 20 ft a back over level No 2. Fill the re and 2 by shooting down the cappin drill holes. After filling these bloc 3 by top-slicing. Simultaneously 4 and 5 like 1 and 2. Repeat this successive blocks along the strike.

Advantages of shrinkage stopi caving: (*a*) it is cheap, because o breaking ground and its small sumption. It saves about 20% per t as compared with top-slicing (Art tilation is good; (*c*) it is safe if pro

Requirements for pillar-caving: of large area; (*b*) orebodies with de aries; (*c*) ore that will stand unsupported in the stope backs; (*d*) ore of f grade, little sorting being possible underground; (*e*) ore which will not pack that will stand unsupported while mining the blocks. These requiremen dently rigid, the method in this form has a limited application.

New Jersey Zinc Co's mine, Franklin, N J. Data from H. H. Hodgkinson in 1916 (309). OREBODY is trough-shaped, of franklinite, willemite, zincite and calcite, in crystalline limestone; it has sides of unequal height and pitches at a gentle angle. The ore in the legs of the trough is 12 to over 100 ft thick, compact and tough, but not very hard (307, 98). GENERAL PLAN OF MINING. Entry is by a 965-ft vert shaft and a 1 500-ft incline. Level interval, 50 ft. A combination of shrinkage stoping and top-slicing is employed.

Shrinkage stopes, 17 ft wide, are run from wall to wall (120 ft), leaving 30 to 50-ft pillars. On completing and drawing a 50-ft lift, the stopes are filled with waste from the surface and lift above is started. Of the ore in pillars, 73% is mined by top-slicing and 23% by shrinkage pillar-stopes.

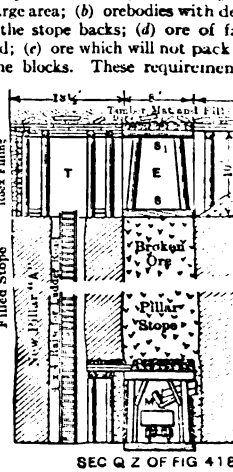

Fig 417. Mining Pillars, N

Mining pillars was started from the top downward, after the stopes at o orebody were completed (Fig 417, 418). A 10 by 8-ft crosscut *M* is driven

ed off at points 30 ft from foot and hanging-walls. a pitch of 50° to 60° to the mat above the pillar. to 5 ft long, are driven from the raises toward the e (pillar stope) is then carried from crosscut *M* to

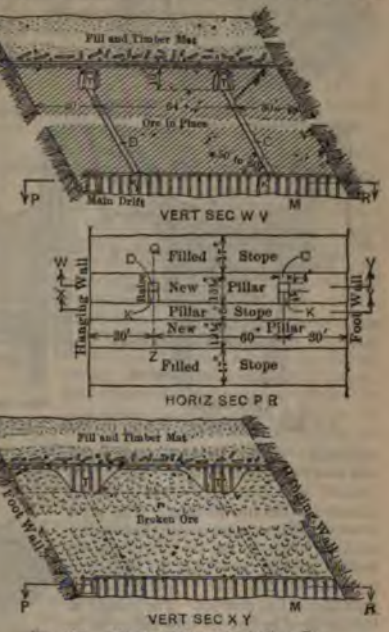

Fig 418. Mining Pillars, N J Zinc Co

op-slicing: (*a*) it is cheaper, because the sliced ore pillar stope without using barrows; (*b*) only 2 raises pplies); (*c*) production begins when the pillar stope nd the output is larger and steadier than if the same nined by ordinary top-slicing; (*d*) no preliminary rosscuts are needed in slices, as the pillar stope gives parts of the slice; (*e*) there is always a safe retreat anger; (*f*) ventilation is good. DISADVANTAGES: hitches for sills *S*, Fig 417, is slow and costly work; netimes break; (*c*) there is a loss of interest in the tied up in the pillar stope (Art 70). If disadvan- omes serious, the pillar stope can be narrowed to 5 ft iving a longer support for the ends of the sills. This ng hitches, but the pillar stope must be widened to tarting a new slice, which may cost more than the (See also *Trans* A I M E, Bull 130, Oct, 1917.)

ey Diamond Mines, South Africa

minated in volcanic necks (pipes) of "kimberlite," 307). The pipes are nearly vertical, of circular or Kimberley mine is about 500 ft. The kimberlite nd" or "blue." Following data and drawings were

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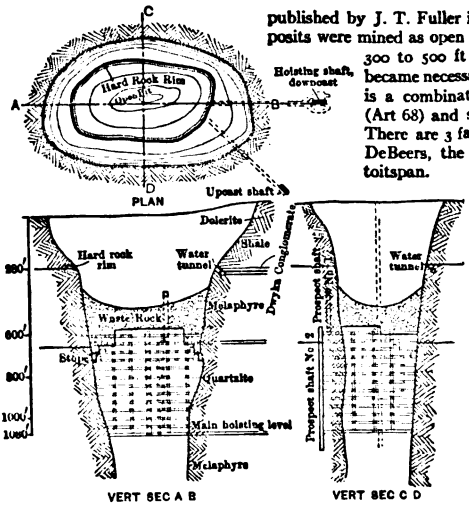

Fig 420. Kimberley Mines, General Development

published by J. T. Fuller in 1912
posits were mined as open cuts (A
300 to 500 ft before
became necessary. U
is a combination of
(Art 68) and sub-leve
There are 3 fairly dis
DeBeers, the Kimbe
toitspan.

Deve
421) is p
in all t
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ment) i
1 000 t
edge of
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opened
at verti
to 520
levels a
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vertically. As the workings deepen, a second shaft (upcast, Fig 420) is s
and safety (required by law). LAYOUT OF MAIN LEVELS depends on ty
1 080-ft level (Fig 421) shows a typical layout for electric haulage. The main crosscut always connects with the prospect shaft and runs to the far wall of the deposit. SUB-LEVELS (Fig 422). Drifts and crosscuts are driven on 22.5-ft centers or multiples of this number. The ground on the sub-level is first cut into pillars 90 to 270 ft sq; all drifts must be exactly over one another on successive subs. Short drifts 22.5 ft apart are driven to rim rock from the nearest drifts and crosscuts, which sufficiently develop the subs to allow stoping to begin. Sub-level drifts are 6 ft wide and 7 ft high, and rarely require timbering. Wider drifts in blue ground usually have 3-piece sets of round timber averaging 10 in diam, and lagged with 2 or 3-in plank. WINZES, 6 to 10 ft diam, connect the subs and haulage level (Fig 420–4 chutes and ladderways, and furnish adde for fast work. Blue ground from driving su through the prospect shaft until the passes chute-gates have been erected in the main

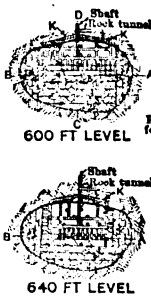

Fig 421. Plan of Levels

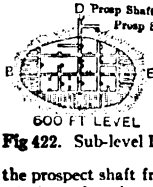

Fig 422. Sub-level Layout

Stoping methods in the 3 systems have s
mon. Ore is broken in shrinkage stopes, in
back may be caved. Stoping on each sub
ends of the deposit toward the central cr
the prospect shaft from the wall opposite. The order of making the va
relations of work on different subs, are shown in Fig 421. The ground
often treacherous; if so, the first stope on a sub must be narrow, followi

on any sub are at right angles to the central cross- last 2 or 3 cuts, short drifts *K*, Fig 421, are driven ts extended to connect with and dumped down a special chutes and ladderways are re covered with heavy tim- d forms a layer of waste on 1912, the Kimberley pit was f waste between its bottom

Fig 423. Plan of Sub-level, Mode of Starting Stopes

). Work begins at the pipe are widened until they con- out on a slope of 30° to 45° (the angle of repose essive slices are taken from the stope back, from 40 to 60% of the broken blue being left in the stope to support the men. A pillar, 5 to 10 ft thick, is left at the top of the stope (see *CD*, Fig 424). If the floor of the sub-level above this pillar is covered with waste, all the broken blue is loaded out and a new slice started at the bottom of the inclined back. The shell of blue over the stope eventually caves, allowing the waste to run in and furnish support for further slicing. If the overhanging pillar is covered with broken blue, not recovered on the sub-level above, the pillar is shot down and the ues until the blue is so mixed with waste that its abandoned and another started in front (Fig 425). be adjusted so that its back will not crush prema-

, requiring much blockholing, and increases admixture with bout 250 ft long, a length now y pipes, partly due to their re- great pressure of the overlying these mines, 100-ft stopes are ases of premature caving, the ig 430), or the stope is aban- ble from the level below. Ex- b-level interval (Fig 426).

Fig 425. DeBeers System

Principal features: (*a*) horiz ways (pole roads) for access to ll vertical pillars (back-pillars) he waste while loading out the . Stopes are started as in the tem, from wide drifts along the are carried as flat-back shrink- Pole roads are semi-circular out 3 ft diam, in the vertical irectly over a sub-level drift. Their stope side is vertical boards, supported by horiz poles about 6 1 to 4 ft apart, set in hitches and held in place by n blue against them. Pole roads are kept up even face of broken blue in the stope. Usually there are per stope, as in Fig 427; a third is advisable in the ng stopes, for an escapeway in case of a sudden cave. Kimberley and DeBeers systems the stope backs ot at an even height throughout, for reasons given itspan method. The height of stopes, and mode of the top of the stope and the sub-level above, are Fig 428 shows work after the stope along the wall

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is abandoned and caved, and a second stope is nearly finished. Bac
thick, are being thinned down as the top of the stope is approached (Fig
pillar is to be blasted out, holes are also drilled to shatter the top of the

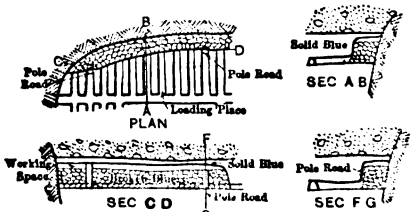

Fig 427. Kimberley System

supports the en
hanging the st
inrushes of w
blue is loaded o
the waste behi
tends to crush
the stope. Th
lar which is left
partly recovere
in" (Fig 430),
level below. O
can be complete
of waste cuts
broken blue (F
cases: (*a*) the waste may be loaded out, if in small quantity; (*b*) the st
doned and the remaining blue recovered from the sub-level below; (*c*)
be run through the waste to the remaining blue (Fig
430). Fig 431 shows the Kimberley method on
several sub-levels; note that the stopes on any sub
come up under stumps of back-pillars on the sub
above.

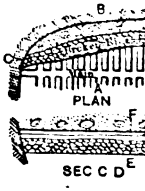

Fig 428. Kim

Dutoitspan system. The pipe has a much greater area than the others; hence the stopes may be long. The Kimberley system, tried first, proved unsatisfactory because: (*a*) caving and crushing of stope backs 250 to 400 ft long was dangerous, and difficult to control; (*b*) miners could not keep such long backs at a uniform height. Premature caves, especially near the middle, often cut the stope in two, stopping ventilation and causi
(*c*) pole roads did not provide adequate ventil
must be completely emptied before another can
of it. Hence the output varies widely. To me
the method shown in Fig 432 (from W. W. M
It combines features of both of the other system
tion of the stope backs (10° to 15°) is parallel to t
stope, instead of across it, as in the De Beers system; this strengthens the back and prevents premature caving. Pole roads are put up from every 4th or 5th sub-level drift. In these stopes, the high end may be loaded out and abandoned, while the other end is advancing; a 2nd, 3rd, and at times a 4th line of stopes can be started before the 1st is completed, thus equalizing the tonnage from one level.

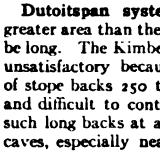

Fig 429

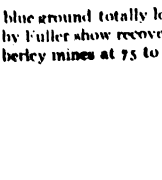

Fig 431. Kimberley System

When a stope reaches the boundary of the pipe, its last 100 ft is completed rapidly by the Kimberley system. Ventilation in the stopes is excellent.

Mud rushes. Most of the waste accumu
pit disintegrates rapidly on exposure to air,
forms it into a fluid mud, which works dow
tually bursts into the mine. Mud rushes h
fatalities, and great expense for clearing out
are now prevented by a system of drainage
with a water tunnel running around the pit
water is pumped to the surface.

Extraction varies widely in different stop
blue ground totally lost in one level may be recovered from below. Lo
by Fuller show recoveries of 40 to 100%. He puts the aver recovery in
berley mines at 75 to 85%; in Dutoitspan, Wesselton, and Bultfontein

Cost at Dutoitspan, for year ended June 30, 1907, was about 62¢ per ton of blue, including development, mining, hoisting, and transport to the weathering "floors" on the surface; in this period 68 916 ft of development were done. Cost in 1911 for all mines was about 76¢ per ton, which is low considering the large amount of development.

For further detail of Kimberley methods, see Bib (308).

DERGROUND MINING

llaneous Methods

applicable to: soluble minerals, as salt; sulphur some kaolin deposits which can be broken by, and neral, the method is used for impure deposits, or r where shaft sinking would be very difficult or its exploitation of otherwise unworkable deposits

es (311). Salt beds occur in sedimentary ervious strata and often interstratified with lescribed in Art 32, 42, are appropriate for g cover; otherwise, boreholes are used.

salt and cased. The casing may be perforated, e salt; a smaller pipe is placed inside the casing, a rod or air-lift pump, and the salt recovered by nty or impure, fresh water may be pumped down nular space between the pipes, or the direction of lves out chambers in the salt, which may connect if so, fresh water is sent down one hole and brine he holes cave, are abandoned and new ones bored.

Salt occurs in lenses and layers, interbedded with ickness of formation, 470 ft; depth below surface, h surface soil with 8 to 12-in pipe; thence, 4 to 6- es, 2 to 3 in diam. Some companies force water mo of a well's operation; thereafter, fresh water the inner pipe. Other companies take brine from he bottom where the densest brine collects. The gr, 1.13 to 1.15. KANSAS salt measures resemble ich of the salt is won through boreholes. A 5⅝-in er 2.5-in pipe nearly to the bottom of the measures. As salt is dissolved, the surface exposed to solu-

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tion increases and more water may be used. An old well will produce abou minute, carrying 2.19 lb salt per gal.

Frasch process for mining sulphur, Union Sulphur Co, La. T about 100 ft thick, is covered by 450 ft of clay, quicksand, and lim underlain by a great thickness of sulphur-bearing gypsum. Water in sand contains H_2S. Deposit is nearly circular in plan; area, about

Following description is based on data published by R. H. Vail in 1912 (one of the Frasch patents (U S patent No 870 620, Nov 12, 1907). Boreholes groups, and spaced 50 to 100 ft apart. 10-in casing goes to the sulphur for hole is continued about 130 ft with a 9⅞-in bit and an 8-in perforated liner is bottom. Then a 6-in pipe, reduced to 5 in through the liner and perforated at is inserted. A 3-in pipe, also perforated at the base, reaches to the bottom pipe and a 1-in pipe to the bottom of the 3-in pipe. Water, superheated pumped down the two outer annular spaces between pipes; this melts the sul collects at the bottom as a liquid with a sp gr of about 2. After 2 or 3 days, co is forced down the 1-in pipe, which, with the 3-in pipe, forms an air-lift pump 23); the molten sulphur rises through the annular space between the 1 and 3-in pipes are galvanized or coated with aluminum. The sulphur-bearing formatic and, as the hot water pumped into the holes runs away underground, its pressu utilized with certainty for raising the sulphur. Large quantities of sawdust into some holes to fill the crevices. About 7 million gal of water were used 1912; 130 boilers (25 000 hp) heated the water, drove pumps, etc; fuel co about 700 bbl oil per day.

Daily output per well, 200 to 500 tons; 1 well produced 73 000 tons before flow. Wells are eventually lost by caving of the ground, which may break fill the chambers at the bottom. Before abandoning a well, the casing is dy quicksand, which runs in and fills the chamber below. For further detail, see

Mining kaolin through boreholes, West Cornwall, Conn (313) occurs in vein-like residual deposits, dipping 50° and intercalated wit broken quartz, feldspar, and mica. As the mixed material could not be ordinary methods and the kaolin washed out at a profit, boreholes we

Holes were cased with 4-in pipe to within a few ft of the footwall of the depos the 4-in pipe was placed a 2-in pipe, terminating in a cap with several nozzle-lik Water under 40 to 60 lb pressure was forced down the 2-in pipe, the bottom of allowed to sink to the footwall as the surrounding material was broken by t washed up the annular space between the pipes. Depth of wells, 50 to 198 ft. matter contained 20% kaolin; the overflow of the wells carried 5 to 10% solids, 60 to 75% kaolin. The percentage of pure kaolin in the solids was inversely p to the velocity of the water current.

Mining gilsonite (Sec 1, Table VIII, under Asphalts). When he certain temperature gilsonite flakes off without melting, a property u mining this mineral in overhand stopes. Steam or hot air jets a against the face under attack, and, as the gilsonite flakes off, it is ca hopper and loaded by gravity to cars below (U S Patent 950 363, Feb

91. Chutes and Chute-gates

Chutes (mills, mill-holes, ore-chutes or ore-passes) are used chiefly and shrinkage stopes, Art 59, 68; also in square-set stopes, Art 47. vertical or inclined passages through the stopes, for handling ore The term also designates chute-raises (Art 68), chute-compartments and chute-gates.

Requisites of a chute: (*a*) a life at least equal to that of the stope (*b*) minimum tendency to clog; (*c*) a cost small in proportion to total passed; (*d*) low maintenance cost; (*e*) suitability to the stoping conditi

Details of construction are important as they affect the life of the chute, w limits the max allowable level interval. The chute's life depends on the am

d the pressure due to the stope filling. The ton- width, interval between chutes, and distance be- ffected by the cost of handling ore in stopes and iven spacing (Art 47, 60–68; also Raises, Art 71). connection with the initial and maintenance costs

The former are preferable, as they are easier ill pass a larger tonnage for a given amount essary in narrow, pitching veins, or near an posits; if provided with a ladderway, they

igh empty chute is very destructive. Chutes the wear on the lining then depends on the and hence practice in Center Star es, Chapin

has a rec- cribbing or In veins less nes built by cular chutes Baltic mine, steel chutes ustralia.

the arching ss-sectional nax diam of rably more.

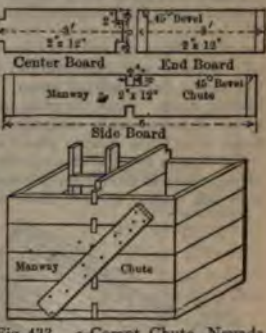

Fig 433. 2-Compt Chute, Nevada Wonder Mine

ually 4 by 4 ft to 5 by 5 ft. The minimum ft; better, 2.5 ft.

val, the chute and ladderway should be of equal e interchanged when the chute is worn out. In d stopes, manways are usually provided only in ry 2nd or 3rd chute. Manways are advantageous ready access to chutes when they clog or need re- s. See below and Art 64 for sizes of circular chutes.

ectangular plank chutes cost less than others; the e form is shown in Fig 433 (see also Art 60); the lerway is sometimes omitted. The first few cribs ve the stulls at the level are of 3-in plank. The es of the planks are toe-nailed together. While fill- is being placed around them, the chutes are held osition by diagonal cleats, as shown. The planks cut, notched and beveled by a swinging circular saw). Plank chutes serve for small tonnages, as in row veins, or where ore is sorted and only a small ount is saved; they will not withstand heavy pres- es from filling in stopes with weak walls.

Cribbed chutes are in general use in the U S for large nages, and for supporting heavy pressures in filled es. Fig 434(*a*) shows open cribbing; 2-in spreaders f vertical plank. Fig 434(*b*) and (*c*) show close and not be lined. Timbers with grain at right angles

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to the direction of falling ore wear faster than when the grain is parallel
Unlined chutes must be heavy enough to last as long as the chute is nee
they are difficult to repair or r
Fig 281, 282, Art 63, for close-cri
Fig 435 shows round cribbin
5.33 ft sq, either single or dou
ment. Crib timbers are 10 t
and have an aver life of 3 mont
work, 6 days a week; hemlock
because of its toughness. Th
of worn chutes are lined with
plate; above that, with 3-in
Though round timber is cheaper
of it are harder to erect and to

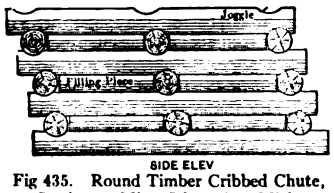

Fig 435. Round Timber Cribbed Chute, Section 16 Mine, Ishpeming, Mich

Cylindrical wood-stave chutes (Fig 436) are preferred to cribbed chutes in large filled stopes at Broken Hill South mine, N S W (314). The level interval is 150 ft. In cribbed chutes (Fig 434), the lining cut out rapidly; the spikes holding it also failed and the planks ripped off. Repairs were excessive and even 4-in lining was inadequate. Cylindrical chutes worked better. They are built in 4-ft sections, of beveled staves. Since the grain of the staves is vertical, the cutting action of falling ore is minimized. The staves are bound with wire hoops while filling is being placed around them.

There are 4 types of chute in this mine: (*a*) chute of 4-ft internal diam, with 10-in staves; (*b*) 3-ft chute with 10-in staves; (*c*) 3-ft chute with 6-in staves; (*d*) 3-ft chute with 5-in staves; 5-in thickness is the minimum for sufficient bearing between staves and for resistance to collapse. Type (*a*) is used for the lower 35 ft of both vertical and inclined chutes (its large diam minimizes clogging); type (*b*) is for vertical chutes, for the sections from 35 to 70 ft, when t ft; type (*c*) is used from 70 to 120 lifts, and from 35 to 70 ft on 100-ft (*d*) is for repairing rectangular ch have failed. The upper 30 ft of the mined by underhand square-settin and hence chutes are not required. Staves are of eucalyptus, sawed from roun timbers. At first, dowels, as in Fig 436, were used to hold the staves togethe could be wired, but as this construction was too slow small dogs (cramps), of 0.5-in iron, are now employed. Successive lengths of chute are simply stood one upon another, as they do not tend to move laterally after being enclosed with filling. Fig 437 shows an inclined stave chute. Fig 438 shows a connection between vertical and inclined chutes; the loosely hanging rails yield to every blow and take wear at the turn.

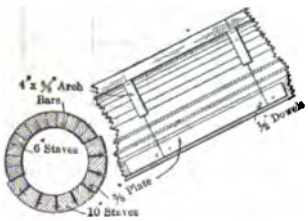

Fig 437. Inclined Cylindrical Chute

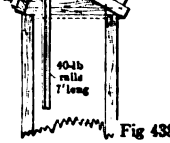

Fig 438

Stave chutes are marked by low first cost and good wearing qualities; they may be withdrawn on finishing a stope. They are serviceable for lining raises and passes for handling filling. E. J. Horwood stated in 1916 that an essential condition for their satisfactory use is that the pressure be approx equal on all sides; hence they are not applicable in stopes having a h ing-wall (242). Table 58 gives the cost of stave chutes, compared with rectangu chutes at Broken Hill South mine. Eucalyptus costs $56.40 per M bd ft; r Oregon pine, $45.60; 2 by 10-in Oregon pine, $48 per M. See Bib (314) for d

utes, Broken Hill South Mine, Australia

te	Cost per linear foot			
	Material	Labor	Total	
	$ 9.35	$0.75	$10.10	
	7.35	0.75	8.10	
	4.40	0.35	4.75	
	3.70	0.35	4.05	
	8.60	1.00	9.60	
10 by 10-in *OP*	11.80	0.20	12.00	
10-in *OP*, 2 by 10-in				
	10.00	1.00	11.00	
10-in *OP*	9.50	0.20	9.70	

OP = Oregon pine
SB = Eucalyptus ("Stringy Bark")

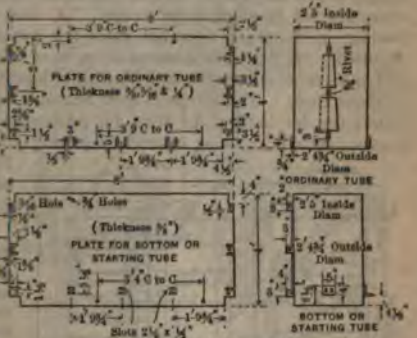

Fig 439. Cylindrical Steel Chute

tes is prohibited. Clogged chutes are loosened by a le by boring a 2-in hole in a piece of 5-in shafting and r to fire a projectile at the hung-up ore. ADVANTAGES chutes: (*a*) moderate first cost and excellent wearing mall size permissible without undue tendency to clog, he chance that men may fall into them; (*c*) adaptabil- onditions, as no special care is needed when blasting them; (*d*) their impervious structure, which prevents and keeps out sand filling (a difficult matter in wet e and cheapness of erection, each section fitting the one mine the steel chute worked as well as any timber otion in any case depends largely on the relative cost ber.

steel chutes (14 by 14.5 in) are used in filled square-set ennial Eureka mine, Utah, for handling high-grade ore fines. They are of $\frac{3}{16}$-in plate, bolted to 1.5 by 1.5 es at the corners and on the outside.

tes may be started: (*a*) by climbing up the chute and hot which will unkey the clog (dangerous work, which ibited); (*b*) if there is a manway compartment, by he chute; (*c*) by using a "cannon" (see Steel chutes); ays effective and causes delay; (*d*) by a 1 by 1.25-in ron hoops on one end. Extra lengths of rod are lashed nt the rod from catching. The hanging ore is poked

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until it falls, or a stick of dynamite is fastened to the rod and exploded at point. This device can be used up to a height of 100 ft (316).

Fig 441. Chute-gate, Homestake Mine

Chute-gates, at bottoms of chutes, control the flow of ore in loading into ca design and details vary widely. Followi points are important: (*a*) The cheapest that will do the work without undue m cost and delays due to clogging is the b cost should be considered in connection wi tonnage passed; elaborate designs are warranted only by large tonnages. (*b*) Gates must be suited to the size of the ore. Simple board gates serve for fine ore; other forms are necessary for handling large lumps. (*c*) For rapid loading, mechanically operated gates are sometimes required. (*d*) Strength is essential, as wear is heavy and gates must often withstand blasting of boulders in or above them. (*e*) Clearances must be ample, to provide room for barring, to allow for settlement of supporting timbe prevent men from being injured by getting pinched between chutes

2" pieces spiked on
Fig 442. Chute- Pelican Min

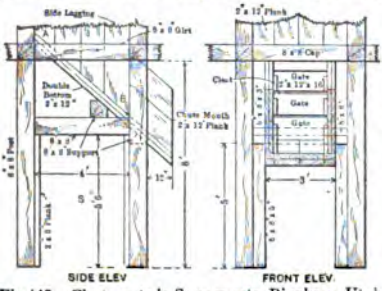

Fig 443. Chute-gate in Square-sets, Bingham, Utah

Following examples are found good under differ tions; see also Bib (31

Simple chute-gates The gate-boards are he cleats; small chutes may board, but 2 give better co flow of ore and facilitate large lumps (Fig 399, Art a larger chute with 3 bo small size material the l *B*, Fig 441, is pried up wi bar; notch *A* is a conven place for board *B* when is running freely. Fig 4 simple chute supported o

der a narrow shrinkage stope; chutes for mill holes in filled stopes are similarly Fig 443 is a chute for square-set stopes (66). The bottom of such chutes is c idly, especially by hard siliceous ore, when the chute is not kept full. Linings of plank, old rails or steel plate, are useful. Wear is reduced by offsetting chute (Fig 445), so that solid rock forms chute bottom. In square-sets, same result is obtainable by lagging chute sets on level and omitting bottom boards *AB*, Fig 443; space *S* then fills with broken ore, which stands at its angle of repose above chute lip and takes the wear. Fig 444 shows a gate and manway in a shrinkage stope (Utah Copper Co), where stope floor is opened at the cap of sets on level (122). See also Fig 404, 406, Art 87.

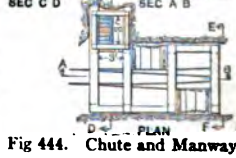

Fig 444. Chute and Manway, Utah C

All the above chutes are designed for loading fine ore into small cars, as in h ming. Board gates for large cars on motor-haulage levels may be similarly s

equired for headroom and for clearance between the conditions practice favors pony-sets for supporting 87, and Fig 395, Art 86).

ed in the Miami mine on motor-haulage levels for he gate ("guillotine" type) is a steel plate, sliding the pony-set posts, and e is rarely as large as (154).

Raise
6 x 8 x 6 ft Cribbing
Sprender 2" x 10" x 5'
Spike First Two Lagging to Sprender for Cross Sill
Lagging
Cap 12" x 10" x 9'
½ Leg Screws
Top of Rail
C.L. Raise
Center of Drift to Center of Raise

ig 445. Chute-gate, Miami Copper Co, Ariz

hout damaging timbers; (*c*) the ore settles evenly, the stope instead of from one side; (*d*) rapid drawing is possible, as several cars may be filled simultaneously. DISADVANTAGES: (*a*) higher cost; (*b*) the platform is only 5 ft above the rails, thus reducing headroom for trammers; (*c*) boards *A* are not easily moved when covered with ore; (*d*) the level timbering must be higher above the track, which involves more shoveling in opening a stope.

A modification of the Chinaman chute (see dotted lines in Fig 446) is used at the Copper King mine, Ariz. A 2-in cleat *D* is spiked to the inside of each platform stull; on *D* 2 by 6-in stringers *E* are fitted between the stulls; on *E* loose crossboards *B* are laid. Two of the boards *A* are permanently removed, and the drawing is regulated by boards *B*, which are easily removed and replaced. At this mine, the platform was raised to give headroom in the drift (83, 317).

"Bulldozing chutes" (Fig 447) are used in the Alaska-Gastineau mine, under shrinkage stopes where the ore contains many large apart; pieces passing the grizzly readily go through in the bulldozing chamber. The 2 pockets together balanced it is easily operated by 1 man standing on

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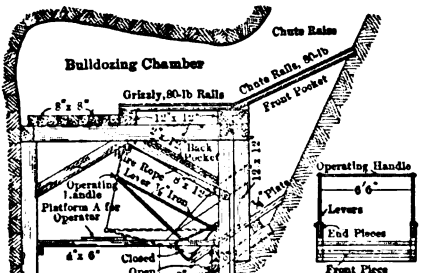

Fig 447. Bulldozing Chute

platform *A*, out
slide gate (not sho
above the arc gate
There are 2 gate
bulldozing chambe
tons of ore were
through one of th

Arc gates (Fig 44
for moderately fin
operated by hand
compressed air cy
DERSWUNG GATES
rising through the
are generally prefe
which cut down th
are more easily cont
apt to jam or allow
into the drift. Ar
ing more than pla
not justified for sm

Finger chutes. Fig 448 shows a standard chute (105), for handling l
of relatively coarse, dry ore. The fingers, held in place by the weight of a

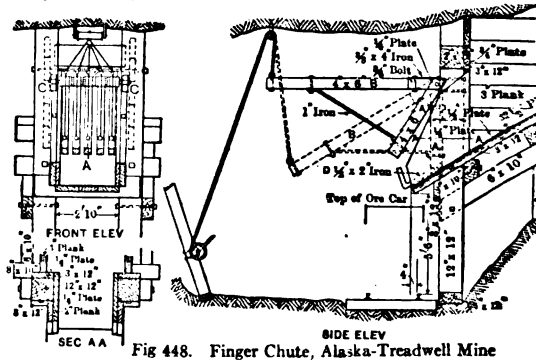

Fig 448. Finger Chute, Alaska-Treadwell Mine

separately hung from rod *CC* so that they move independently. Short ro
arms join a main rope, passing over a roller to a small windlass. To draw or
are raised high enough for it to pass; when drawing is completed the fingers fall by gravity and stop the flow. If a large piece catches on the lip, it will hold up only 1 or 2 fingers, the rest dropping to their normal position. To prevent leakage when the chute is not in use, a tail board is placed across the lip in the angle irons *D*. Cost of a chute-raise, with a finger chute complete, is as follows (R A. Kinzie): 12 ft of raise, @ $9 per ft, $108; 80 cars of dirt, @ 20¢, $16; cutting station for windlass, $40; 40 cars of dirt, @ 20¢, $8; building chute, $1
700 ft B M, @ $12.50, $8.75; iron and steel, $40; blacksmith work, $7; ro

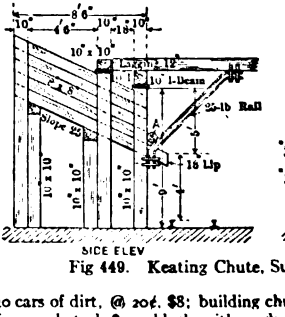

Fig 449. Keating Chute, Sudbury,

es: quick loading, freedom from clogging and from prohibitive except for large tonnages; in two of the eplaced by simple chute-gates. Fingers of bent rails

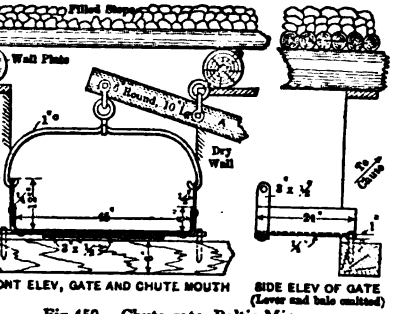

Fig 450. Chute-gate, Baltic Mine

mill holes, by the Copper Range Consol Co, Mich. the edge of the car when loading. It is inexpensive and well adapted to the Baltic method (Art 64), because, as the mill holes are used alternately as chutes and ladderways, the gates must be dismantled or installed when the change is made.

fesabi

Hanging chutes (Fig 451) are used on Mesabi Range while mining the lowest sub-level in top-slicing (Art 71, 72). They have the advantage of holding several cars of ore in a short vertical distance, and so avoid interference between miners on the sub and trammers on the main level. The sub-level track is carried on short drift-sets, which also support the rds, rests on 30-lb rails, which run lengthwise of the the rails hang from the caps of small sets just high evel. The chutes are sometimes 20 ft or more long, by removing some of the bottom boards (35).

ng Ore in Flat Stopes

as the breast stopes of S E Missouri (Art 30), apen LOADING costs by employing mechanical . Various devices are also used to cheapen he levels. This problem is especially serious ncrease operating costs and often add to the he level interval (Art 20). Minimum dip on 33° to 45°, depending on character of ore and o° cars may be run directly to the stope faces; ort is difficult.

a. A THEW SHOVEL (Sec 3, Art 8) operated by "sheet-ground" mine (Art 30), at Carterville, Mo. s enclosed in an armored housing to protect it dur- weight, 12 tons; clearance of boom, 12 ft; swing dipper with bottom open, 5.5 ft; dipper capacity, nted on 4 wheels with wide tires, running on plan-

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laid on the stope floor. Table 59 gives a cost record; 2 men ran the shovel and Feb loaded buckets were trammed by hand 400 ft to a shaft; mule used in Mar. As the ore-body was only 12 to 15 ft thick, the shovel had to move frequently to get its ore. (See Art 30 for contract prices for loading buckets by hand in this district.) The shovel was installed not so much to reduce costs as to overcome difficulties caused by an insufficient and irregular labor supply. The MYERS-WHALEY machine, made at Knoxville, Tenn, for shoveling and loading, is oper pressed air or electricity. It runs on the mine tracks and is self-propelling. end is a toothed shovel or scoop, which is thrust into the ore pile, lifted and d ward onto a conveyer, which delivers to an inclined conveyer loading into rear. The shovel is supported on a jib, which swings sidewise so as to cle ore within a 10-ft radius on each side; the conveyer is carried on a sim allowing the use of a double loading track behind the machine. E. H. Lesli the data in Table 60 on work in breast stopes at Flat River, Mo (Art 30). was driven by a 22-h p motor, 1 man controlling all its movements. Th

Table 59. Cost of Operating the Thew S

		Jan	Peb	M
	Tons loaded	2 778	3 238	3
Cost per ton	Shovel men	$0.0459	$0.0493	$0
	Conveying to shaft	0.0665	0.0521	0.
	Compressed air	0.0161	0.0191	0.
	Oil and gasolene	0.0020	0.0024	0.
	Repairs	0.0040	0.0086	0.
	Total	$0.1345	$0.1316	$0

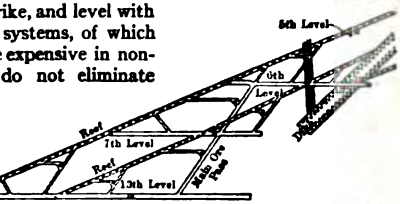
Table 60. Work of Myers-Whaley Shoveling Machine (3

Federal Lead Co, Flat River, Mo, Shaft No 1		St Louis S & R Co, Fla Shaft No 4	
Tons shoveled in 27 days	6 962	Tons shoveled	
Total time (2 8-hr shifts), hr	432	Total time, 67 shifts, hr	
Delays inherent in machine, hr	79.4	Delays inherent in machine,	
Misc delays, due to local conditions, hr	137.3	Misc delays, due to local co tions, hr	
Net running time, hr	215.2	Net running time, hr	
Aver tons shoveled per hr	32.3	Aver tons shoveled per shift	
" " " " shift	129	Labor: runner, sledger, shifter; 4.3 men per shift	
Total repairs per month	$60	Labor cost per ton	
Repairs per ton shoveled	1¢	Repairs per ton, labor and ma	

charges at the St Louis Smelting & Refining Co's mine include part of cost and repairmen's time in going to and from the machine. H. A. Guess comments on this machine, as used by the Federal Lead Co, as follows: handling about 7 000 tons per month since Feb, 1913, on 2-shift work. to be profitable in stopes less than 20 ft high, as the small rock piles in necessitate frequent moves, thus reducing efficiency. Even in high sto per ton of loading probably averages no less than that of hand work. especially useful during labor shortage, or for increasing production. Myers-Whaley machines were installed in the Crown mines, Ltd, Transv 14-ft drifts, on the 13th and 16th levels (*E & M J*, Jan 16, 1915, p 150). shoveler, resembling the Myers-Whaley, is made by the Lake Shore E Marquette, Mich, and has had a limited use in drifts and flat workings i Superior iron mines. See Bib (320) for description of this and other loadi

Methods of handling ore in flat stopes: (*a*) By costly shovelin footwall. Ore from the upper parts of a lift may require severa before it reaches the level. Shoveling is better adapted to small w grade ores. (*b*) By using intermediate levels and raises, to which shoveled. Lateral transport on intermediate levels may be in

te system at the Crown Mines, Ltd, Transvaal about 300 ft apart; chute-raises (ore-passes) that the max shoveling distance in a stope is for tramming to chute-raises; their tracks are ike, and level with systems, of which expensive in non- do not eliminate



452. Development at Crown Mines, Transvaal (*E & M J*)

ing, but as passes cost money for construction onomic balance in their frequency." (See also k City, Utah, *E & M J*, July 7, 1906, p 12, and) (*c*) By tracks laid on the footwall at an angle in to the stope faces (Fig 179, Art 39). This is vith regular footwall, and dip of less than 10°, but (*d*) By breast stoping to the dip (Art 31). (*e*) ry or shaking chutes, or other stope conveyers. e transport is limited to open stopes in regular where hanging-wall is heavy, requiring close irregular shape and dip (20).

, Empire mine, Grass Valley, Cal. Ore occur- tar mine (see below); dip is about 30°.

(Fig 453), near middle of oreshoot, intermediate drifts 100 ft apart. At *A*, just above the highest drift, are vith a brake. One drum has a friction clutch for regu- of rope to suit the different levels. The raise is double- nsheets at the intersections with drifts. Cars lowered ets by gravity dump automatically into chutes at the To give room for loading, there must be a minimum n between top of car and hanging-wall; hence planes le in veins less than say 4.5 ft thick. For details of ee Sec 11, Art 17.

" **planes,** North Star mine, Cal. For ore occur- 17 and Note F, Table 29, Art 28; aver dip of O. Kellogg (322) gives following data.

ft cuts the vein at 1 630 ft (4 000 ft on the dip). t follows the vein to present bottom of mine (about levels are 333 ft apart; from them, stopes are opened s (Fig 454, 455). Beginning at a level, the first stope the raise and to the limit of the oreshoot, the ore being ts laid over the level track. After making 2 or 3 cuts, is built, at least as high as a car. If the dip is steep ars by gravity. When the stope face has advanced full length. Then the pillars next to the chute are t about 5 ft from the face. These are lagged to retain de, the first intermediate track is laid (Fig 455) and s work at the 6th lift.)

, is lowered to the main level by double-track pl-

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("go-devils"). The headblock (Fig 456) comprises 3 sheaves, with a t
block between them, applied by the lever to all three sheaves. The he
pended at the top of the plane by a bolt through a post. A ⅝-in rope
cu-ft cars. The trammer fills a car on a lateral track, trams it to a t

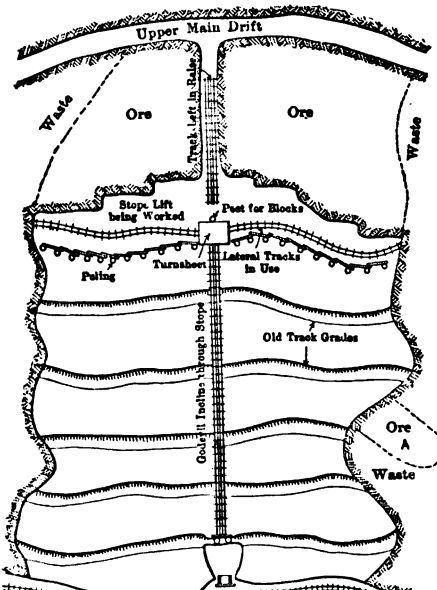

Fig 454. North Star Mine (stope, showing successive lifts)

intersection w
hooks on the r
car over the c
sheet on to the
and grasps th
The descendi
pulls up the
track is 20-in
rails on 4 by 6
cars in a sto
product of 4
10 to 15 go-de
daily product

Stationar
open steel-p
of curved o
cross-sec, lai
floor, or on
small timber

Ore is sho
chutes at the
to them in ba
Rand (Art 17
circular chut
ore on dips o
is about the
at which ore w
ter dips are po
is thrown or d
chute, thus h
momentum.
are in 10-ft
wide, of ⅛-i
tangular chut
by 5 in deep, a

to 34°, are used at Golden Cross mine, Cal, for dips of about 30°, as d
raising at this mine in Art 22 (116). Similar chutes are used for cleanin
Mohawk copper mine, Mich (Art 40). Stationary chutes are cheapest
and simplest. During blasting they may be protected by piling ore on
each side. In a few cases on the Rand, a small stream of
water has been used to assist movement of ore, when chutes
are nearly at the critical angle.

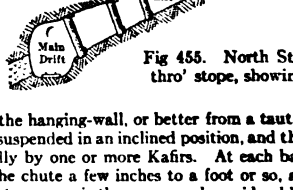

Fig 455. North St
thro' stope, showin

Shaking chutes, widely used in flat stopes on the Rand, are best liked of the stope conveyers, and are especially suited to the local conditions.

They are shallow, sheet-iron troughs, of curved cross-sec, hung by chains from eye-bolts in holes in the hanging-wall, or better from a taut turnbuckle (Fig 457). The chute is suspended in an inclined position, and th into it while it is swung longitudinally by one or more Kafirs. At each ba the ore moves by its inertia down the chute a few inches to a foot or so, a almost continuously. In large, flat stopes, ore is thus conveyed considerabl two or more chutes in series; also, the successive lengths may be placed at

sive, with large capacity and
; chutes from eyebolts: (*a*)

678 by a ... of ... dikes ... some- ... inch

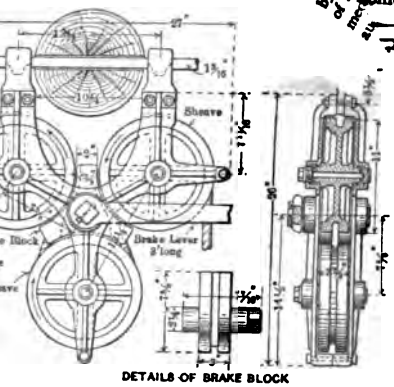

Fig 456. "Go-devil" Sheaves

and, chute lengths are 8 ft long by 18 in wide, of
slips into upper end of the next one, and is fastened by 2 bolts fitting loosely in slots, to allow flexibility. As the stope face advances, the chute is lengthened, to keep within shoveling distance (325). Fig 458 shows a chute of ¼-in plate, reinforced by straps (324). On slopes less than 8° or, 10°, ore will not move forward efficiently. See *Jour* Chem, Met & Min Soc of So Af, Oct, 1916, for note of a crank device giving quick forward acceleration and slow return, for chutes on very flat dips; it is operated by a small electric or compressed-air motor. See Combined stopes (Art 41) as to having stope faces parallel to the dip, so that chutes may be set at the max possible slope. Long chutes may be swung by an air cylinder mounted on a

aking

ated respecting shaking chutes in the New Kleinfon- labor depends on the length of conveyer; it varies 4 ft, to 3 men for 32 ft (325).

supported
on frames.
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re to travel
This is more
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).

enderson-
the Geld-
veraging 7

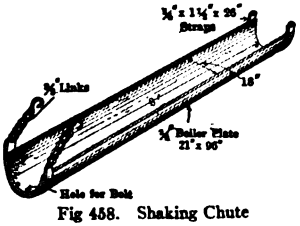

Fig 458. Shaking Chute

a small pocket, for loading cars on the level; upper
ny desired point in the stope, and the rope is taut-
nd bucket descend by gravity, and are pulled back

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("go… small air or electric hoist on the drill column. This device is suit… block… y length, but, for spans exceeding 100 ft, the rope must be support… point by a clamp attached to the arm of a drill column. The … automatically. The rig can be dismantled in 15 to 20 min, and erected in… irrespective … Moving the … upper end tak… During blastin… removed and … (326). Two t… sometimes use… in balance. T… has separate c… otherwise buc… at one place … shows a sim… overcome this… vented at Ne…

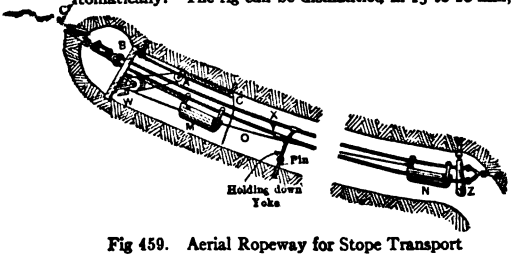


Fig 459. Aerial Ropeway for Stope Transport

mine. *X* and *Y* are the main ropes; *Z* is a swinging stop, to take up shoc… are side-tipping trucks, holding 4 to 10 cu ft. Haulage rope *O* passes aro… wheels *W* and over pulley *A*, which traverses rope *BC* and so varies workin… This conveyer will work in a vein 45 in wide, and on dips of 15° to 35° (3…

Miscellaneous devices. ELEVATED MONORAILS AND TRUCKS. The W… system, proposed for levels and very flat stopes on the Rand, has not b… Difficulties in supporting the rail and cost of installation offset the adva… decreased friction. BARROWS WITH GROOVED WHEELS ON MONORAILS (3… veins at dips up to 25°, E. M. Weston used a 2-cu ft wooden Cornish … grooved wheel running on a wooden rail; or for regular dips to 18°, a 3-cu … on steel rail. Under rear end of the body is a grooved block, for guidi… in sliding on the rail. Wheel is a hardwood disk, with slightly larger ½… on each side, forming a groove to fit the rail. It will run on the stope … smooth. Several barrows may operate on 1 rail, an empty being tipp… on meeting a loaded barrow.

83. "Sand Filling" of Stopes, Metal Mines (See also …

General. At some of the Transvaal gold mines, tailing from … (here called "sand") is utilized for delayed filling (Art 59) of oper… 41). The sand goes underground through boreholes, or pipes in s… carried to stopes in launders or pipes, water being added as necessa… PULP which will flow on the grades available. BARRICADES along … areas to be filled retain the sand; the water drains off and is pump…

Conditions on the Rand which led to sand filling: The area of exhaust… very large; as depth increased the hanging-wall often caved, causing seriou… timber is costly; ore, low-grade; and ventilation became difficult in dept… ing these conditions, sand filling is an efficient and relatively cheap proc… circuiting of air currents through old stopes is prevented, stope and le… be recovered, and incidentally the storage of tailing on the surface is o… operation differs only in detail from FLUSHING, as practiced in Pennsylva… coal regions (Art 110). Note use of mill tailing for contemporaneous s… Australia, Art 63, 66, and at Baltic mine, Art 64.

Principal problems in sand filling: treatment of sand to des… salts, which may form HCN by contact with acid mine waters, t… of tailing from mill to stopes, and construction of barriers to ret… during filling. Following notes are from papers by E. Pam, W. … O. P. Powell, B. C. Gullachsen, R. E. Sawyer and others, on prac… 1910 and 1916 (251, 329).

Preparation of stopes. Fig 460 shows typical filling operat… cades are erected in spaces between level-pillars (box-holes), ar…

possible, work is so arranged that faults, dikes etain filling. On flat dips, the sand is some-

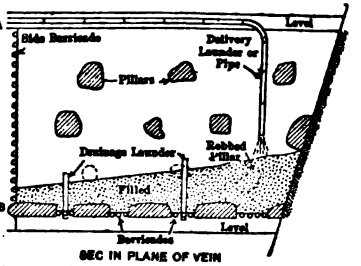
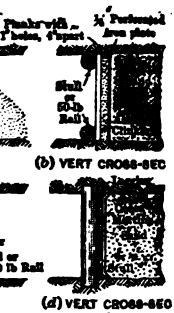


Fig 460. Sand Filling, Transvaal Gold Mines

dam
filled
pulp
on of
posi-
e ex-
from
free
port.
d for
ough
oken
re is
the

g and permanent; the water should be isolated and s of barricade in Fig 461 have been successful; (*a*) and (*c*) are common at bottom of open stopes; (*b*) is built in box holes; (*d*), less expensive, is used on sides of filled areas; see Examples below.



Treatment of sands. Cyanide salts break up in tailing lying on the edges of old dumps, exposed to air and sun; such material may be sent underground without danger. But, to save cost of rehandling, it is best to use tailing direct from the cyanide vats ("current tailing").

nd Filling on the Rand

g treatment to destroy cyanide salts. $KMnO_4$ and sed, the amount depending on strength of the last ture content of the sand from the vats. At Simmer 12% moisture, carrying up to 0.02% total cyanide. sed (5% solution) per ton of sand. Frequent moistering the borehole, which has never contained more o traces of HCN have been found in stopes being O_4 from decomposing pyrite, some alkali (as lime) ry mine, in 1916, ashes from coal were found to conthis purpose. Slime should be removed from sand tion in the mill or in cones erected outside the mill. the drainage layers of barricades.

. See Examples of practice below.

Boreholes for this purpose are 6 to 10 in diam to equalize cost of lateral handling on surface eep is nil. Clogged holes are easily reopened stream of water. Sand may also be conveyed erway (note use of a wooden box for fairly dry). The sand is carried laterally to the stopes pipes or open launders. Launders are best if

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Fig 462, published by W. A. Caldecott in 1914 (329), shows the launder carrying Rand tailings in pulps of different fluidity; it is platted from followin

$$W = \frac{12}{G-1} = \frac{P}{100-P}; \quad P = \frac{100\,W}{W+1} = \frac{1\,200}{G+11}; \quad G = 1+\frac{12}{W} = \frac{1}{}$$

where W = ratio of water to solids, by wt; P = per cent of water, by wt; of launder, per cent. These formulas are based on observations (within the

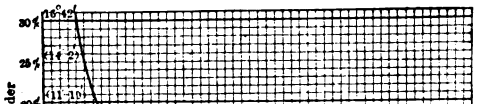

Fig 462. Launder Grades for Sand Pulp (Rand)

ted) upon under large ing condit refer to o sands, conta 4% pyrite; 10% of th retained o aperture s exact grade many factor eral, the con ing to requ launder gra water ratio volume of p sharp curve wrong shap

internal surface of launder, unevenness of grade, large size of sand particles a centage of pyrite. Conversely, a large volume of pulp flowing in a well-desig launder without curves, and containing much slime and few coarse or pyrit requires less grade than is usually needed for sand pulp. In installing a l desirable to increase the grade by say 20% around curves, as well as for some distance at the head of the launder, to overcome the initial inertia of the pulp " (Caldecott).

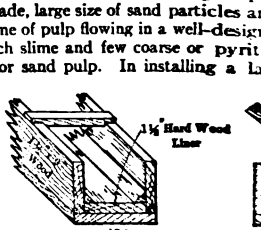

Fig 463. Launders for Handli Underground (Rand)

Fig 463 shows FORMS OF LAUNDER. The simple V launder (*a*), with hardwood liner, is good and easily shifted. Type (*c*), with concrete liners 3-ft long, has been used in the Robinson mine. Launders are run on the floor of the level or hung from the roof; they may terminate near the top of the stope or be carried down the dip (Fig 460). PIPES, required for moving sand horizontally along levels, wear fast and are turned to distribute wear. Ordinary C-I, steel, and wood-lined pipe, and some wit lining, have been used; wood-lined pipe was popular at first, but gave trouble little used. Steel pipe is preferable to that with special linings and in flat long life; at Village Main Reef mine, an passed 75 000 tons of sand before the we siderable; vertical pipe wears fastest. ping and before starting the sand, clear through the pipes for a few minutes, to g ging. If the depth is great and the flow pipe too rapid, the pipe may be broken hoppers every 300 ft. See Art 110 for horiz length of pipe through which sa forced by the pressure in a vertical pip

Fig 464. Drainage Launder for Sand Filling (Rand)

Draining sand in stopes. Water runs off from the sand through pipes or in barricades at the sides and bottom of filled areas. Drainage launders laid on the footwall and extending through the barriers, remove water rapidly

Cost of sand filling varies widely (Table 61). Filling operation be continuous rather than intermittent. If current tailing is used,

the water content of pulp should be as low
t.

' Sand Filling, Witwatersrand

ver tons owered r month	Aver cost per ton, cents			Per cent of cost on surface
	Surface	Underg'd	Total	
19 171	6.73	13.11	19.83	33.9
19 490	4.64	9.22	13.86	33.5
27 387	7.64	7.48	15.12	50.5
25 815	3.55	15.29	18.84	18.8

from E. Pam). The plant handles 200 tons of r conveyed to a brick lined bin (500 to 600 tons y 8-ft tunnel runs from side of bin to the shaft; l discharges through 30 by 18-in holes into which unnel slopes 20%; sluicing water at 50 lb pressure /6-in nozzles, the mixture in launder being 35 to the sand flows down a 5 or 6-inch unlined pipe, ions can be readily turned and replaced. Pulp es; on the 2 000-ft level, a 600-ft length of horiz v of sand can be stopped only at the surface, a re prepared (Fig 460) with bulkheads composed which fits close to floor and roof, all crevices being off top of filling through 12-in apertures in the ked up as the sand level rises. Area to be filled ce pressure on barriers and give free overflow for stopes, as well as old ones. Barriers impervious ce, due to its incompressibility, any water left in ral pressure; such barriers are not used elsewhere (Caldecott and Powell). Current tailing in form m of water containing $KMnO_4$ through a tunnel is 4.5 by 6 ft, 1 125 ft long, and runs from sand borehole 390 ft below surface; it has a vertical in diam at surface and 7 in at 1 729 ft depth. ater. Underground handling is like that at the K MINE (Caldecott and Powell, A. R. Hughes). a mixing box 10 by 5 by 4 ft, where water contain- parts water to 1 part sand by wt) is pumped by pipe, 900 ft to diaphragm cones erected on surface underflow of the cones, containing 30% moisture, with a 30% grade. Two 8-ft cones, handling 200 me. At bottom of the borehole a wooden launder, lp to stopes, more water being added. In 1910, y 9-in plank, supported by 8 by 8-in stulls; planks oa matting. Box holes at bottom of stope were them, through which a 6-in pipe projected into (Fig 464) ran from the pipe to a point 2 to 12 ft der was covered with cocoa matting filter frames. inker filter bed (Fig 461). Sand filling is carried oles. CINDERELLA CONSOL MINE (R. E. Sawyer). eyed to shaft by mechanical haulage and dumped y 11-in cross-sec. The conduit gave no trouble ntained less than 5 to 6% moisture; wetter sand oked it. The conduit, 3 900 ft long, discharged on stream of water washed the sand into a launder; l to the stopes by pipes or launders. Capacity ctually reduced pumping, as sand sent down aver- pes retained 10%. This system is stated to be e, as the plant is small and pumping is

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by sending down dry sand. WITWATERSRAND DEEP MINE. 22 000 to of current sand is sent underground, part down a borehole and part c practice is similar to that at Simmer and Jack mine. Filling entering 28% moisture; pillars and hitherto inaccessible ore are being mined under

94. Choice of Underground Metal-mining Metho

General. The selection of a method of mining for a given orel by making an inverted statement of the limitations and applica methods already described, but no concise statement can be frame the variations that occur in nature. Usually several methods are adapted to the size, shape, and attitude of the orebody, and th its ore and wall rocks; from these a choice may be made or a me which will best suit the geological, economic, and local conditio Table 62 groups the mining methods for this purpose; it sugg available under ordinary conditions, each of which may be studied with peculiarities of the orebody in question. A mining method i from all viewpoints; it is generally a compromise between confli

Cost and extraction. A low-cost method yielding a low ex give greater total profit than a more perfect method which reco proportion of the orebody.

The following equations (H. L. Smyth) show relations between the fa

$$\frac{p}{p'} = \frac{Q - X}{Q} \quad \text{or} \quad \frac{X}{Q} = \frac{p' - p}{p'}$$

, where Q = total tons of ore in a deposit the most perfect method; X = number of tons abandoned by any other profit per ton by method Q; p' = profit per ton by the other method; p per ton effected by the other method. When $p'(Q - X) = Qp$, the tw equally desirable. The proportion of the deposit which may properly be s

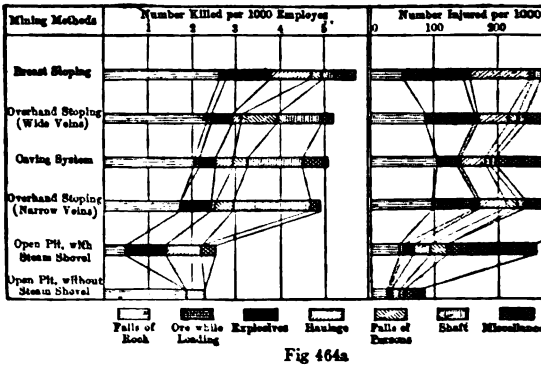

Fig 464a

fore depends on the ratio of the saving to the profit per ton. This rat the profit diminishes, hence for a given saving a larger proportion of ore high value may be sacrificed (172).

Calculations like the above may be elaborated by varying the millin smelter charges on ores of different values; in such cases, the effect of the the point of max profit may often be determined graphically (Fig 86 and

Safety. Few data are available as to the comparative safet mining methods.

Fig 464a shows statistics collected by U S Bureau of Mines for the ye based on returns from 258 mining companies, open-cut mines (Art 95)

f Underground Metal-mining Methods

ength walls	Possible methods of mining	For details, see Art No
Stg	Breast Stoping	30, 31
	Systematic Room and Pillar	32
	Open Overhand Stopes	38 to 41, 92
	Coal Mining Methods	102 *et seq*
Stg	Breast and Bench	30
Stg	Mitchell Slicing System	56
or Stg	Same as for Masses	.....
or Wk	Resuing	61
Stg	Open Underhand Stopes	33
	Open Overhand Stopes	36 to 41
	Filled Flat-back Stopes	60, 67
	Filled Rill Stopes	66, 67
	Shrinkage Stopes	68 to 70
Stg	Breast Stoping to Dip	31
	Systematic Room and Pillar	32
	Open Overhand Stopes	38 to 41, 92
	Leaning Stope Sets	58
Wk	Filled Flat-back Stopes	60, 67
	Filled Rill Stopes	66, 67
Stg	Open Underhand Stopes	33
	Square-set stopes	46, 55
	Filled Flat-back Stopes	60, 67
	Crosscut Method	65
Wk	Filled Rill Stopes (Calamon)	66
	Top-slicing	71 to 75
	Filled Square-set Stopes	46, 55
.....	Same as for Thin Beds	.....
Stg	Open Underhand Stopes	34
	Underground Glory Hole	35
	Pillar and Chamber W'rk'gs	42
	Sub-level Stoping	43
	Open Square-sets	46, 55
	Shrinkage Stopes	68 to 70
	Combined Methods	84, 69
	Filled Square-set Stopes	46, 55
Wk	Filled Flat-back Stopes	62 to 64, 67
	Filled Rill Stopes	66, 67
	Combined Methods	84
	Filled Square-set Stopes	46, 55
	Crosscut Method	65
Wk	Top-slicing	71 to 75, 83
	Sub-level Caving	76 to 79, 83
	Combined Methods	88
Stg	Open Underhand Stopes	34
	Top-slicing	71 to 75, 83
	Sub-level Caving	76 to 79, 83
	Block Caving	80 to 83
	Combined Methods	84
.....	Same as for Masses	.....
Stg	Underground Glory Hole	35
	Pillar and Chamber W'rk'gs	42
	Square-set Stopes	46, 55
	Filled Rill Stopes	66, 67
	Shrinkage Stopes	68 to 70
	Combined Methods	84 to 89
or Wk	Filled Square-set Stopes	46, 55
	Crosscut Method	65
	Top-slicing	71 to 75, 83
	Sub-level Caving	76 to 79, 83
	Block Caving	80 to 83
	Combined Methods	84 to 89

= Strong. Flt = Flat. Stp = Steep.

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for comparison. The figures for overhand stoping in narrow veins are fr
panies working veins less than 15 to 20 ft wide, chiefly in open stulled sto
amount of work in underhand, filled and shrinkage stopes is included.
overhand stoping in wide veins are from 61 companies and include square-se
and filled stopes, and a little underhand work. Figures for breast stopi
41 companies, of which 22 work Missouri lead and zinc mines. Data on ca
include work by top-slicing, and sub-level, block, and pillar caving, in 53 mi

OPEN-CUT MINING

95. General

Open cuts ("Open workings," "Open pits," "Open casts") are surface
in connection with coal mining they are called STRIPPINGS. (For quarries, s

Field of use is in mining orebodies that outcrop or lie under sha
The upper parts of narrow, rich veins may be thus mined, with small
and no expense for development; this is a favorite "poor man's
obtaining capital for subsequent work, regardless of ultimate econon
open cuts, involving extensive development and equipment, are o
on the outcrops of wide, veinlike orebodies, because they produce o
than is possible with underground methods. Veins can be worked by
only to a limited depth, largely on account of danger of falls of wall r
walls and flat dips decrease the possible working depth. Open cut
only profitable method of mining thin, flat beds, with shallow cove
99). Large bed-like masses, covered by rock or alluvium, may often
cheaper by open cuts than by underground methods (see Mesab
Consol, etc, Art 97). Large orebodies that can be excavated in
hillside benches are well adapted to open cutting (Art 96). An or
seldom be entirely mined by open cut; underground work is usuall
in depth, and in some cases to mine the ore around the edges of the c

General plan. Open-cut methods are combinations of loosenin
and transporting earth and rock in surface excavations (Sec 3, 5).

The combination selected depends on the shape, size, and depth of pit,
raphy, and output required. Output and method should be adjusted to the
body, to secure minimum cost of production, including interest and amortiza
capital for equipment and stripping. STRIPPING may be completed before
mining, or mining and stripping may proceed simultaneously, when a sur
has been uncovered to avoid interference between the two operations; the
is best, as it reduces the initial investment required before production begins.
may be done in one or more slices, depending on the depth; stripped materi
disposable within a reasonable distance, and at points where dumps will not
subsequent mining. BREAKING GROUND. Open-cut faces are usually worked
(for details, see Art 96-101). Hydraulicking (Art 99, and Sec 3, Art 10) m
for breaking down unconsolidated material. In general, ground is broken
open cuts than in stopes, because of the large faces and use of heavy blasts.
is by hand or mechanical excavators, depending on the scale of operations;
holes (Art 100) are feasible, chute-gates are used. TRANSPORT (see Sec 3, 5)

Classification. An arbitrary grouping of open-cut methods, based on
loading ore is used in Art 96-100 for presenting details. For small-scale work
and shovel or plow and scraper, see Sec 3. See also under placers, Art 120.

96. Open-cut Work with Hand Loading of Ore

Field of use: (a) in small pits, where total tonnage does not wa
first cost of mechanical excavators (Art 97, 98); (b) in large pits,
location or other features of which make mechanical loading or
methods (Art 100) undesirable, or (c) large-scale work, where labor is

e generally carried in benches (Fig 465, 466), he factors outlined in Sec 5, Art 5. With sev- g 466) of each should be sufficient to provide e rocks falling from above. Some foreign laws t for safety; this is usually exceeded.

one quarries (Sec 5), or in large underhand stopes illed by hand or by machine drills. Deep churn- asts (Sec 5) are apt to e blockholing for hand- buckets or stone-skips rt methods from pit to d depth of pit, and the ollows:

often mined underhand uckets on skids on the to a haulage drift (Fig as were thus worked to loose slabs of wall rock,

n hillsides are easily s one method. A long ore being shoveled into ates hand-tramming or cars, without delays or nill or shipping bins, to r to a gravity plane for opography usually fur- dumps, as at *D*. The ceeds the economic max y to workmen from falls ght of face. Hence the rith tracks on each for transport; this involves the ccessive benches.

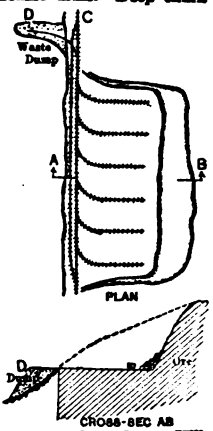

Fig 465. Open Cut on Hillside, Hand-loading

Examples. At ERTOCITOS, Cana- , Mex, there are 3 aches, which in o were each 80 to ft high. Ore is led into 0.5-ton ; the track lay- is like that in Fig , but with a single ge-track connect- e lowest bench is ing to bins at the s dumped down a e at the 2nd bench, as by aerial tram- Art 62) mines oxi- e up to 30 ft high; afford numerous ut. Ore from all a haulage tunnel nd operations. If s, it is trammed in

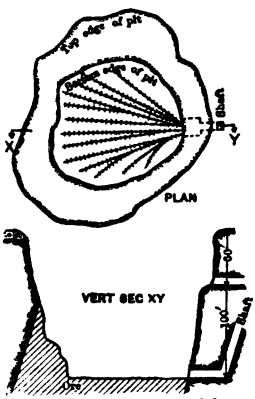

Fig 467. Open Cut, Creighton Mine, Ontario

ds about 13 cu yd and trams it 50 ft (178). EISENERZ, igh and wide, have been opened on the face of a moun- of siderite. The lowest benches are worked by steam

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shovels (Art 97), but about 67 % of output is mined by hand, and trammed int
Fig 466, connected by raises with a haulage tunnel *H*, leading to R R bins at
In parts of the mine, surface self-acting planes (Sec 11, Art 17) connect the be

Deep pits of small area require hoisting apparatus. ENGINE PLANES (
17) may be built on one wall of the pit. Cars serve for dips less than 30°;
skips are better. See Anthracite stripping, Penna (Art 97), for large-scale
Art 121. SHAFTS may be sunk near the pit and connected with it by crosscut
shows early work of this kind at Creighton mine, Sudbury, Ontario. The shaft
has 2 skipways
track crosscuts
the orebody at
160-ft levels, wi
tween levels and
face. The pit
as a glory hole
on reaching a le
bottom was ke
radiating tracks
from the cross
Blasted ore fell
floor, where it
into 1.5-ton cars
this pit was ab
diam by 160 ft
CABLEWAYS AN
allow extraction
deep pits witho
nary undergroun
ment. Fig 468
application in a
450 ft long, 300 f
over 200 ft deep
Ore or waste was
hand on the pit
2.1-cu yd stone sk
were picked up
ways or long-boo
and placed on trucks on surf
near rim of pit (353). Der
121 and Sec 5) and cableway
Art 23) have comparatively
pacity.

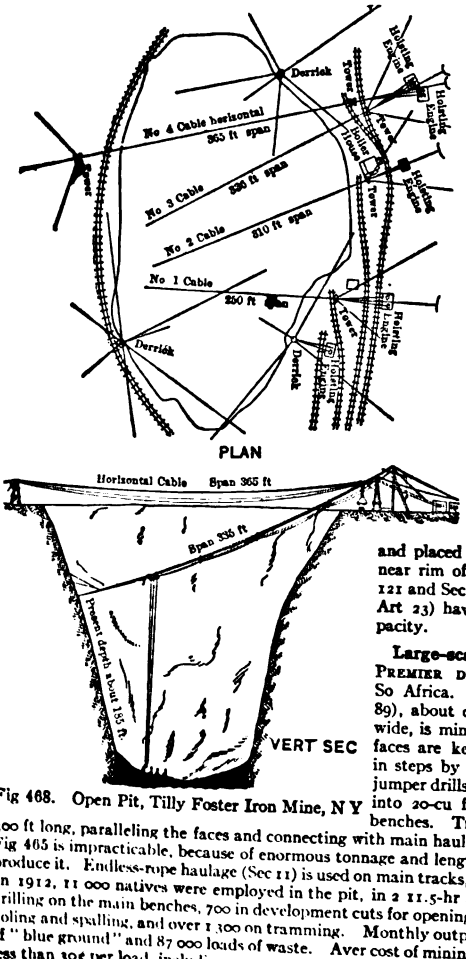

Fig 468. Open Pit, Tilly Foster Iron Mine, N Y

Large-scale work, chea
PREMIER DIAMOND MINE, T
So Africa. A kimberlite "pi
89), about 0.5 mile long by
wide, is mined in 50-ft bench
faces are kept steep, and ar
in steps by vertical holes dri
jumper drills. The material is
into 20-cu ft cars at the foo
benches. Tracks are in loop
300 ft long, paralleling the faces and connecting with main haulage tracks. The
Fig 465 is impracticable, because of enormous tonnage and length of benches nec
produce it. Endless-rope haulage (Sec 11) is used on main tracks, and on inclines to
In 1912, 11 000 natives were employed in the pit, in 2 11.5-hr shifts; 2 100 per s
drilling on the main benches, 700 in development cuts for opening lower benches, 60
holing and spalling, and over 1 300 on tramming. Monthly output, 1 000 000 16-cu
of "blue ground" and 87 000 loads of waste. Aver cost of mining and haulage was
less than 30¢ per load, including waste. The pit in 1912 was 200 ft deep; it is ho
the large area and strong walls of the pipe will allow open-cut work to 1 200 ft (3

lining with Steam Shovels

hallow overburden from coal, iron ore, phos-
by other open-cut methods; (*b*) stripping and
d tonnage, which are fairly uniform in char-
d may or may not be covered by overburden.

ods involve large capital outlay, for equip-
hence they are limited to work in orebodies
ıy plus interest during the life of the mine;
nous outputs at low unit costs.

vel mining appear simple; but good management,
tic work are essential to success. The shape and
termined, to allow intelligent planning of approaches
elays and handling a max amount of ore by the
rack, cars and shovels are also important. As the
ries inversely with the output, there should be no
ımples; also Art 101 and Sec 3, 5.

ows a method employed by the Hartshorn and
for mining a 4-ft horiz coal seam, overlain by 20
of shovel:
om at 45°,
ft, at bot-
66 ft from
3.5 cu yd.
l seam; the
tarting from
rt from each
The uncov-
y hand into
floor of the

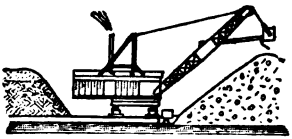

Fig 469. Stripping a Thin Coal Seam, with Revolving, Long-boom Shovel

o 1 600 or 1 800 cu yd per 8 hr; to keep pace with
required for breaking and loading coal (334).
N Y for flat beds of Clinton iron ore, 22 to 24 in
or shaly limestone, in turn covered by 10 to 13 ft
be loosened by blasting before using the shovel.
e drilled with Keystone drills (Sec 9) ahead of the
t and the same distance back from the face; each
mite (335).
ilarly mined. They occur as nodules and boulders
Beds are nearly flat, averaging 1 ft thick, and yield-
re; they are overlain by alluvium. Steam shovels
overburden, taking side cuts about 20 ft wide. A
bucket, stands on the surface on the unmined side,
and loads into R R cars. At times, the phosphate
buckets, and handled to cars by the crane.

ove method is the cheapest kind of open-cut
rating costs are low, as there is no expense
al to waste dumps, and the shovel output is
incident to loading into cars. Revolving
o 90-ft booms; height of dump, 33 to 65 ft;
93 ft; reach from pivotal center to center of
l is not adapted to thick or inclined deposits.
ction of shovel capacity, depth of stripping,

. E. van Barneveld in 1912 (35). See Art 11 for
GENERAL PLAN OF WORK. Orebodies are thoroughly
e contours are platted on maps, on which complete

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plans for stripping and mining are laid out before excavation begins (Art
burden is stripped by steam shovels, loaded into 5-yd side-dump cars or sta
cars, and hauled to waste dumps; ore is excavated by steam shovels and
standard R R cars to Lake Superior ports. Small pits are usually stripped
before mining begins; in large pits, stripping and mining proceed simultaneou
stops and stripping continues in winter. Mesabi open pits vary in area fr
up; in 1912, the Mahoning-Hull-Rust pit covered over 400 acres. The
problems involved in opening and operating these mines are largely those of
and the arrangement of tracks and approaches; track layouts are often com
presence of layers of paint rock, etc, and the necessity for grading the ore as
APPROACHES. As the topography is fairly flat, approaches are excavated
cuts on descending grades; a 2% compensated grade is the max desirable;
grades are required. Sometimes separate approaches are cut for stripping a
for stripping, they are located to minimize the haul to waste dumps; for mi
must be a track system which will reach the max amount of ore without u
grades. STRIPPING faces usually have a 1 : 1 slope, with a 20-ft berm at
of the bank. In deep overburden, the banks are broken at least every 75 f
with a 25 to 30-ft berm. Stripping is so arranged that shovels work a
to 25-ft bank; a 12-ft bank is the economical minimum. On small co
to 90-ton shovels are used; 100 to 120-ton shovels for larger operations.

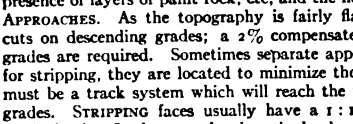
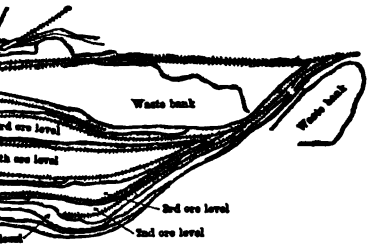


Fig 470. Spiral Track Layout, Mesabi Range

done with side cuts just as wi
graded in R R work. Regula
stops about 8 ft above the or
remaining cover is removed b
"clean-up" shovel, with a cre
laborers. As the top surface
bodies is uneven, final cleaning
scrapers; small revolving sh
also proved useful. The ove
mostly glacial drift, containin
from 2 to 12 ft diam; large b
"chained out" when uncovered,
holed. Overburden is loosene
the shovel by blasting; gop
(Sec 5, Art 7), 15 to 25 ft ap
to 25 ft deep, are drilled at the base of the bank and pointed downward 1
usually 5 to 8 holes are fired together. From 5 to 15 two-man gangs are kep
this work. Layers of taconite are broken with lines of vertical holes, 8 to 1
from the face and 6 to 10 ft apart, drilled with jumpers (Sec 5, Art 10).
DUMPS are located on barren ground, or where open cuts are impossible. A
sloping away from the initial dumping point and offering a downhill haul
Dumps are started by dumping alternate cars on opposite sides of the track, a
ally jacking up the track until the dump reaches the required height. After
dump is "fanned out" by moving the track sidewise. Usually, dumps a
from trestles, 20 to 25 ft high and strong enough for the empty, but not t
train. Loaded cars are dumped singly, as they reach the unfilled part of t
Good single-track trestles 25 ft high cost $2.50 to $4 per linear ft. Desirabl
of dumps are 1 200 to 1 400 ft; heights, 20 to 40 ft, preferably the latter; 60-ft
excessive, as the tracks settle badly. Straight dumps are best; curved tracks
to shift. In dry weather, a 14-man crew can shift 60-lb track; in wet weat
sticky dirt, up to 22 men are needed. The muskeg swamps of the Mesabi R
unsatisfactory locations for dumps; the dumps settle, and the muskeg bulges
side; but swamps may be employed with advantage in winter. MINING OR
of benches in ore are sloped 0.5 : 1. A 1 : 1 slope is allowed on faces carrying
tracks and switchbacks. Ore is mined in side cuts; benches, 12 to 25 ft hi
is generally loosened by blasting, with vertical jumper-drill holes, 8 to 10 ft
to 24 ft deep and 15 to 20 ft from edge of the bank. Holes are sprung, loaded
to 75 lb black powder, and fired in groups of 3 to 5. (For sizes of steam sho
costs of their work, see Sec 5, Art 10.) TRACKS AND LAYOUTS. 80-lb rail, we
and ballasted, is used on large scale work; such track cost $9 032 per mile in
which $2 000 was for laying, leveling, tamping, etc. Temporary tracks ar
and laid with less care. Grades of main tracks should be below 2% or at m

navoidable. Special tracks on 5% or 6% grades quick return of empties to the pit. 7.5° curves are lowable, but 50° curves are required and operated l show general layouts; these are variously modi- nnect benches in small pits or in deeper parts of) is ideal. It permits easy grades, curves, and turn- t is limited to large pits of regular outline. Com- s extremes of steam-shovel practice. It occupies erground before it was changed to open-pit. The d of overburden were stripped and dumped on the hed a height of 87 feet, or 201 ft above the deepest % grades were necessary to reach top of dump. A compensated 3% grade; in one season, much load- 65-ton locomotives were used on 3% grades; 80-ton 5, 336). SHAY VS ROD-TYPE LOCOMOTIVES. At the that the rod-type is best on good tracks and low ades over 3%, especially in starting a loaded train. n a 40-ft wheel base, as against about 9 ft for the connected Shay drivers are an advantage on sharp n start a train of 4 loaded cars (40 to 50 tons each)



Mountain Iron Mine, Mesabi Range

would need a running start. DRAINAGE. Many gal of water per day. As the topography usually ts, drifts are run under the pit to collect water and ence it is pumped to the surface. FIRST COST OF , cars, tracks, etc, is $190 000 to $435 000. Con- 80 000 to 125 000 cu yd per month, costs about ely with amounts and character of overburden 12¢ to 40¢ per cu yd; contract prices, on 500 000- d. Interest and amortisation charges on capital r variations in total mining costs. Total costs, s, but excluding royalties or interest on fee in- e. Outside of the Mahoning-Hull-Rust orebody, ral Mesabi district is below 40¢ per ton on board low 30¢ under favorable conditions. For further Work, Bucyrus Co; also Bib (35, 275, 336, 337). ee Art 101.

ev (307, 338, 339). The Copper Flat orebody is alcocite deposit in monzonite porphyry, with a shows mode of opening the pit, which has since each bench form a closed loop connecting with e layout for handling cars. Benches are about ovel and car tracks. Aver slopes of bench faces, cluding berms), 1.92 : 1 to 1.76 : 1. · On reaching a final slope of about 1: 1 (Art 101). Though and capping require blasting. Keystone drills 3 to 5 ft below bottom of bench. For a 50-ft

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bench, holes are 55 ft deep and about 37 ft apart; they are usually sta
of the edge of the bank, thus putting a 60-ft burden on the toe of hole
sprung with dynamite and charged with 500 to 2 000 lb black powder.
ore, gopher blasting is resorted to as an aid to churn drills. Cost of ben
was 38.67¢ per ft. Explosive us
per cu yd. In 1913, there were 7
with 5-cu yd dippers, and 1 70-to
dump, 18-cu yd cars handled t
ton steel, bottom-dump cars we
Twelve 65-ton, 6-wheel, saddle-t
were used in the pits and for ha
dumps; 5 85-ton locomotives h
mill, 25 miles away. Eight No
Keystone drills, for bench drilling
a locomotive crane. This equipm
264 000 tons of ore and 258 000
per month. In 1916, over 3½ m
were mined and nearly 4 million
removed. Total mining cost, 1
ore; taxes, 11.28¢; a charge of 3
stripping suspense account, mad
53.7¢ per ton. Cost of stripping
per cu yd.

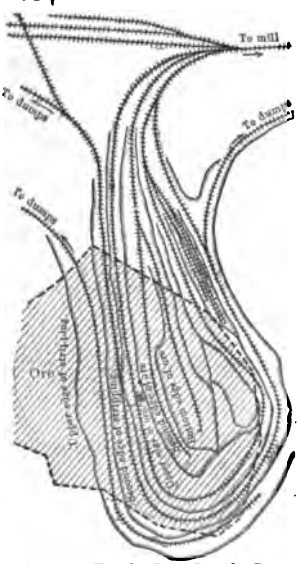


Fig 472. Eureka Pit, Nevada Consolidated Co

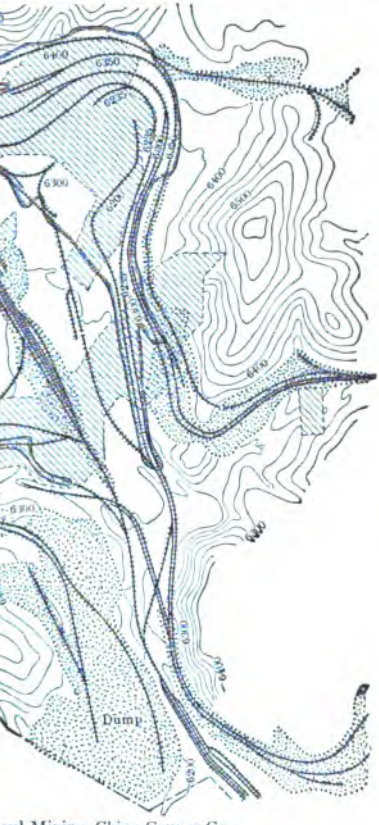
Chino Copper Co, Santa Ri
from D. F. MacDonald and C.
(340). Orebody is a disseminate
posit, in monzonite porphyry, w
capping from a few ft to 150 ft t
Bottom surface of the deposit is
thickness, 400 to 500 ft; aver t
The deposit was thoroughly ex
drilling (Sec 9). Fig 473 shows
the work. The benches are 30 t
up to 100 ft wide; shovels take
Though soft and broken, ore and
loosening by blasting; holes, ma
No 14 Cyclone drills (Sec 9) with 4 and 8-in bits, cost 38¢ to 40¢ per ft.
toe holes are drilled by 3¼ or 3⅝-in piston drills; cost, 32¢ to 47¢ per ft.
low-freezing dynamite (30, 40 and 60% Ngl) and Trojan powder No 2, are
and springing holes. Some large gopher blasts have been fired; see Bib (3
drilling and blasting. For a 3-month period in 1913, working at normal capacity, explosives cost 3.75¢ per cu yd of ore loaded; aver price, 9.55¢ per lb. The 40-ton shovel (Table 63) can load a car standing alongside in the cut; the long, high lift of the 100-ton shovel enables it to load cars standing on the bank of a through cut 22 ft deep. Shovels (excepting the 40-ton) handle 1 400 to 2 800 cu yd per 2 10-hr shifts; the w
due to occasional car shortage, local hard digging, and breakdowns. Sho
gineer, $190 per mo; 1 craneman, $135 per mo; 1 fireman, $3.50 per day;
6 to 8 pitmen, $2.25. Cost of loading ore is usually 6¢ to 12¢ per ton;
loading costs per shovel vary from 4¢ to 29¢ per ton. Equipment for t
and waste: 14 42.5-ton and 7 50-ton, 4-wheeled Porter locomotives, wit
and 16 by 24-in cylinders respectively; 45 12-yd, 40 6-yd, and 4 30-yd ca

Table 63. Steam Shovels at Chin

Number in use	Model	Max lift, ft	Allowable dist between cut and loading track, ft	Di cap cu
1	40-ton(a)	.....	.....	...
6	91-ton	19	10	3
2	92-ton	21.5	12	3
1	100-ton(a)	35	26 (b)	3

(a) For through cuts, in opening new bench
(c) Run of mine.

ctory, but the 6-yd are too light for large shovels, andle. Fine ore, that can go to the mill without eel R R cars furnished by the Santa Fé R R Co. ore. Wages of locomotive engineers and firemen, s of coal per shift @ $4.30 per ton; 2 locomotives kept with steam up as a spare. Total repair and



vel Mining, Chino Copper Co

r ton of ore. Tracks are standard gage, of 60-lb er 33-ft rail. Maintenance of pit tracks requires $4.55 per day, and 4 gangs of 13 men each, @ tracks, about 2.25%.

(153, 341). Orebody is trough-shaped, in mon- 1, 1917, nearly 40 million cu yd of capping and oved. The steam-shovel benches (Fig 474) are nps are on the hillsides, at ends of the switch- eight of benches, 40 to 75 ft or more. Faces in

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capping slope about 40°; those in ore, 50° to 55°; general slope of l
45°. Ore and capping, though fractured, require blasting, by toe hol
and churn-drill holes; the first two methods are preferred. After each
are trimmed by men who climb down on ropes. Most of the shovels
3.5-cu yd dippers; each requires 6
loaded into 50-ton steel R R car
side-dump cars are used for capp

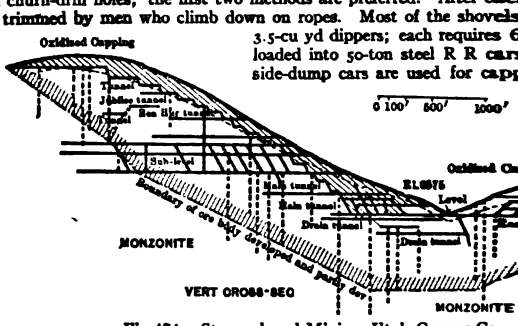


Fig 474. Steam-shovel Mining, Utah Copper Co

standard gage, of 65-lb rail; grades, 2 to 6%. Output in 1916 was abou
about 6 million cu yd of capping were removed. Direct mining cost
total aver cost, including fixed and general charges, 28.12¢ per ton, of
deferred charges for stripping (data from 1916 Ann Rep).

The steep topography makes necessary a large number of benches, a
trackage than would be needed for a flat orebody of same size. Bu
faces aid in maintaining a large output and allow early completion of st
topography gives ample dump room. As the surface of the ore is incli
some benches are partly in ore and partly in capping; in such cases
mine capping with ore, thereby lowering the grade, than to handle it se

Chile Exploration Co, Chuquicamata, Chile. Data from P. Yea
H. W. Moore (344), in 1916. Orebody is a massive deposit, in a zon

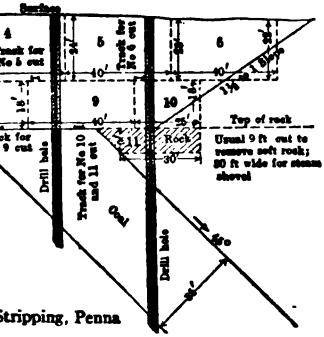
Table 64. Comparative Data on the "Porphyry" Copp

		Nev Consol	Chino	
Size of orebody	Length, ft	3 600	.....	. .
	Width, ft	1 200	500–1 000	. .
	Known area, acres	100†	.....	226
	Aver thickness, ft	200	107	524
Est'd unmined ore, million tons		52.84	94.49 (a)	369
Assay, % copper		1.41	1.655	1
Thickness of capping, ft		50–200	85	115
Volume of capping, million yds		46.68	65	42
Capping stripped to Dec 31/16, mil yds		21.66	16.89	39
Ore mined to Dec 31/16, mil tons		21.10	11.59	54
" " , 1914, " "		2.51 (e)	2.11	6
" " , 1915, " "		2.99	2.60	8
" " , 1916, " "		3.34	3.22	10
Capping removed, 1914, mil yds		3.04	3.17	5
" " , 1915, " "		2.76	3.13	5
" " , 1916, " "		3.99	3.56	5
Cost, cts per ton, st shovel mining, 1914		15.17 (c)	22.13 (d)	22
" " " " " , 1915		15.24 (c)	19.47 (d)	16
" " " " " , 1916		12.42 (c)	19.88 (d)	20

† Calculated. (a) Of this, 81 765 000 tons will be mined by steam s
acres completely stripped Jan 1/17. (c) Direct mining cost, excluding
velopment. (d) Total cost per cu yd of ore and stripping was: 36.52¢ (19
37.97¢ (1916). (e) Steam shovel; 123 300 tons by underground mining.
million tons "probable ore." (g) Rate for last 3 mo. * Total cost (1

le 64). Brochantite is the principal mineral in
res outcrop in many places; some areas have a
but yields many large blocks; there is a coarse-
blockholing in the pit. The topography permits
novels and ore trains can operate without inter-
gh, parallel to the long axis of the orebody; at
several 20-car trains. Ore is broken chiefly by
churn-drill holes; for details, see Bib (343, 344).
l ore for a 10 000-ton mill; 45-ton locomotives
pit tracks do not exceed 3%.

Penna. Data from J. B. Warriner, in 1917 (345).
been carried on for years; 1 to 2 million-cu yd
e, an area of 53 acres is being stripped to a max
.5 million cu yd of overburden, Bucyrus 70-ton
sed. Approx equipment for a one-shovel opera-
otives, 20 5-yd dump cars, 1 churn drill, 1 rock
th shop, rails, pipe lines, tools, etc; total cost,



Stripping, Penna

requires: 1 foreman, 1 shovelman, 1 craneman,
jackmen, 3 locomotive engineers, 1 dump boss,
2 drillers and 8 helpers, 1 fireman, 1 blacksmith
shovels and locomotives), 1 driver, 1 switchboy;
00; shovel engineer, $140 per mo; craneman,
. Wages have recently increased from 7 to 15%.
In clay or soft material, 30 ft is the proper max
lly 22 to 25 ft high, but in hard rock practice
llower holes break fewer large blocks and reduce
ost 10¢ to 14.5¢ per cu yd. Steam-driven drills
deep; churn drills for deeper holes. Holes are
black powder. Tracks are of 60-lb rail; curves
5° curves are used in forming dumps. Grades
umps. Locomotives weigh up to 120 tons, which
dump of any height. A 20-ton engine will push
% grade; 8 or 6-car trains on 3 and 4% grades,
least 1%; 4% grades are common, with some 7%.
nd slip in wet weather. Eastern, side-dump, 4.5
ump, 4-yd cars, are commonly used; some 8 to
hovel outputs vary with the season, averaging
d 10 000 to 12 000 cu yd in rock. Single-track
ed hoists, are often used for strippings too deep
city, 150 cars per day, or about the output of 1
foot of the plane, and from the top to the dump.
me are 50° or 60°. For larger operations, using

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2 or more shovels, double-track planes are better. Normal slopes of sid
1 : 1 in clayey earth, and 1.5 : 1 up to 2 : 1 in sandy ground. In rock,
cuts can be
of bank d
1 shovel cu
depths 0.5
lowed, or ev
is shaley.
stripping s
10 to 15 ft
footwall of
475); otherv
taminated
washed in
Costs of s
anthracite
20¢ per cu
and 35¢ to
"earth," as
cal strippin
cludes any e
coal, shale, s
stone, etc;
solid sands
more than

Table 65. Cost of Hoisting Planes, Anthracite Strippings

	Length of plane, 300 ft	Single track	Double track
First cost	Hoisting engine	$ 500 (*a*)	$5 000
	Track, rope, etc	700	1 100
	Grading	1 000	3 000
	Hoist house, piping, etc.	.....	800
	Total	$2 200	$9 900
Operating cost	Labor per day (*b*)	$17.88	$26.21
	Power	4.30	6.48
	Int and deprec, 15%	1.00	4.00
	Total	$23.18	$36.69
	Operating cost per car	$ 0.155 (*c*)	$ 0.122 (*d*)

(*a*) Second-hand hoist generally used. (*b*) Crew for single-track plane: 2 top-men, 2 bottom-men, 1 locomotive engineer, 1 hoist eng, 4 dumpmen and a boss. For double-track plane: 3 top-men, 3 bottom-men, 2 locomotive and 1 hoist eng, 7 men and a boss on the dump. (*c*) On basis of 150 cars per day. (*d*) On basis of 300 cars per day.

boulders of more than 1 cu yd. Some contracts are let at 20¢ to 25¢ per c
classifying the material. Coal is generally mined from stripped areas by ha
cars, or by steam shovels, the approx costs per ton being the same. Hand loa
but gives cleaner coal. Open-cut work yields 80 to 95% of the coal uncovere
and handling some coal goes to dumps. A 60-ton shovel uses 2 to 3 tons fuel p
locomotive, 1 ton; churn drills and forges, 0.5 ton each. Dump losses occu
previously worked by underground methods. For further detail, see Bib (345

98. Open-cut Mining with Drag-line Excavators

General. Drag-line excavators (Sec 3, Fig 4) for stripping
have booms 40 to 110 ft long, with 1 to 4-cu yd buckets. In soft
the excavator should retreat as the cut advances. Cars are lo
or from a hopper; with small cars, the hopper reduces delays and
large area reached from one position of the machine adapts it to st
shallow for steam-shovel work, especially on irregular bedrock.

Mayarí iron mines, Cuba (54). See Art 11 for ore occurrence. T
of orebodies, their softness, and freedom from overburden, favor use o
excavators. Drag-lines are used for ore 20 ft thick or less, when both sur
rock are irregular; steam shovels, tried first, could not clean bedrock wi
it up, and their small radius of action necessitated frequent moves. In
lines, with 2-cu yd Page buckets (Sec 3, Fig 3), handled 1 000 tons per
(346). The excavator works from the original surface, the bucket swing
of 60 ft and readily removes all ore to bedrock for a width of about 100
hummocks of bedrock, stumps, etc, are discarded. Ore is loaded into
dump, steel cars, on tracks on surface. Crew: 1 operator, 1 fireman and 3

Balkan mine, Menominee Range, Mich. Data from C. E. Lawrence i
Drag-lines were used for stripping and mining a hematite deposit overlai
quicksand, clay, hardpan, and boulder gravel. Before beginning work a st
the property was diverted, and a shaft sunk outside the deposit, with dri
for draining the overburden. Fig 476 shows the pit, about 1 150 ft long b
the spiral tracks (Art 97) have a grade of 2.6%; depth of stripping, 60 to 1
of banks are 2 : 1 in fine sand near surface, and 1 : 1 in underlying gravel an
could have been steeper in dry material. The stripping (1.2 million cu
tracted. 2 drag-lines had 85-ft booms, 4-cu yd buckets, and 24-ft turn
were mounted on hardwood rollers, running on 4-in plank; working wt, 15

hr, loaded through a hopper into 4-cu yd Western
ed 10-car trains 0.5 mile to a dump. Height of
o 30 ft by the flat angle of repose of the soft ma-

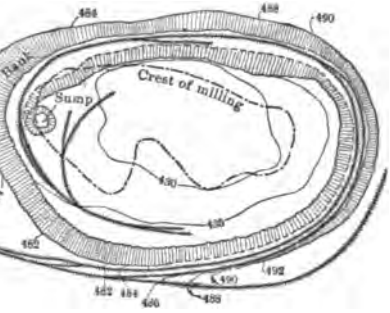

Fig 476. Balkan Mine Pit, Mich

the pit bottom; they worked successfully in the

been successful in the middle West; the work
els (Art 97, Fig 469); in Kansas, as much as 10
ncovered have been removed profitably.
g radius of 125 ft, has been used in the Penna
over $30 000. J. B. Warriner says, in 1917: In
bor represents about 70% of total cost. Large
ide instead of loading into cars, reduce labor cost
k indicates a saving of 10¢ per cu yd, allowing for
greater than for a 70-ton shovel requiring a full
ack, etc. Cost of moving these large machines
nd from a R R is high, but not prohibitive. The
region will probably increase (345).

Data from J. A. Barr, in 1914 (58). Residual
"brown rock" occur in 2 to 8-ft beds, lying on
ain by alluvium (307). See Art 11 for prospecting
are stripped by plow and scraper (Sec 3), at a
Most of the stripping is now done with drag-lines,
reas. Class 14 Bucyrus excavators, with a 70-ft
d to the removal of overburden averaging not over
n becomes too narrow. In Hickman Co, a Class
.5-cu yd bucket, handles overburden averaging 30
14 machine per shift: 1 runner, @ $150 per mo,
75; 1 ground man, $1.50; 1 foreman, $3; 1 team,
wear, $5; repairs, oil and supplies, $3; int, 6% on
, $5.20; total, $38.95. The above is for a machine
round man is needed with the roller-mounted ma-
d per 10 hr, rising to 1 300 or 1 400 cu yd under
es have about the same unit cost. Stripped phos-
novel and loaded into small cars. Attempts to use
posits, have generally failed commercially, because
for sorting out clay and flint while loading.
g is also done in this district (Art 99).

c Stripping and Mining

draulicking is cheap and rapid for stripping
dated deposits. It requires large volumes of

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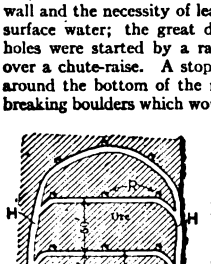
put up in them, or they may be quarried and loaded out by
often clog, causing costly delays. Grizzlies of timber or old rails
placed near their tops, to catch boulders and allow them to be
pare Fig 264, Art 59; see also Mesabi, below). The proper int
raises depends on their height, the shape and size of orebody,
on which broken ore will slide; raises should be spaced so as to
first cost against cost of removing the ore in the ridges betw
Raises 30 to 50 ft high give less trouble from clogging than highe

Limitations. Glory-holes may be adapted to masses or th
examples below). The ore should slide on moderate slopes
in the raises. Bad weather seriously hinders the work; wet
give trouble in raises; snow and ice make work on steep slopes d
ous. Surface water must be kept out by ditching. The work
in veins (Fig 477), unless the walls are strong; loose slabs are
before they become inaccessible, but their menace always lim
of the glory-holes. This danger and trouble with hung-up
chief DISADVANTAGES. The great ADVANTAGE is in providing cl
and loading, in connection with the cheap breaking common to o

Alaska-Treadwell mine (105). For ore occurrence and undergrour
Art 69. In the first 6 years of its life (1890 to 1896), the entire produ
glory-holes; the resulting excavation was 1 700 ft long, 450 ft deep, wit
of 420 ft. This work was then stopped, because of a large slide of rock
wall and the necessity of leaving a pillar of ore to protect projected lowe
surface water; the great depth made the danger from falling rock exc
holes were started by a raise from the intermediate level (Art 69), ge
over a chute-raise. A stope, 20 to 30 ft high and covering 3 chute-rais
around the bottom of the raise, for storage and to serve as a bull-dozi
breaking boulders which would block the chute-raises. The glory-hole fac
in a series of benches around the raise; so
large boulders were sledged in the pit. Pisto
made 12-ft holes. In small pits, where settir
from 150 to 200 tons were broken per 10 hr.
larger and the sides steeper, setting up was m
the output decreased to about 70 tons.



Mesabi Range, Minn (35, 183, 275, 351).
97. The MILLING SYSTEM is used here in dep
be profitably stripped, but are too small to war
ment for steam shovels, or are so situated or sha
shovels and locomotive haulage are impracti
is also used to recover ore from deep portions
pits, inaccessible by R R tracks on economical
OF WORK. A shaft is sunk outside of the oreb
50 to 100 ft apart; the 1st level is us
below the top of the ore. Fig 478 sh
layout of levels for deposits of la
elliptical main haulageway *H* facilita
age; crosscuts *C* are 40 to 50 ft apart
raises *R*, 4 by 4 ft to 5 by 5 ft, ar
surface every 40 to 50 ft, with chu
bottom of each. Another plan is t
level a rectangular system of drifts,
a main haulageway. In vein-like de
drift is driven with crosscuts to wall
intervals, from which raises are put

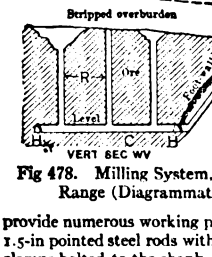


Fig 478. Milling System, Mesabi Range (Diagrammatic)

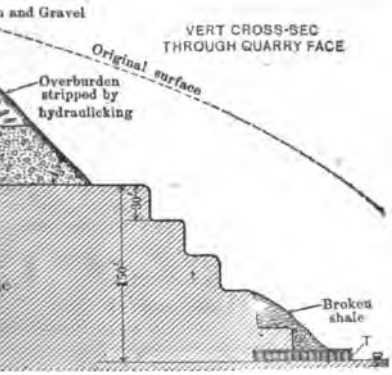
Several mill-holes are opened sim
provide numerous working places. Ground is broken by 5 to 20-ft holes, m
1.5-in pointed steel rods with sledges. A stuck bit is loosened by R R jack
clamps bolted to the shank. Holes are chambered and then charged with

ole, the sloping faces are attacked first at the raise; ceed up the slopes; when the dirt will no longer run abi ores break small, and run readily on 45° slopes; rred down after each blast, by men supported by ll-holes for men to grasp in case of a sudden slide of increases with size of the mill-holes. Raises often ill-holes. Clogs are lessened by breaking large nd by drawing the ore down frequently before it e, sub-drifts were driven above the haulage level, rtunity for barring down clogs; main-level drifts facilitate barring. At the Iroquois mine, a chain raised by a small hoist and pulled down by men raises are more apt to clog than shorter ones; at from 85 to 30 or 40 ft materially reduced delays. is difficult to design chute-gates that will prevent raise and destroying its taper. Production from enced largely by the weather. Heavy rain stops es of the mill, which must be filled by broken ore ains also block the raises with fine ore and water, ten rushes into the level and obstructs tramming. ill-holes approach the level they are tapped by n used at some mines to dig this ore and drop it

compared with steam-shovel mining: (*a*) a smaller tial investment, because a relatively small amount t production; (*c*) worked-out mills serve as dump- djacent areas; (*d*) the approach (Art 97), often inated. DISADVANTAGES: (*a*) high cost of hoist- of raises (note, however, that most steam-shovel anger of flooding the mills with sand and slime surface ditching.

to 20¢ more per ton than that from steam-shovel to 30¢ per ton less than for ore produced by top- h, up to 25 or 30 tons per man-shift or more.

D. T. Farnham in 1914 (352). This is a uring cheap loading costs in open cuts without



nnel Method, Renton, Wash (Diagrammatic)

ilt between sets. The tunnels are about 25 ft long, 8 sets erected outside. About 20 tunnels are driven

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20 ft apart along the face. The bank is then blasted down, the grizzlies be with 3 by 12-in plank and a 2 or 3-ft layer of shale; from 2 000 to 3 co down in 1 blast on a group of 3 tunnels. The planks are then removed fr 1 or 2 at a time, starting at the outermost set; the broken shale is bull-do and worked through into cars below. The ridges of shale between the tu by hand. A 3-man crew loaded 2.3-ton cars and trammed them 200 ft an 1.9 cars per man per hr in summer and 1.7 per hr in winter, or 4.37 tons p fore using tunnels, the duty of labor, in loading 2-ton cars by hand from on a 100-ft tram, was 2 tons per man-hr.

101. Miscellaneous Data, Open-cut Mining

Economic limit of depth. Open cutting loses its advantag where the stripping cost per ton of ore exposed exceeds the diff between open-cut and underground mining. For large, flat fairly uniform cover of moderate depth, the problem is simp per ton of ore for excess stripping to provide safe slopes aroun small that errors in its estimation are unimportant. But, on sm for deep overburden, the cost of stripping slopes is important deposits, where one or both walls must often be cut back to s amount of stripping is not a simple function of the depth, an cates calculations of the economic limit of open cutting. In c enough underground and surface mining have been done to e accurate unit costs, close estimates can be made from borings to determine the feasibility or limits of open cuts. The max de able stripping is often expressed in terms of a ratio of cu yd of ton of ore (or ft of overburden per ft of ore).

At Metcalf, Ariz, glory-hole work is profitable when the stripping ra ceed 4 tons of overburden per ton of ore exposed (152). In the anthra stripping ratios are from 2 cu yds of overburden per ton of coal to 4 : 1 Mesabi Range, in 1912, the economic limit was put as follows: (*a*) 1 cu y per ton of ore; (*b*) less than 2 ft of glacial drift to a 1-ft depth of ore (1 or taconite is put equal to 3 ft of drift); (*c*) a max depth of stripping any conditions (35). These ratios are useful if applied intelligently; are many chances for error; ratios applicable in a well-known district m in a new region.

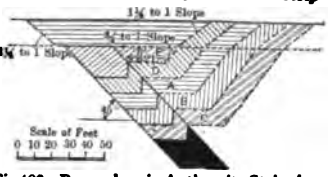
Mesabi estimates (35). Unit prices in 1912 for comparing under and open-pit (Art 97) mining costs were: Stripping glacial drift and per cu yd; stripping broken taconite, 75¢; stripping solid taconite, $1 pe shovel mining, ordinary conditions, 15¢ per ton; underground mining ditions, 75¢ per ton. Sometimes a choice of method can be made by ins estimate (Art 12). Usually the relative cost of underground and open-p be calculated by applying the above unit costs to a column 1 yd sq. drill hole shows 50 ft of drift and paint rock, 15 ft hard taconite, and 36 ore; general conditions allow use of either method. Comparison of costs

Underground mining. Cost of mining a volume of ore 1 yd sq by 36 @ 75¢ per ton (1 cu yd = 2 tons)	
Open-pit mining. Stripping a column of overburden 1 yd sq by 50 ft deep @ 30¢ per cu yd	
Stripping 15 ft of taconite = 5 cu yd @ $1 per yd	
Steam-shovel mining of 36 ft of ore = 24 tons @ 15¢ per ton	
Saving by open-pit work per sq yd of area	

Similar calculations are made for other drill holes, and all are combine to the tonnage represented by each. This preliminary estimate is supplem exact figures when necessary. Questions of adverse topography, dun presence of swamps, quicksand, etc, are local factors which do not permit

Pennsylvania anthracite stripping (145) **(see also Art 97) is gener** when it is cheaper than underground mining. But strippings yield hi

of prepared sizes (Sec 34), and allow close adjust-
tors are often decisive in making a choice. Strip-
other-
mining
f strip-
ed on a
moved
conom-
ng the
e areas
is the



reas *B* Fig 480. Pennsylvania Anthracite Stripping
respec-

loss, even though a considerable profit comes from
operation, marked advantages must be gained by
g *B* and *C*. Fig 481 shows a crop stripping common
uthern anthracite fields; either the cover must be
, or a thick chain pillar of coal left, as shown. The
w the pillar is minable at as low a UNIT COST for cut-
loading as could be realized by mining the entire
o ft of vein from gangway *W*, after stripping the
len. By the latter method, DEVELOPMENT COSTS
e less by a few cents per ton. But, comparing the
ith the case in Fig 480, the ratio here is 4 cu yd
per ton of coal in the chain pillar, or a ratio of 1 : 1
the total coal above gangway *W*. Hence, the coal
lassed as stripping coal is mined at a loss (345).

Copper Co, N M. Estimates of economic strip-
ts in porphyry copper ores sometimes involve other
, due to fluctuating prices of copper and because
opper may be carried by part of the overburden,
nce it must be excavated, may sometimes be milled
fit. Following formula, used at the Chino mine,
s the mode of determining the lowest grade of rock
y be classified as ore (340).

$A(R+E+c) = AB\frac{100}{105}$, where

	Present value at Chino mine
	$0.4683
dry ton	0.1111
	0.5776
mill to smelter	1.1191
ntrates	6.3520
per lb copper paid for	0.0106
er lb copper paid for	0.0014
	$B + 100$
sis	13¢
	65%
	25%
be calculated)	?

above equation:

$$\frac{M + F_0 + m}{\frac{B}{} - \frac{(f_c + S)}{1900\,D} - (R + E + c)}$$

mined until the copper is sold; factor $100\,B \div 105$
nspositions, the equation can be solved for any
Bib (340).

ovel mining. Following data, from E. E.
experience at the Nevada Consol (Art 97).

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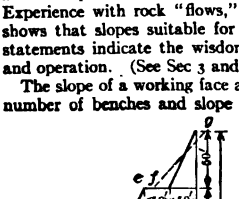
The slope at which a bank will stand depends on the depth to be e as character of the material; hence no general rule can be formulated. than 1 : 1 will not stand in the leached porphyry and fractured schis nated copper deposits. Blasting shatters harder rocks, so that bank much steeper than 0.66 horiz : 1 vert. Faults if present may necessit Experience with rock "flows," in deep excavations in soft rock on th shows that slopes suitable for ordinary work may not stand on great statements indicate the wisdom of using generous slope allowances fo and operation. (See Sec 3 and 5 for data on angles of repose and safe s

The slope of a working face as a whole is affected by the height and number of benches and slope of each face (Fig 482). For benches o


g
e
f
50'
50'
50'

limited to small depths. (*e*) The plant is often nits. (*f*) Work is stopped or seriously hindered cts snow and rain, and tends to drain water from ld a serious expense for pumping to the cost of be partially met by surface ditching; note also erley diamond mines, Art 89. (*h*) Even where from working faces, men at the base of the faces

ING METHODS

pt, Colonial Colliery Co, Natalie, Penna

s deal with the methods common to nearly all are more varied than those for coal, due to the e of metalliferous deposits, and to avoid repeti- ted chiefly to the distinctive methods employed, ms are "beds," they are generally mined like Striking exceptions exist in the pitching basins here the extreme folding and varied contours of ted methods. For details of prospecting, devel- avations, and underground handling of minerals, For other data relating especially to coal mining 21, 23, 34, and 35.

ions, and General Considerations

een the methods of mining anthracite and bitu- oper method depends far less on the kind of coal he seam, roof, and floor. For example, similar of the northern anthracite field of Penna and in likewise, some of the "pitch-mining" methods hing bituminous seams of Colo. Standard un- ions, fall into two groups:

nown as room-and-pillar, pillar-and-breast, those in which, from haulageways (gang- mparatively wide openings (breasts, rooms, . These openings correspond to the stopes een rooms and other openings, support the S, both anthracite and bituminous, is mined average recovery in FIRST MINING, that is; s 30% to 50%. Final ROBBING commonly r more, depending on local conditions.

divided into ADVANCING and RETREATING , but, until recently, have found little favor s, with intervening pillars, the coal is mined long faces. As the work advances or re- he haulage and airways being kept open by very approximates 100%.

to levels and drifts in metal mines, are called s mines, U S), or GANGWAYS (anthracite mines, e called AIRWAYS or AIR COURSES, also MONKEYS entry, whether by tunnels, drifts, or vertical or y on the conditions governing the development hy, and the pitch and physical characteristics of ations in development necessary to provide the , see Art 104, 108.

imes adopted for coal seams, or parts of seams,

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102. Choice of Method

The best method in any case is that which will yield a maximum r minimum cost per ton, in the best marketable condition, and with to the miner.

General factors influencing choice and details of method are as

Roof pressure is an indeterminate quantity depending on the depth of longwall mining, heavy roof pressure is advantageous, as it tends to break after it has been undercut, thus facilitating mining at the face. In pillar-min an increase in roof pressure requires larger pillars and smaller openings; thi true in thick seams, or in soft, friable coal. When robbing, the removal of may so shift the roof weight that the adjacent pillars will crush, causing a ger with loss of coal. Foreign practice favors longwall methods for all deep, mo seams, where other factors are at all favorable. American practice employs under all conditions of roof pressure, in both flat and pitching seams; the pil times relieved by artificial supports (Art 110).

Strength and character of roof and floor. The strength of a roof is support itself over mined-out areas. A strong roof allows large opening timbering requirements. A very strong roof transmits the weight to t advantage disappears when, with increasing area of workings, the pressur pillars and causes a squeeze. A roof that will break down and "choke" room is desirable, as it eases the weight on adjacent pillars. In pillar metho facilitated by a roof that is self-supporting over moderate areas, but whi "square" near the working face; this eases the pressure on remaining parts lessens the chance of a squeeze. A hard, brittle roof, that tends to flake of and difficult to manage, requiring narrow openings and much timber. "hangs" over moderate areas and settles gradually is advantageous for lon

The character of the floor also influences the size of openings. A soft bo requires narrow openings and large pillars, especially where firm coal is n strong roof. The pressure on the pillars tends to force them into the floor, a floor to bulge; this is known as a "heave" or "heaving bottom." A firm sirable for all methods of mining.

Character of coal. Strong coal may be mined by either pillar or long The mining of soft, friable coal is always difficult, and in pillar methods pillars. Flat seams of friable coal under heavy cover are best mined by lo character of roof permits.

Inclination and thickness of seam. In general, the maximum allo opening decreases with increasing inclination ("pitch") and thickness Friable coal in a pitching seam sometimes tends to run; this emphasizes the of large pillars and small openings. Practice abroad favors longwall for pi of moderate thickness; in the U S, practically all pitch-mining is done by pi

Presence of explosive gas and dust indirectly influences choice of me a direct effect on the details of all methods. Advancing longwall in a gassy s constant danger of gob fires; hence, where conditions permit, retreating long able. In pitching seams containing occluded gas, the coal often tends to bre the face and pillars (called "bumpy" coal), and requires narrow openings and Where there is danger from explosions of gas or more especially of dust, it is b a divisional or panel system (Fig 505), to confine explosions to small areas. also makes low-velocity air currents desirable and influences the cross-sectio

Product desired. In most cases, the best method of mining is that which the largest amount of lump coal; coal for coking (Sec 35) is an exception. produce the most lump coal; from this point of view under favorable condit methods are best. Excessive use of explosives should be avoided, and syste and haulage carefully designed to avoid breakage.

Character of labor and market. In longwall mining, there must be a s for the product and a steady class of labor; even a short stoppage may re damage to the working face. In pillar methods, this is less important.

194. Room-and-pillar Methods

al. Because of the frequent presence of explosive dust and gas (Sec 14), ventilation is more important for coal than for metal mines. This driving DEVELOPMENT OPENINGS IN SETS. Fresh air enters by one or ings, the INTAKE, and after passing through the workings the foul air mine by other openings, the RETURN.

pment in flat or slightly pitching seams. SINGLE-ENTRY (rarely Sec 14, Art 4). A single opening is driven, from which rooms are turned oe both directions (Fig 484). acts both as a main haulage- an intake airway. The venti- rent circulates as shown. A f may cut off the flow of air. hows a typical DOUBLE-ENTRY Main entries are driven from or from the surface; from these, itt entries are opened. Rooms d off the cross entries. The the split system of exhaust n (Sec 14), the main haulageways being intakes. Advantages: (*a*) in cident in one entry, the other affords an escape; (*b*) the mine is divided ately ventilated sections, so that a fall of roof or complete closure of f butt entries will not affect ventilation in other parts of the mine; (*c*)

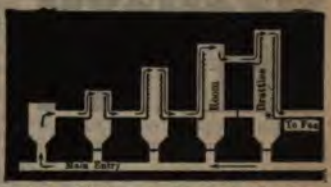


Fig 484. Single-entry System in Flat Seam

Double-entry System in Flat Seam (U S)

main or cross entries may be driven ahead as far as desired and ventilation maintained without turning off rooms. Fig 486 shows TRIPLE-ENTRY applied to main entries, with double-entry in cross entries. The middle entry is the main intake and haulageway; the outer ones form the return. While involving more narrow work than double-entry, this system is well adapted to gassy mines of large working area. It is also used where local conditions (bad roof, etc) prohibit driving a single entry of sufficient width for double-track haulage; the middle entry is then used as the return and the outer entries as intakes and haulageways. In QUADRUPLE-ENTRY, 4 main entries are driven in parallel (Fig 487), of the mine being served by a separate intake and return. One intake y be used as a traveling way, and the other as a haulageway, or both ways. This system facilitates *circulation of large volumes* of air and pted to high-*speed endless-rope haulage (Sec 11)*. By leaving the center *ct. i e, eliminating the* BREAK-THROUGHS *B*, the two sides of the mine

become wholly independent of each other, and may be considered mines, each opened by double-entry. QUINTUPLE-ENTRY is the s preceding, with addition of one more entry, used as a traveling wa

Fig 486. Triple-entry System (Coal Miners Pocket-book)

Entries in flat or slight seams. SIZE of entry depe factors: (*a*) Entries must be l to insure adequate ventilation. prescribe a minimum quantity per man or animal, and a max current (Sec 14). Airways sho minimum perimeter for a giv square entry is preferable to a one. In haulageways, both width are influenced by size method of haulage. (*b*) Cost ing wide openings may be prohi is bad, or the floor has a tender or the coal is friable or easily crushed; in such cases, narrow entries, to drive, are in the end an economy. In seams containing more o or in thin seams, where floor or roof must be blasted to secure headr are usually driven 16 to 20 ft wide and then narrowed to 6 to 10 ft the waste into a packwall along one of the RIBS (Fig 488). (The rib is the face of solid coal along the side of a working.) The extra cost of driving is partly or wholly paid for by the coal taken out and transportation of waste is eliminated. DISTANCE BETWEEN ENTRIES. Main entries in flat seams are commonly driven on 30 to 60-ft centers, leaving, for a 10-ft entry, a 20 to 50-ft pillar. Large pillars are required to protect the main entries during life of mine. Cross entries need not remain open after the area served is exhausted; they are on about 30-ft centers, leaving a 20-ft pillar for a 10-ft entry.

Fig 487. Quadruple-ent

For economy in driving, entries should be as close together as possil the cost of breakthroughs. Distance between sets of cross entries length of room and method of mining; length of room depends on roof, floor, and coal, system of haulage and ventilation. Where room

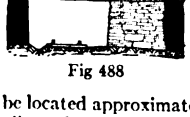
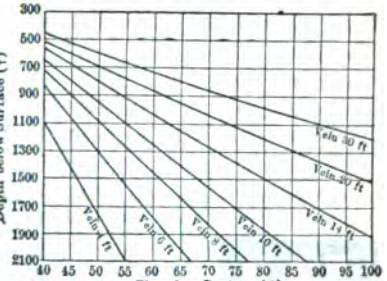


Fig 488

in one direction only, their length is comm 350 ft; where they are turned in both direc length is 600 to 700 ft. DIRECTION AND LOCA largely on the size and shape of the prope conditions, cleavage or CLEAT of the coal (in mines), dip, amount of water, etc. Main en be located approximately to bisect the property. In mines opened b allows development to proceed in 3 directions, reduces the length of and permits final robbing at an earlier date, decreases cost of protrac nance of long haulageways, facilitates haulage, and permits a materia output on relatively short notice. To favor haulage and drainage in sli ing seams, main entries are driven directly on the pitch and cross en parallel to the strike. In bituminous coal, the cleat may alter this plan of driving is often increased if entries are not parallel to one of the cleavage planes. ALINEMENT AND GRADE should be as uniform

e and wear and tear on track and rolling stock, nd hence reduce efficiency of ventilation. Uni- haulage and drainage. Cross-entry haulage- 75 to 1.5%) favoring the load. To obtain this or bottom rock and to fill local depressions, seams may be driven at an angle to the strike following formula (355) is useful: $\sin A = \tan X$ n cross entry and strike of seam, or angle be- room is to be driven on a grade; X = pitch of of seam in degrees (see Table 70). In turning curves should be avoided. This is sometimes CUT-OFFS.

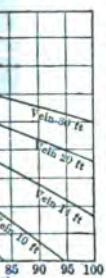
slightly pitching seams. Rooms are usually entry in one or both directions, depending on



ig 489. Relation Betw'n Depths and Room Centers for Different Thicknesses of Vein for 20-ft Rooms

th below surface, ft; z = distance center to

pillar, ft; h = total thickness of seam, ft. It overlying strata is 144 lb per cu ft and that the safe load on a cube of anthracite is 1 000 lb per sq in. Fig 489, 490, plotted from this formula, give results corresponding with practice in Penna.



Room Centers
24-ft Rooms

Bunting states that, in the application of any formula to the calculation of the size of pillars necessary to resist the pressure of the overlying strata, consideration must be given to the nature of the seam and its contiguous strata, and to the dip. Moreover, the proper factor of safety varies with local conditions, such as the relative location and extent of workings, and the seriousness of possible disturbance to the overlying strata and surface.

nings, DOUBLE ROOMS may be used; they have k on each rib, gob being stored in the middle. Fig rooms. Haulage in rooms, unless mechanical, ha-

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Table 68. Dimensions of Rooms and Pillars in Flat Seams

Ex No	Location	Rooms		Room necks	
		Width, ft	Length, ft	Width, ft	Length ft
1	Pittsburg Seam	21	.....	.....	.....
2	" "	24	.....	.....	.....
3	Southern Colorado	20	200	8	30
4	Dawson, N M	24	350	6	20
5	Oklahoma	20–25	200–275	8–10	.....
6	Hisylvania, Hocking Valley, O.	30	250	.....	21
7	Saline County Coal Co, Ill	20	300–350	6	.....
8	Michigan	30	150	.....	.....
9	Primero, Colo	18	300	8	.....
10	Loup Creek, W Va	25	400	.....	.....
11	Whitwell, Tenn	42	200	6	21
12	Franklin, Ill	21	.....	10	.....
13	Steubenville, O	22	250	.....	.....
14	J. K. Dering Coal Co, Ind	22	150	.....	.....
15	Castle Valley, Utah	22	400	12	20
16	Stearns County, Ky	40	.....	.....	.....
17	" " "	24	.....	.....	.....
18	Tennessee	30	215	10	21
19	Melcher, Ia	60	.....	10	.....
20	Kaylor, Pa	42	.....	12	21
21	Gary, W Va	20	.....	12	.....

comes difficult on pitches of 6° to 8°; rooms may then be turned off obliquel this means of reducing haulage grades in rooms is feasible in seams pitching culations for grade are made as for oblique entries This practice increases the difficulty of robbing pill

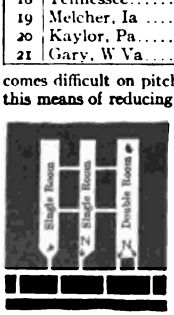

Fig 491

Influence of cleat. Bituminous coals usu two cleavage planes or cleats, running approx at to each other and to the dip. Advantage is taken of them to facilitate breaking down the coal and to release occluded gas by driving the rooms at different angles to the more pronounced cleat. FACE CLEATS are longer and more regular than END CLEATS. Fig 493 shows methods of driving, with their names. In driving FACE-ON, the face of the room is parallel to the face cleats. This is the gene where conditions permit; it requires less powder and undercutting

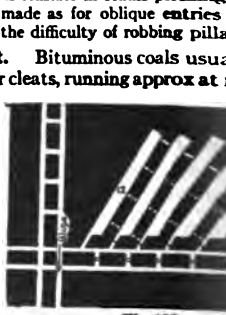
Fig 492

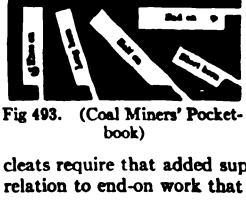
Fig 493. (Coal Miners' Pocketbook)

others, and yields more lump coal. In L work, the room face makes an angle le with the face cleats. Coal breaks in and with well-placed shots there is a fa lump coal. In HALF-ON work, the roor 45° to the cleats; adapted to coals break well on face and end cleats. In SHORT-E the room face is between 45° and 90° cleats; adapted to cases where prono cleats require that added support be given to the coal face and bear relation to end-on work that longhorn does to face-on. In END-ON

sions of Rooms and Pillars (356)

Rooms		Room pillars, width, ft	Cross-cuts, c to c, ft	Thickness of coal, ft	Pitch of bed, deg
ngth, ft	Width, ft				
300	25	10–20	50	4–6	2–40
–300	20	10–20	20–50	2–4	2–20
300	25	25	50	8	0–15
250	25	25	50	5–6	0–10
250	25	25	50	10	2–4
–250	18–30	12	40	3–9	4–12
–200	30	15–20	60	5	0
200	40	20	60	5–6	0
–300	18–30	10–12	40	3.5–6	8–40
–210	21–30	9–15	60	3.5–7.5	0
–300	.....	40	105	12–14	0–10
250	12	32	105	6–9	5–12
250	12	72	105	6–9	5–12
250	21–24	12–18	75–100	4.5–7	nearly flat
–200	21	15	75	2.7–6.3	0–3
150	25	18–20	65	7–11	0–5

cter of	Roof faults or slips	Character of bottom	Depth of cover, ft	Time of drawing pillars
	(*b*)	FC	100–600	R
	N	H	50–300	R
	N	H, FC	50–400	(*c*)
(*d*)	N	H, FC	50–600	(*c*)
(*e*)	N	H, FC	50–200	(*c*)
(*e*)	ST	H, FC	50–400	ND
(*h*)	(*i*)	H, FC	200–400	ND
(*h*)	N	H, FC	400–500	ND
	ST	H, FC	50–700	ND
, Ls	(*l*)	FC	200–300	(*c*)
(*d*)	(*m*)	FC or Sh	50–300	(*c*)
	N	FC or Sl	90–350	R
	N	FC or Sl	350†	R
Sl	N	FC + Ls	50–350	R
	N	FC	20–300	R
	N	FC	50–750	R

oss lings c, ft	Entries width, ft	Pillar width, ft (*o*)
66	8	10
00	8	10
00	8	12
50	8	12
00	8	12
–300	8	18
–400	12–20	40
00	20	40
–300	8	18
–400	8	(*p*)
000	8	.....
00	8–9	52
00	9	52
–500	8	30–40
....	8	15
60	10	20–30

B = Soft. A = Hard. N = None. F = Friable. ST = Sometimes. ND = Not drawn. OF = Face-on. WG = With grade. NC = No cleats. UP = Up pitch. Sl = Slate. Ss = Sandstone. Sh = Shale. Ls = Limestone. FC = Fireclay. H = Hard. † Or over. R = Retreating. (*b*) Draw slate 2 to 10 in. (*c*) As soon as room is driven. (*d*) Soft. (*e*) Sandy. (*f*) Some cleat. (*g*) End-on. (*h*) Blocky. (*i*) Horse backs. (*j*) Parallel to main cleat. (*k*) Across cleats. (*l*) Local slips. (*m*) Small slips. (*n*) Heavy draw slate. (*o*) Betw entry & air course. (*p*) Main, 25–50 ft; cross, 12–25 ft.

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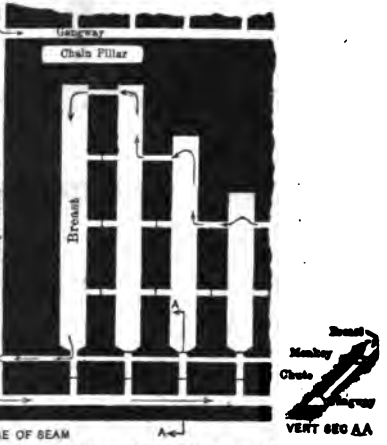
Table 70. Distance from Center to Center of Rooms or Brea Measured on Entry or Gangway

Angle between room and entry, deg	Width of room + thickness of pillar, ft									
	20	25	30	35	40	45	50	55	60	65
	Distance measured on entry, ft									
90	20.0	25.0	30.0	35.0	40.0	45.0	50.0	55.0	60.0	65.0
85	20.0	25.1	30.1	35.1	40.2	45.2	50.2	55.2	60.2	65.3
80	20.3	25.4	30.5	35.5	40.6	45.7	50.8	55.8	60.9	66.0
75	20.7	25.9	31.1	36.2	41.4	46.6	51.8	56.9	62.1	67.3
70	21.3	26.6	31.9	37.2	42.6	47.9	53.2	58.5	63.9	69.2
65	22.1	27.6	33.1	38.6	44.1	49.6	55.2	60.7	66.2	71.7
60	23.1	28.9	34.6	40.4	46.2	52.0	57.7	63.5	69.3	75 1
55	24.4	30.5	36.6	42.7	48.8	54.9	61.0	67.1	73.3	79 4
50	26.1	32.6	39.2	45.7	52.2	58.7	65.3	71.8	78.3	84.9
45	28.3	35.4	42.4	49.5	56.6	63.6	70.7	77.8	84.9	91.9
40	31.1	38.9	46.7	54.5	62.2	70.0	77.8	85.6	93.3	101.1
35	34.9	43.6	52.3	61.0	69.7	78.5	87.2	95.9	104.6	113.4
30	40.0	50.0	60.0	70.0	80.0	90.0	100.0	110.0	120.0	130.0
25	47.3	59.2	71.0	82.8	94.6	106.5	118.3	130.1	142.0	153 8
20	58.5	73.1	87.7	102.3	117.0	131.6	146.2	160.8	175.5	190.1
15	77.3	96.6	115.9	135.2	154.5	173.9	193.2	212.5	231.9	251.2
10	115.2	144.0	172.8	201.6	230.4	259.2	287.9	316.7	345.5	374.3
5	229.5	286.9	344.2	401.6	459.0	516.3	573.7	631.1	688.4	745 8

room face is parallel to the end cleats; this, with short-horn, is adap stand strong roof pressures, but yields less lump coal than other method containing occluded gas at high pressure, face-on methods may cause bursts of gas, whereas end-on rooms cut the face cleats and allow g gradually. Under these conditions, by using long-horn or short-horn r pressure of gas may be controlled and utilized to assist in breaking dc

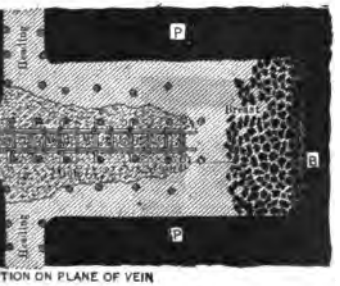
Development in pitching seams. The following notes desc anthracite practice, which may be considered standard for the U S. mining, development openings are driven in pairs, consisting of a hau GANGWAY, and an airway or MONKEY. These openings are connected corresponding to the breakthroughs of flat mining. In thick, gassy extra monkey may be added, to provide a separate split of air in ea place. Fig 494 shows typical development for moderately thick variations, see Art 106. SIZE OF GANGWAYS depends more on size method of haulage than on character of coal, roof, and floor, and t seam. Gangways for mule haulage are commonly 7 ft high by 10 tc in the clear. For mechanical haulage the height may be less, th practice leaves at least 2 ft clearance above top of car. DIMENSIO KEYS depend primarily on air requirements (Sec 14); thus, if an area is required to pass the air needed in a split without undue friction, a n 6-ft seam would be 6 ft high. DISTANCE BETWEEN DEVELOPMENT In Penna, the distance between gangway and monkey in moderately t is commonly 30 ft. In thick seams, this distance depends largely c the monkey and gangway are on the same or opposite walls of the is customary to space gangways 300 ft apart along the pitch, leavir PILLAR 20 to 50 ft wide (along the pitch) underneath each gangway This allows a breast length, including the chute, of 250 to 280 ft. In mining, on pitches over 40°, it may be advantageous to divide a lift c

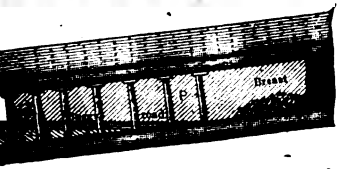
ey gangway. In many cases small mine cars, ngways; if so, the coal is lowered to the main



nt of a Pitching Anthracite Seam, Penna

(Art 106). Standard cars on counter gang- anes (Sec 11), or the coal handled in counter





Chance, Vol AC, 2nd Geol Surv Penna)

ngways are increased safety and better facili- mber. DIRECTION, LOCATION, AND GRADE

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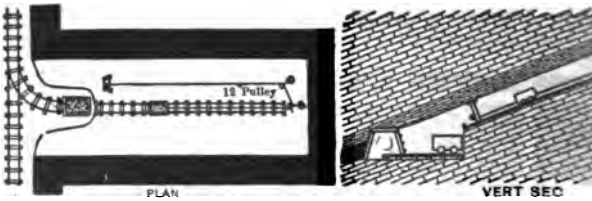

Fig 496. Buggy Breast (Coal Miners' Pocketbook) Pitch exagger

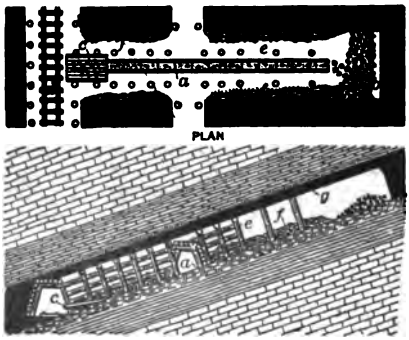

Fig 497. Chute Breast (Coal Miners' Pocketbook). a = sheet-iron chute. c = loading platform. e = props. f = gob. g = top coal

GANGWAYS. Ga
driven in the sea
its contours alon
and are on a grad
1.5% in favor of
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directions from th
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rooms in flat mini
ence shows that
thick seams may
most economicall
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and floor, and
thickness of coal,
same as in flat sea
coal in a pitching

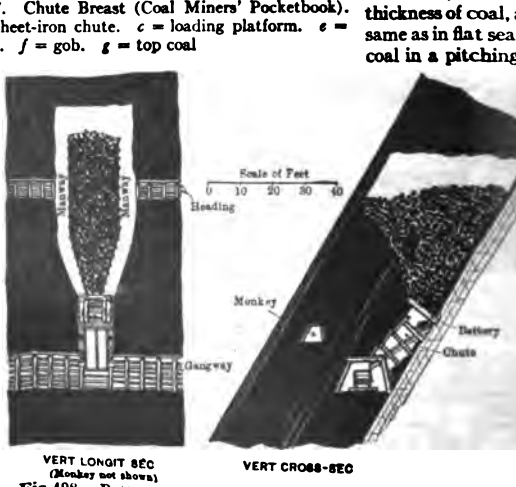

Fig 498. Battery Breast, Mammoth Seam, Hazelton (after Cha

to break away from the face and ribs, causing a "run-away" with
disastrous results: in such coal, narrow, well-timbered breasts are

cording to the degree of dip: WAGON BREASTS g, and are used where the pitch is less than 12°. e face. BUGGY BREASTS dip 10° to 18°, buggies ort coal from the face to the gangway (Fig 495

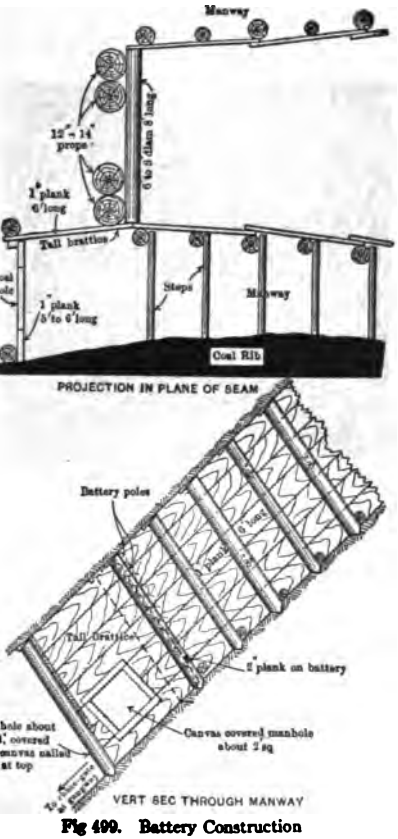

Fig 499. Battery Construction

d; manways are carried up each side and the ddle by lagged or planked props. The broken ipies approx 50% more volume than solid coal. excess broken coal is thrown down one of the the gangway. When the breast is finished the battery as desired. Fig 499, by W. L. Cross, Jr, Minersville, Penna, shows typical construction.

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165. Examples of Room-and-pillar Mining in Flat Sea

(See also Tables 68, 69)

George's Creek district, Md. Data from H. V. Hesse in 1909 (357). F unsystematic methods employed in 1850 for mining a bituminous seam 6 t All workings were driven at random, with no attempt to recover the pillars. total coal, not including the top coal, which was left standing for a roof, makes reworking or r cult and costly. This dicates the necessity f mining to secure max ery. In this seam, n and wide pillars are the character of roof a 501 shows one of the re atic plans of working, numerous experiment the coal is obtained (Art 107); a total reco is claimed.

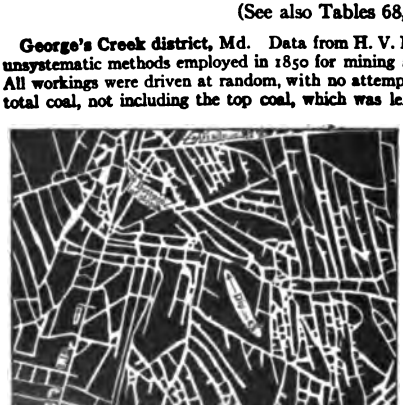

Fig 500. Unsystematic Mining, George's Cr, Md

Pittsburg region. F. Z. Schellenberg in Seams are regular, per tematic development; cheat in the Pittsbur important factor in workings; rooms are us face-on. Panel systen eral use. Common d openings: butt entries rooms, 21 ft wide by 250 ft long; neck, 9 ft wide by 21 ft long; pillars, 15 to Thickness of seams, 6 to 9 ft; dips, 0° to 6°. Fig 502 shows the method used b Coal Co; Fig 503, 504, the general method of the Monongahela River Cons Coke Co. Panels are 500 to 600 ft wide by 1 400 ft long; rooms are usuall both directions from the butt entries. Pillars are robbed during either advance or retreat; the faces of robbing operations make an angle of 45° with the butt entries. Mining in the Connellsville region resembles the above, except that, as the coal is firmer, pillars between entries are smaller.

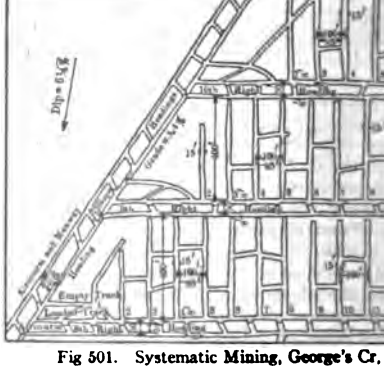

Fig 501. Systematic Mining, George's Cr,

West Virginia. Due to the regularity of the seams, definite systems of mining are planned and carried out with only slight variations to meet local conditions. Main entries are triple or quadruple; section entries, triple or double; cross entries, double. Panel systems are common. Usual dimensions are as follows, with noted below. Main and section entries are 40 to 60-ft centers and 8 to 12 ft entries, 30 to 50-ft centers, 8 to 12 ft wide; interval between pairs of cross ent 600 ft; rooms, on 40 to 80-ft centers, are 15 to 30 ft wide and 200 to 400 ft lon

ft, in rooms, 100 ft; barrier pillars, 100 to 200 ft wide. sburg seam, is opened by drifts. Roof is poor, re-; floor is an 18-in layer of fire clay on shale; coal is system is used. Section entries run face-on at 1 200

Fig 502. Systematic Mining, Pittsburg Seam

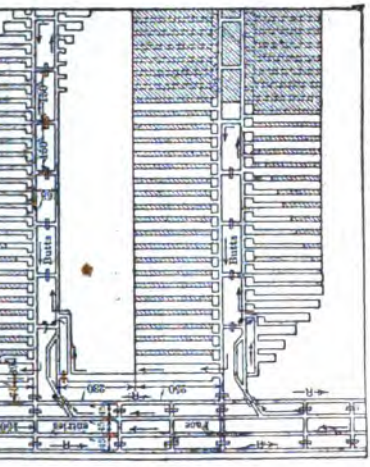

Fig 503. Systematic Mining, Pittsburg Region

at right angles on 348-ft centers; all entries are 9 ft ft wide and on 40-ft centers; necks, 9 ft wide. Pillars t 107) (359). At the FULLER MINE in same district a e following: double-entry system; entries, 11 ft wide entries turned at 352-ft intervals; rooms, 22 ft wide

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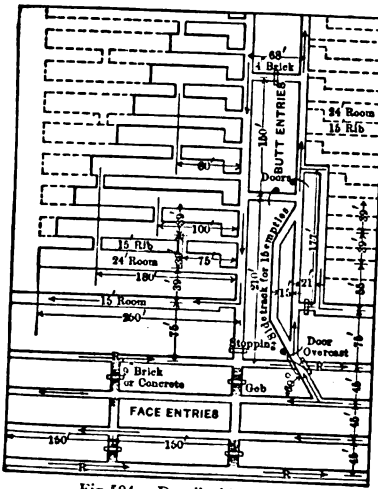

Fig 504. Detail of Fig 503

on 35-ft centers; necks, 25 ft long. A 30-ft chai between the room faces entry. Pillars are ro advance; the roof brea along joint planes (359). FIELD. Seams, 3 to 10 f 505 shows the general plan of the Pocahontas C Co, providing for quadru tries, triple cross entries panel entries; alternative shown in panels 1, 2, 3. directions, issued by the dicate the distinctions be methods. PANEL NO 1. rooms on the 3rd cross e as they are reached. Be (Art 107) when the secc completed, and rob advan 2nd and 3rd cross entries t ft of the 2nd cross entry. cross entry drive the last and rob retreating, taking rier pillar left on the 2nd PANEL NO 2. Drive entries before turning rooms, excep

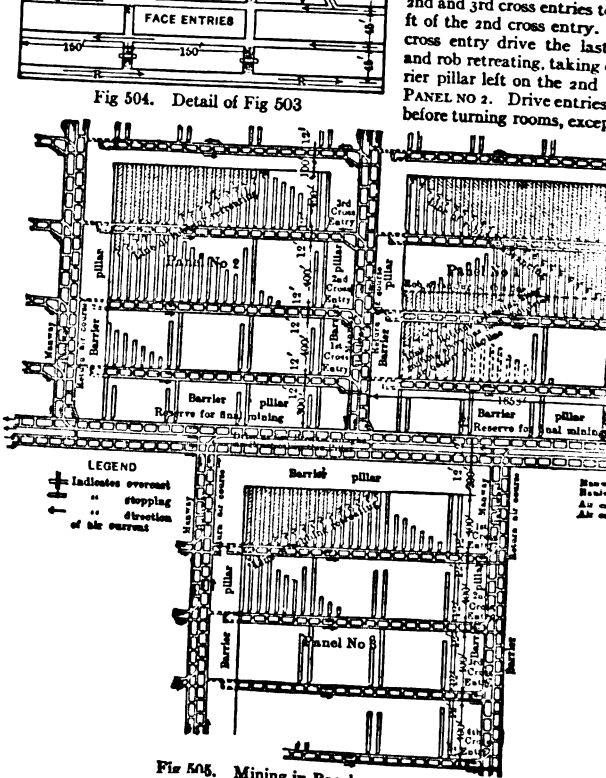

Fig 505. Mining in Pocahontas Field, W Va

nois Room-and-pillar Practice

Entry width, ft			Entry pillar width, ft		
Main	Cross	Room	Main	Cross	Room
6†	.....	.....	.....	.....	.....
21°	21°	21°	35	35	.....
7	6†	.....	16	12†	.....
8	8	.....	12†	12†	.....
12	12	.....	42	15	.....
12	12	12	60°	60°	.....
8	8	8†	25	20	8
8	8	8†	35	35	30
12	12	.....	40	40	.....
14	14	14	50	50	.....
9	9	.....	25	21	.....
7	7	.....	35	30	.....
21°	21°	21°	40	40	.....
9	9	.....	20	20	.....
14	21	.....	42	42	.....
8	10	.....	30	34	.....
12	12	12	50	50	.....

	Width of room-pillar, ft	Room necks		Dist from entry to full room width, ft	Distance between rooms, centers
		Width, ft	Length, ft		
	15	6†	15	25	30
	43°	21	12	24	70°
	6	9	9	.....	30
	18	8	10	20	42
o	19	8	7	12†	45
	30	12	12	30	60
	10	8	10	15	34
	9	8	9	27	35
	30	12	20	50°	60
	25	22°	22	28	55
	4	9	19	.....	47
	3†	9	18	.....	27†
	40	21	25°	50°	70°
	20	9	4†	20	44
	40	21	18	36	80
	10	10	9	30	38
	19	9	9	25	54

Dist from entry o 1st crosscut, ft	Width of room stump, ft	% of coal recovered
.....	24	45
60	49	50
.....	21	81
.....	34	44
.....	37	96°
50†	48	50
.....	26	54
.....	27	56
60	48	58
65	33	63
.....	38	55
.....	18†	68
50†	49	56
.....	35	55
50†	39°	63
.....	28	42†
70°	45	55

† Minimum for district. ° Max for district.
(a) Small section, worked by room-and-pillar.

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Turn the last room and the 3rd cross entry first. Begin robbing at inside corn develop rooms only fast enough to keep in advance of robbing and bring robbin a uniform "break-line," pleted to the barrier pill NO 3 illustrates a contin Drive entries to the l turning rooms, except Turn the last room on t entry first and begin soon as the second ro pleted. Develop room enough to keep in adva bing, and bring robbing a uniform break-line un of mining is reached. form conditions such pl quite closely followed (

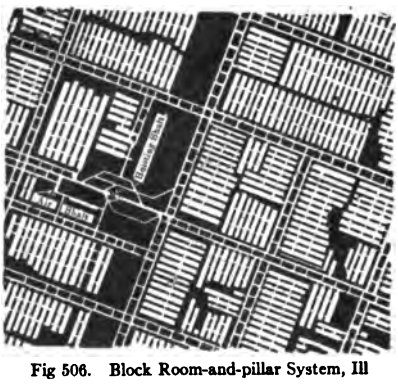


Fig 506. Block Room-and-pillar System, Ill

Illinois. Seams are flat; aver depth, 200 ft. is firm, with partings widths; seams, 2.5 to Roof characteristics va generally fire clay, likel when wet. Straight room-and-pillar, panel systems, and a modified panel syst as the "block room-and-pillar" or "semi-panel" (Fig 506), are all in use. For of openings see Table 71 (361). Some long-wall mining is also done (Art 108).

Indiana. Panel systems of room-and-pillar work with double-entry development are in universal use. The following figures of practice of the Brazil Block Coal Co are fairly representative; seams are from 4.7 to 10 ft thick; depths, 80 to 600 ft. Roof is sandstone or shale; floor, fire clay. Rooms, 18 to 30 ft wide, usually 21 ft; necks, commonly 9 ft wide and 12 to 20 ft long. Width of main-entry pillars, 18 to 40 ft; cross-entry pillars, 15 to 30 ft; room pillars, usually 9 or 10 ft wide, occasionally, 20 ft (362).

British practice. Room-and-pillar methods in England (bord-and-pillar, post-and-stall, pillar-and-stall), and in Scotland (stoop-and-room), resemble those in the U S, except in dimensions of openings and percentage of recovery on first mining. Common dimensions (363): entries (walls), 6 ft wide; rooms (stalls), 12 to 15 ft wide; room pillars, as large as 132 to 198 ft square. Recoveries are about 30% to 35% in first mining; first-mining recoveries of 40% to 50% are considered dangerous. Most of the coal is won during robbing (Art 107).

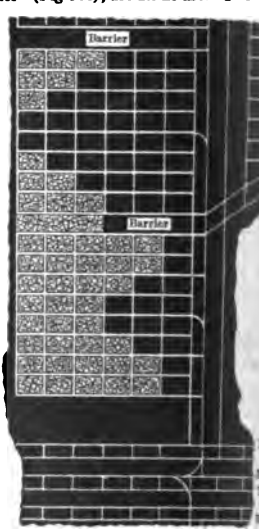
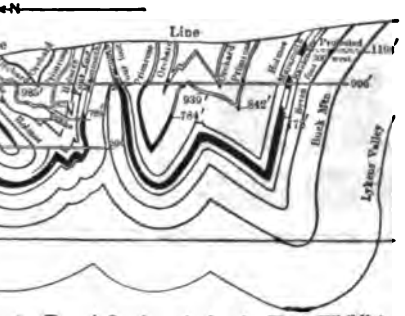


Fig 507. British Practice, Bord-a on Panel System (Hughes)

106. Examples of Room-and-pillar Mining in Pitching Sea

General. Pitch-mining practice has been developed most extensivel anthracite basins of Pennsylvania; Fig 508 (364) shows the relative po several seams and indicates a wide variety of local conditions. Virgin s

ith buggy, chute, or battery breasts (Art 104), obbed (Art 107); the Natalie Colliery illustrates ine more complex methods for special conditions.



on ng Tunnel, Southern Anthracite Field (Whitfin).

e Field, Penna. The Lykens seam occurs here in 2 hard conglomerate. The seams are very irregular; in the texture of the coal are common. The under- bout 4 ft of shelly, friable, badly crushed coal, with ck; it contains a little gas. Roof and floor are hard

Lykens No 2) averages 4 to 5 ; roof and floor, conglomer- vertical depth of workings, -track slope, driven on pitch sted to the surface. Gang- apart along the pitch, are rock, on a grade of 0.75% ft above the gangway, has rying with the thickness of ickness of seam, are turned 50-ft centers. Breasts are ving a 10 to 30-ft chain pil- Headings, turned on 60-ft Coal is conveyed from the utes. Lykens No 2, opened larly. Forced ventilation is due to opening of doors for se of small quantity of gas.

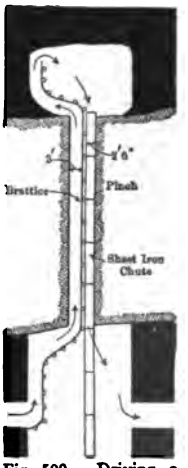


Fig 509. Driving a Breast Thro' a Pinch (Natalie Colliery)

mine in the Middle Western ns have an aver dip of 20°. ick; roof and floor, good. ers and worked with sheet- he car, miners doing their easts advanced in one sec- gradually from 20° to 60°. occurred, and the fact that r basis of payment, it was er gangway and work bat- provide safety and support of coal, staggered batteries were employed when the ows poor mining, but illustrates variations of prac- ogical conditions. Many other variations are in use.

widely employed in mining contiguous seams. in the underlying seam and rock-holes (inclined

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raises similar to chute-raises, Art 68) are driven to the overlying seam gangways or breasts, or both. Breasts may be opened in the overl directly from the rock-holes, or from small gangways driven to connec of the rock-holes. This practice often effects large economies in dev

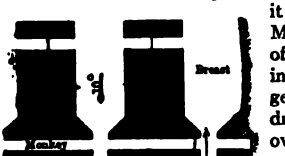


Fig 510. Driving a Gangway Through a Pinch (Natalie Colliery)

it is especially useful in reworking Mammoth seam, which contains large of coal left by the crude early methods. ing old gangways in the Mammoth sea gerous and costly; it is often impossibl drive new gangways in the vicinity of over areas.

One or more contigu may be opened by t across the strata from a ageway in one seam. T poses the possibility of maintaining gangways i seams. Where a series tunnels is driven to reach an adjacent seam they are known as SECTIONAL TU are spaced at intervals depending on their length, speed of development desire condition of the seam to be opened. These tunnels avoid the breakage of co holes, provide numerous working faces and therefore allow rapid devel- opment. Examples of rock-hole practice follow.

Lawrence Colliery, Schuylkill region, Penna. H. H. Morris, Supt, contributes the following. The Holmes seam, averaging 8 to 10 ft of firm coal, is mined by rock-holes from development openings and breasts driven in the Four-Foot seam, which is 5 to 8 ft thick; the seams are separated by about 10 ft of rock; pitch, about 60°. Gangways and monkeys are driven on 30-ft centers in the Four-Foot seam, with chutes turned on 50-ft centers. A few feet above the high side of the monkey a battery is built, and enough coal blasted to allow a 6 by 8-ft rock-hole to be driven on a pitch of about 40° through the top-rock to the Holmes seam. The rock-hole is partitioned off to make a manway and coal chute. On entering the Holmes seam, a battery breast is driven 24 ft wide up the pitch to the desired length. To secure ventilation, 2 or more rock-holes, breasts and headings must be opened in the Holmes; a split of air from the Four-Foot monkey is carried up the manway of one rock-hole, through the workings in the Holmes and down another rock-hole manway into the Four-Foot monkey. After the Holmes breasts have been driven and drawn, breasts of the same width are driven in the Four-Foot seam directly under those in the Holmes. Pillars must all be of the same size. Sometimes the mined coal in the Holmes breasts is held in reserve and not drawn when a breast is finished, in which case an extra battery is built in the rock-hole, for opening the breast in the Four-Foot. In this part of the mine, surface conditions prohibit robbing; it would be feasible, however, to rob the pillars in the Holmes seam before driving the breasts in the Four-Foot. In another part of this mine, several seams, pitching about 65° and 1.5 to 30 ft thick, are mined by counter-gangways, counter-chutes and rock-holes (Fig 512). The colliery is opened by a "gun-boat" slope in the Seven-Foot seam. The main level is at 634 ft elev; counter-levels at 820 and 1 000 ft elev; surface at outcrop is approx at 1 200 ft elev. (In Penna a mines, the elev of gangways is given in ft above sea level.) The method of m thinner seams is about the same as that at the Natalie Colliery (above), excep breasts are of the battery type and worked full. It was found impossible to dr ways in the Mammoth seam, due to the condition of its old workings; hence, r (6 by 6 ft) are driven from the Skidmore or Leader seam at 60-ft centers on the and at 60 ft centers on the pitch from the Skidmore breasts. Interval between and Mammoth seams, 20 to 60 ft. All coal mined on the counter-levels is tra to a counter-chute in the Leader seam (a 12 to 18-in split of the Mammoth app

Fig 51 gered

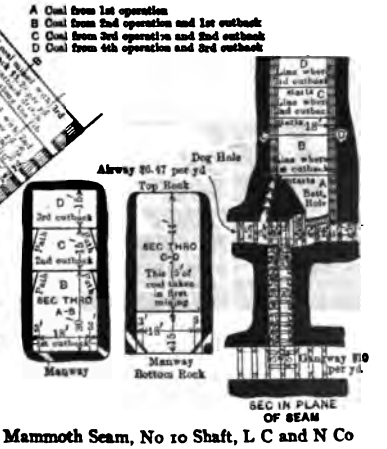
re and Mammoth). This counter-chute extends in angway to a point above the 634-ft level, where a

Fig 512. Counter-chute Method (Lawrence Colliery)

are
d for working
ings are driven
is 90 to 100 ft
vals of 300 to
d connected at
e Mammoth.
gangways, and
he rock-holes,

s. By driving the buggy gangways in both direc- of development doubled. The rock-holes furnish in robbing; the method avoids maintenance of mmoth.

Lansford, Penna. Data from W. G. Whildin in -backs" are used for mining the Mammoth seam.


A Coal from 1st operation
B Coal from 2nd operation and 1st outbreak
C Coal from 3rd operation and 2nd outbreak
D Coal from 4th operation and 3rd outbreak
Dog Hole
Airway 86.47 per yd
Top Hook
SEC THRO
Manway
Manway
Bottom Rock
Gangway 110 per yd
SEC IN PLANE OF SEAM

Mammoth Seam, No 10 Shaft, L C and N Co

e gangways and airways are driven along the top e driven back on a 30° pitch; this provides a safe

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working place for loaders, and allows better control in drawing the broken c seams, gangways are driven on the bottom-rock and the airways either al or top rock; distance between gangway and monkey, 20 to 50 ft. The results in a lower maintenance cost of gangways.

The method of opening a breast is shown in Fig 513, 514. Chutes 6 by 6 off to make a manway and a loading chute, are driven from the gangway o

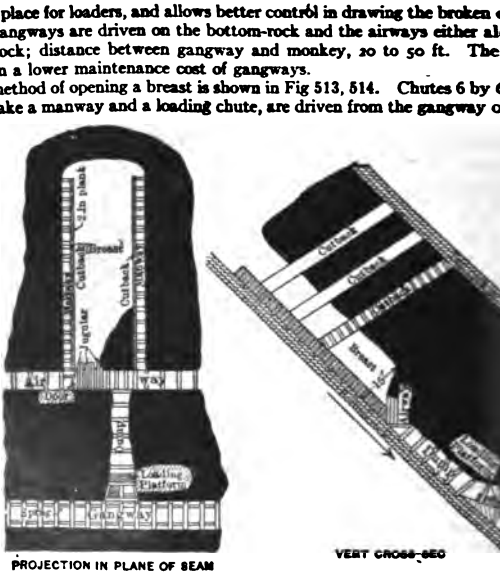

Fig 514. Cut-back System, Mammoth Seam, Penna

to the bottom rock and then up the pitch to the height at which it is decid stump heading. Here a jugular battery is put in and a 4 by 6-ft stump he the next chute. This is used both for ventilation and access, and is also "bottom-breast crosscut." From it, a "front" manway (dog-hole) is dri of the breast 20 ft up the pitch, and a "back" manway is carried up the o breast is started 10 ft wide by 8 ft high next to the back manway and is gra to 18 ft at the top of the dog-hole, leaving a stump pillar (Fig 513). Minin

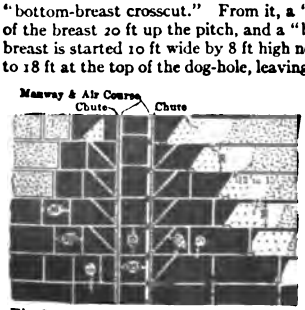

Fig 515. Mining Method, Tesla, Cal (Coal Miners' Pocketbook)

by advancing an ordinary batte wide by 15 ft high, on the bot distance of 21 ft above the botto cut. At this point a cut-back, 1 the pitch, is carried up to the to to rib of the breast. "After th has been completed the breast farther up the pitch, where a s is made. Where the pitch is 45° does not fill the whole space n case the miners carry a 'path' al the cut-back. Two miners blas traveling his own path, until t reached and all the coal is take the first and second cut-backs. again driven up the pitch 15 ft, where a third cut-back is made. If the brea away, or the top coal does not fall, a cut-back is made every 15 ft to the limi In steep-pitching veins, no paths are necessary, the miners standing on the loo the cut-backs." For further detail and variations of this ingenious method

Tesla, Cal (Fig 515) (376). The seam averages 7 ft of clear coal; pitch, tem was devised to get coal rapidly in a part of the mine where a short-gra rock came in over the coal, making it difficult to keep props in place. Th

e roof is an excellent sandstone; there is a small but double-compartment chutes are driven up the pitch gangways are driven from the chutes parallel to the long the pitch. These are continued 300 ft from each end of the pillar between the highest small gangways orked on the cleavage planes. The resulting breast e drift or gangway through which the coal is carried left between the breasts to keep rocks from falling on each breast, 45 to 48 ft long. Light iron chutes are mall gangways; chutes are moved along as the face m breakage is small. Coal is dumped from cars into e main chutes on slopes of about 45°; these serve to ch breast tinuously, ast gang- The small s is main- vanced 12 oof caves the faces crosscuts, intervals. sed in a d breaks pidly. A are con- o-ft inter- driven up these are rt. This which are

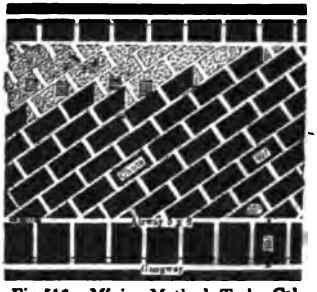

Fig 516. Mining Method, Tesla, Cal (Coal Miners' Pocketbook)

ard. Rows of cogs are built about 80 ft apart, to delay aining pillars from crushing; little timber is required.

Robbing Pillars

illars consists in removing the coal left for roof support d. The character of roof and floor, texture of coal, e of gas and other local conditions, all influence the ch it should be done.

re are 2 general systems: robbing during the RETREAT. A further distinction may be made LETE OR FINAL robbing.

shows its application in a flat seam with rooms First-mining is followed closely by preliminary pillars; the stump pillars are left to support the pillars are removed on the retreat system after bbing have been completed to the limit of the d. It is always important to draw the pillars faces in a straight line (Fig 485); this avoids ny one pillar and secures a uniform line of break e line of retreat depends chiefly on the character as robbing proceeds; otherwise the pressure on room pillars may be sufficient to crush them or NTAGES: (*a*) max production per unit area of ss caused by deterioration of pillar coal on long eld of lump coal in soft or friable seams, because pressure can cause extensive crushing; (*d*) im- us mines; (*e*) smaller room pillars may be used,

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due to short life and limited area of support. DISADVANTAGES: (*a*) accumulate in caved areas, which can not be ventilated, and there is alw from fire in such areas; (*b*) where roof strata are water-bearing, the brea roof increases amount of inflow; (*c*) comparatively large stump and entr necessary to prevent squeezes and avoid excessive cost of maintenance

Retreat system is the reverse of the advance. Development driven to their limit, rooms turned, and first-mining is completed thro entire area before robbing begins. Robbing is begun at the inside l area and retreats toward the main or sectional entries; the robbing is *i e*, the room, stump, and entry pillars are extracted in one operation. nation system known as the ADVANCE AND RETREAT SYSTEM is sometimes used where rooms are turned in both directions (Fig 503). Rooms are turned in one direction only as the development entries advance,

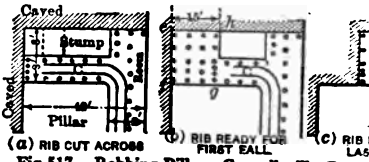

Fig 517. Robbing Pillars, Connellsville Regio

and room-pillar robbing is carried on during the advance. When th the entry is reached, rooms are turned in the opposite direction and retreats toward the main entries, final robb carried on during the retreat.

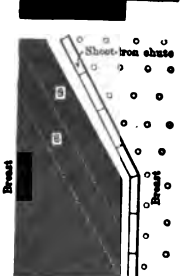

Fig 518. Robbing Pillars, Inclined Seam

Examples of American practice. Details vary underground and surface conditions and the quality of labor available; many cases require special methods. CONNELLSVILLE REGION, Penna. Fig 517 shows pillar-drawing in flat seams. Robbing in any pillar begins at the face of the room and retreats toward the entry. A cut *C*, 8 ft wide, is driven through the pillar, leaving an 8-ft stump next to the caved area. Cut *C* is timbered with props. Cuts are then made across the stump, as indicated by dotted lines in (*a*), and the roof is picked up by props until the work reaches stage (*b*). The props in area *efgh* are next drawn or shot out, forcing the roof to fall. The rest of the stump is then mined and caved, and the process repeated until the whole pillar is drawn back to the entry pillars. The latter are removed by similar methods (358). This general plan, with modifications of detail, is in universal use in the U S for drawing pillars in flat seams. ANTHRACITE MINES, Penna. Methods are unsystematic and often left to the miners. Pillars are usually robbed by taking slices ("skips") off the ribs, the roof being temporarily supported by props. Robbing begins at the upper end of breasts. Fig 518 shows work in chute breasts. Skips *S* are driven at an angle across the pillar, so that the chute may be kept close to the face and loading cheapened. Fig 519 shows a method of robbing on steeper pitches in thin seams. The skips are 5 to 6 ft wide; the direction of work is indicated by arrows. This method is feasible only where a passageway for men and coal can be kept open along the pillar rib. pitches (over 60°) in thicker seams, and where the roof caves close to the pillar ri

Fig 519. Rob lars by "Skips. Pitching Sea grammatic)

(Fig 520) is used, either of necessity or because of its a chute P, 4 to 6 ft wide, driven from the gangway. rried on by inclined skips, as at (*a*), blasting out the small pillars L with skips up the pitch from a heading, should be employed for robbing; the of pitch and thickness of seam. A port "flakes" of roof-rock, and gives r miners to reach safety. No props roof comes in, and aid in keeping to the working face.

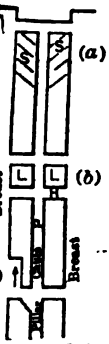

Fig 520. Robbing Pillars, Thick, Steeply Pitch'g Seams

n and R. A. S. Redmayne give the s in England (365). SEATON BURN COLLIERY, Northumberland. Fig 521 shows robbing in a flat seam, averaging 4 ft thick, at a depth of 360 ft; pillars are 30 yd long and 16 yd wide. A "way" A (skip) is driven 6 yd wide the length of the pillar. A small pillar P 2 yd square (a "stook") is left to support the haulageway; a track is run along the solid rib and cogs are built on the open or caved side 3 ft apart. When the "way" has been driven the length of the pillar, a "lift" B, 6 yd wide, is driven across the pillar. Five

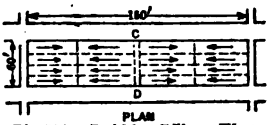

Fig 522. Robbing Pillars, West Durham

lar. This seam nd-on. MARLEY ILL COLLIERY, West Durham. Fig 522 shows method used in flat seam, which verages 5 ft thick nd contains a 4- n and a 1.5-in parting in the middle. Depth of seam, 32 ft. Pillars are 50 by 20 yd. The pillar is first split n its short dimension by a "sliding-over" or "half-illar wall" CD. From this and from the haulageways -yd lifts are turned and the work completed as at. Seaton Burn Colliery. MURTON COLLIERY, Durham. Fig 523 shows the method in a 46-in flat seam, at depth f 470 ft. Lifts about 6 yd wide are driven at right quence shown. This method retains the square shape ding heavy pressure, throughout the entire operation.

ongwall Methods

complete removal of the entire seam in one operation. long, continuous working face (hence the name long- g the roof to cave behind the face.

rried out in a flat seam, is shown by Fig 524 maintain the hoisting and air shafts; successive nced outward from the shaft pillar, are indicated breast is advanced, the roof between it and the re built on each side of the numerous haulage- through the caved area (GOB or GOAF) to reach The weight of the roof often causes the pack-

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walls to settle; headroom is maintained by breaking down (brus
over the haulageways; the waste thus produced, together with tha
is utilized in packwalls, the excess being thrown back into the g
roads are known as MAIN and BRANCH or CROSS ROADS, haulagew
(Fig 524). Fr
roads, short o
STALL ROADS
kept open at s
they are appro
lar to the face
advances, new
stall roads ar
the old ones al
keeps the sta
and saves mai
Haulage tracks
from the stall
along the face
coal may be sh
into cars; var
conveyer have
use for transpo
the face (369).

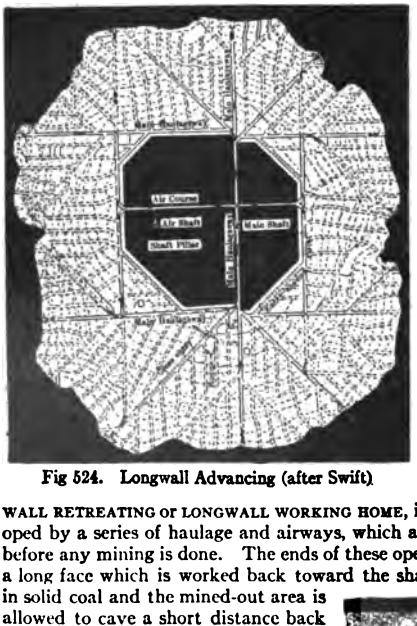

Fig 524. Longwall Advancing (after Swift)

This genera
is known as
VANCING OR LO
ING OUTWARD.
another form
WALL RETREATING OR LONGWALL WORKING HOME, in which the sea
oped by a series of haulage and airways, which are driven to the
before any mining is done. The ends of these openings are then j
a long face which is worked back toward the shaft; the roads a
in solid coal and the mined-out area is
allowed to cave a short distance back
from the face.

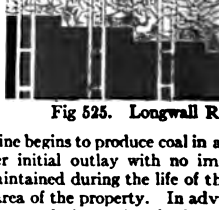
Fig 525. Longwall R

Advancing *vs* retreating systems. CAPITAL OUTLAY. Longwall advancing re-
quires smaller outlay for development than
any other coal mining method. Narrow
work in driving haulage and airways is
eliminated, with the exception of the devel-
opment openings in shaft piliars, etc. The mine begins to produce coal in a
time. Longwall retreating requires a larger initial outlay with no im
Extensive development openings must be maintained during the life of th
requirements are in direct proportion to the area of the property. In adv
MAINTENANCE of haulage and airways is a constantly increasing deadwor
openings pass through caved ground and the packwalls supporting them
trouble from settlement. Longwall retreating entirely avoids the cost
maintaining packwalls; all haulage and airways are in solid ground.
more efficient in the retreating than in the advancing system. Caved c
are difficult to ventilate. Gas if present may accumulate in the gob, a
sions and gob fires seriously affecting the active workings. Retreating sy
the necessity for ventilating the gob. Longwall retreating is rarely used,
of the large investment required; what follows refers to longwall advanci

Longwall *vs* room-and-pillar methods. ADVANTAGES of longwall, a
connected with investment and quick return: (*a*) smaller powder co

ble storage underground for waste; (*c*) fewer roadways ng first and maintenance costs; (*d*) easier and cheaper cident from roof falls; (*f*) it requires less timber; (*g*) and causes less damage; (*h*) duty of labor is greater; ve DISADVANTAGES of longwall: (*a*) labor must be et demand must require a uniform production at all ob fires in advancing systems; it is difficult to divide to localize effects of gas or dust explosions; (*d*) cost the successful application of longwall are quite rigid. ehind the face; a very strong roof or one that breaks f longwall. Faulty ground makes it difficult to main- Waste for packwalls must be cheap, whether obtained oof or floor (see Art 103).

essure caused by the settling of the roof is utilized ig 526, 527 show sections through typical faces. e is sup- bout 5-ft ly by the ance be- nly 10 to re placed ry 3rd or a timber

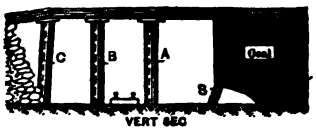

Fig 526. Longwall Face

The face is undercut by hand or machine, and is *S*, set on 4 to 6-ft centers. Sometimes lateral ermegs" *E*, Fig 527, is also required. When sprags are knocked out and the face is allowed s the roof pressure breaks the coal down; fre- ed or blasted. The broken coal is then loaded, w of props set near the new face, packwalls are extended, the excess waste is gobbed, and the tracks are shifted. Props *C* are pulled if possible and if sound are reused. For further detail, see Practice in Illinois, below.

Distance between stall roads varies from 36 to 150 ft. Hughes (363) summarizes the factors determining this dimension as follows: (*a*) the face between 2 stall roads must be capable of being advanced regularly and rapidly, which is difficult s spaced too closely involve excessive first and main- lor cars to run along the face require close spacing of ll roads depends somewhat on the output required. ce" (the section of working face between 2 stall roads) (*e*) custom as to how many men work at one "place" e distance between stall roads.

or inclined seams are chiefly to secure a working irections of which allow convenient handling of oads are run to the rise; the working face being ore horiz. On DIPS OF 5° TO 12° OR 15° the roads , thus keeping grades within the limit of hand °, are sometimes worked to the dip instead of to ll roads by gravity, pulling a hoisting rope after is about 30°, where packwalls begin to slide. ON wall) or stepped-face stopes are best (Art 60); the or mill-holes. See Bib (355, 363, 367, 370) for n longwall practice in flat and inclined seams.

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Practice in Illinois. Data from S. O. Andros in 1914 (366). This is
American regions where longwall has been successful. Table 72 shows
acteristics of the seams and dimensions of workings in 11 mines.

Table 72. Data on Illinois Longwall Mines

Mine No	Depth of shaft, ft	Thickness of bed, in	Size of shaft pillar, ft	Distance between cross entries, ft	Angle betw cross and main entry = angle betw room* and cross entry, deg	Distance between room centers at face, ft	Wid road: Main and cross
1	413	39	400×600	225	45	42	9
2	465	44	250×250	225	45	42	9
3	398	42	550×550	240	45	42	9
4	546	40	(*b*)	200	70	42	10
5	135	36	360×560	275	45	42	7
6	100	36	150×300	225	45	42	7
7	200	37	350×450	.....	45	42	8
8	300	40	.....	225	45	42	9
9	(*a*)	42	.....	320	30	40	8
10	480	42	500×500	225	45	42	9
11	530	34	600×3600(*c*)	225	45	49	8

(*a*) Slope. (*b*) No pillar. (*c*) Protects 3 hoisting shafts. * Sta

Note. — Immediate roof is shale, except for small areas of mine No 5 sandstone. Floor is fireclay, hard, sandy or soft. Mine No 1 has sandsto into fireclay; in mine No 3 a hard sandy shale underlies coal in places. C contains lenses or bands of clay, pyrite, or sulphur balls, ranging from 2

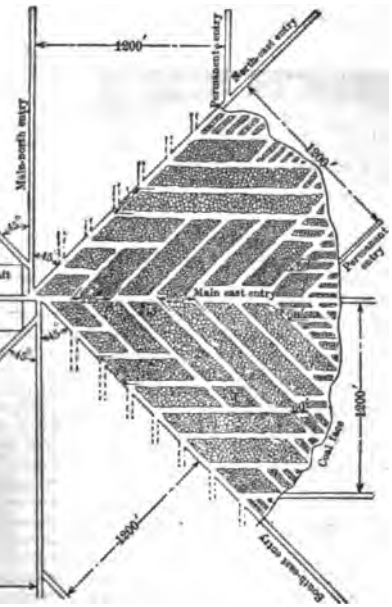
Fig 528. Blocking Out Shaft Pillar, Ill

Fig 529

All of the mines are worked by longwall advancing and with one exceptio shafts. The greatest difficulty in starting operations is to form the shaft tablish the working face. Usually in this district, after the hoisting shaf have reached coal, a main entry is driven about 225 ft from each side shaft. From the air shaft 2 entries are driven in opposite directions at the main entry, to the edges of the proposed shaft pillar. The latter is out by driving around it a 9-ft entry *E* (Fig 528), called the "entry-around-pi Large pillars are desirable, to protect the shaft and provide a long working 9-ft entry is then driven around the shaft pillar, parallel to *E*, leaving a 15-ft the two; breakthroughs between the entries are about 42 ft apart. Fig alternative method, used where the coal in the shaft pillar spalls off; th

he 15-ft space between the 2 entries of Fig 528 is also entry 27 ft wide is driven around the pillar and 2 pack-es. One packwall 12 ft wide is built alongside the shaft ture longwall face, leaving a haulage road 9 ft wide be-sary openings through them for haulage.

rst break of the roof, as the workings extend outward ent and will destroy the entry-around-pillar unless the onths are required for driving entries through the shaft hoisting and air shafts reach the coal. Actual mining ries-around-pillar are connected and direct ventilation no pillar was left to support the roof around the shaft; e roof to settle gradually till it rested on the floor. In



530. Longwall Mine

pported on soft wood 12 by 12-in posts, and the coal is ft. The space thus left is filled with soft-wood cogs ing and mining rock. Through the gob a 7-ft roadway each end of the shaft. As the roof settles the packs ueezed into the fireclay bottom till roof and floor meet. and timbered. Advantages claimed: the expense of ced, and the roof weight begins sooner to ride on the ft pillars state that these advantages are offset by the subsidence sufficiently to prevent the shaft from being d Table 72 show the usual arrangement of entries and ngement used in 1 mine see Fig 530. In pitching seams working faces are kept parallel to the dip and coal is sheet-iron chutes.

. The "places" are 42 ft long; their limits are marked e to scarcity of labor, about 50% of the "places" are

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worked by 1 miner; the rest by 2 miners. The crew on a "place" is responsible ing packwalls and for gobbing as well as for mining. Where possible an undercu in high and 2 to 2.5 deep is made by picking in the fireclay floor. Sprags are apart; props are set 2 to 5 ft from the face and 6 to 8 ft apart. With an aver undermining, a good miner can undercut about 20 ft of face a day when work clay 8 to 12 in thick. For loading a car, that portion of the coal is taken which standing longest on sprags. These are knocked out, and if the gob has been filled, so that the roof weight rides on the face, the coal breaks from the roof and for loading; otherwise, the coal is wedged down. Under fairly good conditions a of the coal exceeds 1.25-in size.

If the floor is sandstone, or if the fireclay is much over 18 in thick, undercutti in the coal itself; this produces much slack and increases the number of gob fire more fine coal goes into the gob with the waste. To save time and labor the m neglects to support the coal on sprags until the usual 2 ft of undermining is c instead he makes a cut 4 to 8 in deep and pries down the coal. This does not slow breaking power of the roof; dents occur, and more slack and sm result than when full undermining upon. For blasting black powde where necessary. No undercutting are used in the district.

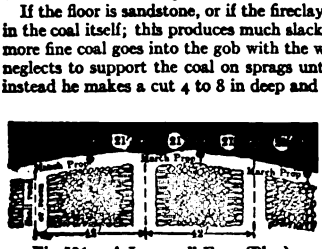


Fig 531. A Longwall Face (Plan)

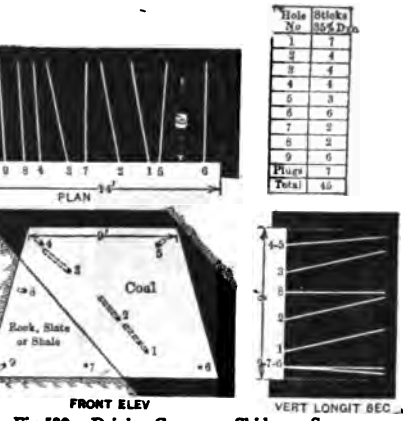
The rock obtained from brushing that remaining after building the and the clay obtained from underm coal, are piled in the gob area be packwalls lining the roads, to help support the roof and control its pressure on After the first break at the shaft pillar and face, if the gob area has been prop so that the roof weight "rides" on the coal, subsequent roof breaks occur every 2 they parallel the face and extend upward and backward from it at angles of 50 cracks are more nearly vertical on faces which are worked slowly. The distanc breaks depends principally upon the character of roof and the packing of the proper packing it should correspond to the width of coal brought down. The c which breaks extend into the roof depends upon the kind of rock; it rarely exc Squeezes which fill a working place with roof material occur when a room is dri of adjacent rooms, or more commonly when the packwalls are defective and th is not sufficiently filled. There should be enough filling for the roof to co gradually without breaking off short at the face of the packwalls, but not so insufficient weight is thrown on the face of the coal. The better the gob is pa better the coal "works." When part of the working face squeezes, the face is verted to pass around the squeezed area, sometimes leaving a small block of coal

The necessity for artificial humidification to prevent coal-dust explosions ha apparent in these mines. Since all the coal is removed from the seam as the face and the excavation is filled with waste rock, the only sources of dust are the wo and the spillings from mine cars. Dust from the face is covered with shale and in a few days after it is made and does not accumulate. As the aver temperat air in these mines is higher than in room-and-pillar mines, the relative humidity is and considerable moisture is absorbed from the dust of ribs and roads. In a the haulage ways are sprinkled at intervals of 1 week to 3 months (Sec 23, Art 1

109. Breaking Ground in Coal Mines

General. Breaking ground (Art 26) is, in coal mines, sometimes ca TING COAL. The principles are the same as those of breaking ore or roc that coal is relatively soft and brittle, and as a lump product is generally the explosives used are of low power and the number of holes in a giv minimum. No definite rules can be laid down with respect to locatio holes and charge of explosive required, for the following reasons: WIDE ING LOCAL CONDITIONS are encountered in different seams, or in succes in the same seam. Slate partings, "blowing benches," pinches or fau slips, cleat, and the character of the coal itself, whether firm or shelly, experience for properly locating and charging holes. CONTRACT WORK

coal or yard driven; they purchase their tools, ay their laborers. Since a misplaced or improp- earnings, both by increasing his supply account



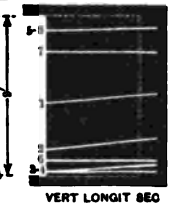
Hole No	Sticks 35% Dyn.
1	7
2	4
3	4
4	4
5	3
6	6
7	2
8	2
9	6
Plugs	7
Total	45

Fig 532. Driving Gangway, Skidmore Seam

cut coal." Thus the helper will learn from his miner, ith or without improvement.

ork to suit the conditions and keep his laborers busy. power augers, mounted on bars or posts. Recently, small hammer drills, like the Jackhamer (Sec 15), have been successful in driving gangways, especially in the Southern Anthracite Field, where work in thin seams requires the breaking of considerable top or bottom rock. To facilitate blasting and obtain a maximum lump product, coal cutters (Sec 16) are widely used in bituminous mining and to some extent in the Northern Anthracite Field. The chain machines are limited to pitches of less than 20° to 25°; those of the "post-puncher" type are successful in the semi-bituminous coals of Washington on pitches as great as 38° (371).

Hole No	Sticks 35% Dyn.
1	5
2	5
3	2.5 to 3
4	2.5 to 3
5	4
6	11
7	4
8	4
Plugs	6
Total	45



Buck Mtn Seam

Explosives used are black powder, low power nitroglycerine or ammonia dynamites, and, in gaseous or dusty ec 4). The rending effect required to produce a large by using black powder or 15% to 20% dynamite; or blasting slate or rock, and modes of charging and

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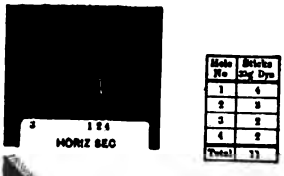


Hole No	Sticks 25% Dyn
1	4
2	3
3	2
4	2
Total	11

Fig 537. Driving Breast, Buck Mo

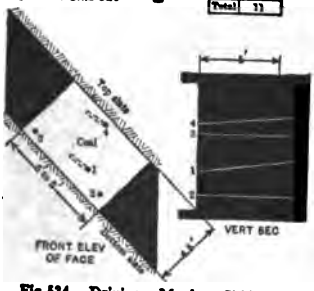


Fig 534. Driving a Monkey, Skidmore Seam

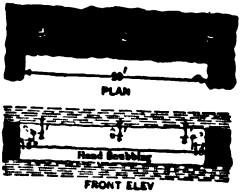
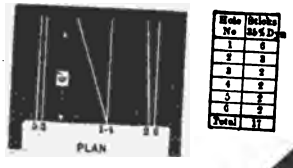


Fig 538. Mode of Placing Shots a Snubbing



Hole No	Sticks 35% Dyn
1	4
2	3
3	2
4	2
5	2
6	2
Total	17

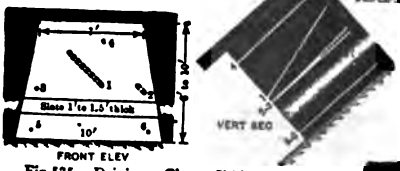


Fig 535. Driving a Chute, Skidmore Seam

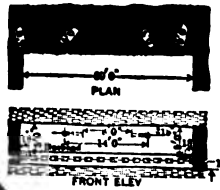


Fig 539. Mode of Placi Puncher Undercuttin

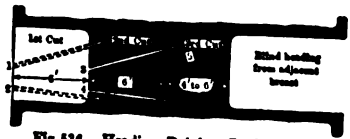


Fig 536. Heading Driving, Buck Mountain Seam

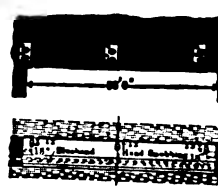


Fig 540. Mode of Placing Shots Undercutting Machin

ERN ANTHRACITE FIELD, Penna. W. L. Cross, Jr, angways, monkeys, chutes, headings, and breasts, in ams at Buck Run Colliery, Minersville. Fig 532, 533 Holes in coal are drilled and fired first, the broken l. Then the rock holes are fired, bottom rock being er round, 5 ft, with an aver consumption of 25 lb of e of driving a monkey in firm and hard coal, drilling n near the chute an advance of 1 cut or 5 ft can be the chute increases, the advance is slower, because 35 shows mode of driving a chute entirely in coal on a han 1 cut of 5 ft per shift is made; aver powder con- riving a heading in the Buck Mountain seam (Fig 536), l heading through the pillar, a heading from the next le No 1 is charged with 4 sticks of dynamite, and the is drilled, charged with 3 sticks, and fired; the result- econd cut is made by holes 3 and 4, after which hole ut breaks through to the blind heading, a second hole, ary, as the rib can be squared up with a pick. The ept that usually there are only 2 cuts. In the Buck in Fig 537. All holes are drilled in the bottom bench ; the portion most easily drilled and giving the best es making an aver round. Cut holes are charged with % dynamite. If it is necessary to break the "middle harged with 1.5 sticks of 20% dynamite. Sometimes m each rib, instead of one in middle of the breast.

. ILLINOIS METHODS. In flat seams, especially in ut (by hand or machine) before blasting. This re- es danger from blow-outs and increases the yield of ng modes of placing holes in faces undercut by hand erican practice.

ing in Illinois Room-and-pillar Mines (361)

Holes			Black powder		Method of firing	Per cent of coal over 1.25 in
No per round	Length ft	Diam. in	Size of grain	Lb per ton of coal		
3	7	2	F	0.14	Sq	(a)
3	6	2.25	C	0.13	.....	72
3	6	2	C&F	0.22	Sq	71
3	5	2	C	0.25	Sq	70
4	6	2.75	C	1.25	Fu	65
.....	7	2	C	1.47	Fu	60
.....	6.5	2.5	F	0.78	Fu	60
7	8	2.5	C&CC	1.00	Fu	86
4	5	2	F	0.16	Sq	73
4	5	1.5	F	0.21	Fu	72
3	4.5	1.75	FF	0.31	Sq	70
3	4.5	2.25	F	0.73	Fu	70
.....	8	1.5	C	1.25	Fu	50
.....	7	2.5	C	1.00	Fu	40
.....	7	2	F	1.56	.....	71
.....	7	2 5	CC	1.39	.....	70
4	5	2	C	0.23	Sq	70
.....	5	1.5	FF	0.22	.....	72
2	4.5	1.5	F	0.50	.....	65
.....	6	1.5	C	0.19	.....	75

e. P = Puncher. Sq = Squibs. Fu = Fuse. OS in. (b) Aver daily output per miner, 7 tons.

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110. Flushing

Flushing ("slushing" or "silting") corresponds to sand-filling in metal mi
It was probably used first in 1884 in Pennsylvania for extinguishing a mine
been extensively developed in the U S and abroad, not only to check and isol
also to regulate subsidence (Art 112), and to dispose of culm banks; its use le
pollution near anthracite breakers. Following notes are taken largely from
U S Bureau of Mines, by C. Enzian in 1913; see also Bib (372, 373).

Materials for flushing. CULM (Sec 34) is excellent where ava
easily drained and when thoroughly packed resists heavy pressures.

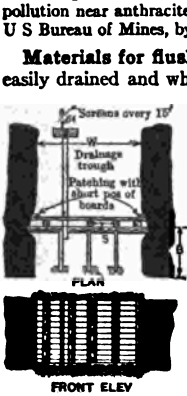
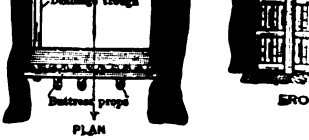


Fig 541. Bulkhead for Dips Under 10°

culm will often stand in a v
without timber, and can be dri
without spiling. Having lo
qualities, culm is readily co
water in pipes and launders.
flushing is decreasing with th
sale for finer sizes of coal. A
times available at mines o
steam power, wear pipes more
but resemble the latter in oth
A serious drawback to their us
cite mines is the fact that the
coal mined next to flushed are
not be separated from the fi
the breaker. BREAKER REF
slate, and bone) has good
qualities. It requires crushing to less than 1.5 in and is harder to t
water than ashes or culm, but rarely chokes properly designed pip
GRAVEL, etc, are efficient, but very abrasive. Though widely used in S
have thus far proved too costly in the U S. GRANULATED SLAG is used abroad in mines near blast furnaces. It is an ideal filling material, having natural cementing qualities and being strongly resistant to pressure.



Modes of handling filling material on the surface, between the surface and the mine, and underground, resemble those described in Art 93.

Few data are available on the lateral distance to which filling may be transported in pipes under a given head. Silesian practice allows 300 ft horizontally for each 100 ft of available head; the distance is obviously affected by size of pipe and thickness of pulp. Water in pulps varies from 40% to 90% of the total volume, depending on grade and size of pipe (or launder), effective head and character of filling material. TROUGHS (launders) (Fig 463) are used where possible

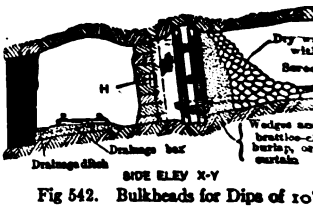


Fig 542. Bulkheads for Dips of 10°

pulp; they are often lined with cement, terracotta, or sheet iron. PIPES are carry pulp under pressure. WOOD-STAVE PIPE resists acid and abrasion, an

sections, is easily installed under difficult conditions; and dries out and collapses when not regularly in use; line; its life is comparatively short. STEEL or W-I aters, and is more quickly abraded than wood; it is od-stave pipe and generally costs pe has been used. C-I PIPE is nd, but resists abrasion and acid hat of W I and 5 times that of considerable deflections. Loca- determines whether this pipe is

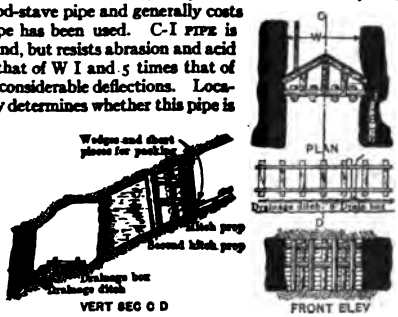
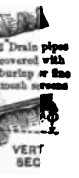


Fig 543. Bulkhead for Dips of 25° or More

U S. PORCELAIN-LINED PIPE is highly efficient for ceptional wearing qualities, resists acid and has a low d in Europe, but is too costly in the U S. In Penna, wood-stave pipes are probably most used for low heads, C-I or steel pipes for higher heads; much flushing is done through unlined bore holes and troughs; the latter are preferable to pipes.



Bulkheads (barricades, Art 93). Their design (Fig 541–544) varies with the dip of seam. They are placed a short distance *B* (Fig 541), the haunch ove the lower end of the working to be flushed. ves empirical formulas for barricades.

hows construction for flat workings; leakage at the evented by manure or dirt packing. Props are lagged of 1.5-in plank, or 2 layers of 1-in. Bulkheads are er on steeper pitches, and their drainage is carefully to avoid subjecting them to the full hydrostatic head hey are drained by troughs and screens (Fig 541, 542); nd dry-walling. A trough returns the water for reuse ther devices. Fig 544 shows an arched concrete bulk- easures and where timber is costly.

d Spacing of Props in Barricades

D = diam of prop, in	S = spacing of props c to c, ft	B = haunch distance, ft
D = L	S = W ÷ H	W + 2
D = 3 L ÷ 2	S = 2 W ÷ 3 H	W + 2
D = 7 L ÷ 4	S = 4 W ÷ 7 H	2 W + 3

height of opening, ft. L = length of prop, ft. Num- e = (W ÷ S) + 1.

present cents per cu yd and are based on data from t least 400 cu yd daily.

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Table 75. Costs of Flushing in the U S (C. Enzian)

Material	Surface transport			Intermediate transport		Underground tr'sp't		Distribution		
	Gravity	Scraper or conveyer	Pump	Shaft or slope	Bore hole	Trough	Pipe	Flat workings	Chute workings	Pitch workings
Culm	1–1½	4½– 6	2¼–4	¼–½	1/10–3/10	1/10–3/10				
Culm mixed with crushed breaker and boiler refuse	4–5½	7½–10	5½–8	¾–1½	3/10–5/10		3/10–5/10			
Local hydraulicked sand, loam, gravel, clay, etc	8–14	11½–18½	9¼–16½	5/10–1¾	3/10–7/10			7¼–9	12½–14½	14–27½
Local crushed sand, loam, gravel, clay, etc	30–40	.....	.....	1–1½	5/10–8/10					
Material from a distance in returning empty coal cars	Cost of material, loading, freight, unloading, preparing and surface transport			1–1½	5/10–8/10	Add 25% to specified				

111. Timbering. Use of Steel Supports

Timbering in coal mines is similar to that in metal mines (Art 1 differences in detail and terminology. Props, and 3-piece sets (Art 2 ways, airways, etc, are of round timber.

Common sizes of legs (posts) are 10 to 14-in diam, of collars (caps) 8 to 1 Penna, legs have a batter of 2.5 to 3 in per ft; in bituminous mines they are o Lagging is 3-in round or 6-in split, in 6-ft lengths. Sets are usually 5 ft apa ground they are doubled, at 2.5-ft centers. In moderately thick pitching ways may be timbered with a single row of lagged props along the high side, c to stull timbering in metal mines (Art 36, see also Fig 172). When the le props exceeds about 12 ft, they are replaced by 3-piece sets on flatter dips, or bar" (half or 2-piece sets, Art 21) on steep pitches. Rooms and breasts are ti props set with a slight underlie (Art 36); their spacing depends on the char Headboards are sometimes used; generally props are wedged against the r rooms or moderately pitching breasts, where the roof tends to flake off, c placed between props, at least over the track or chute. For further detail, s

Steel supports are increasingly used in both coal and metal mines for wide openings and those of semi-permanent character, as main ha pumprooms and shaft stations. STEEL *vs* TIMBER. Steel lasts much l cially in moist, foul air. It is fireproof and procurable in sizes tha heavier loads than any available timbers. Less excavation is required erection is usually less than for timber of equivalent strength. Disad steel are its relatively high first cost and the fact that it cannot be reac fitted underground. It may be required by heavy pressures regardl factors; its installation is often justified by saving in maintenance.

Proper size of steel for given service cannot be definitely computed, impossible to measure the weight to be supported. Steel members for repl

r equivalent strength. In a new region, their size is eel under similar conditions elsewhere.

d H-sections, of old rails, or other available material. boards are generally necessary, to prevent props from 545, 546 show examples of gangway sets made by the and sections of leg and collar. Legs are commonly ls (Fig 545); collars of I-beams; joints may be rigid

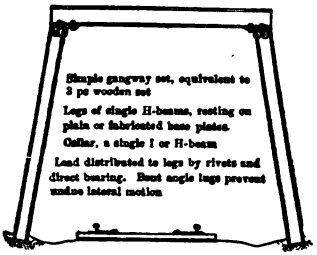
Simple gangway set, equivalent to 3 pc wooden set
Legs of single H-beams, resting on plain or fabricated base plates
Collar, a single I or H-beam
Load distributed to legs by rivets and direct bearing. Bent angle lugs prevent undue lateral motion

Fig 546. Gangway Set, Style F

oofs are often supported by I-beams, H-sections or pillars or walls, or in hitches in the coal ribs. Lag- construction, of old rails, sheet steel or corrugated ometimes turned between collars of adjacent sets. einforcement, may be built over the collars in pump- pports in shaft stations and pumprooms, see Sec 12 ription and data, see Carnegie Steel Co catalogs and

BSIDENCE

f Economic Geology, Sheffield Scientific School, Yale University

lems of Subsidence

15 is obtained from *Bull* 91, Eng Experiment Sta, 5 (465).

L MINES, the UNDERGROUND RESULTS of sub- f mining; the SURFACE EFFECTS are the chief nderlie valuable ground, and the mines usually ice of mining method, as affecting subsidence, ditions. The value of the coal left in pillars, support the surface, must be compared with subsidence should occur. In METAL MINES, ler; their workings are less extensive, they y underlie valuable areas. In them the chief d until the ore has been removed. In SALT is important, because entering water would trouble.

dence by filling might equal or exceed the value of of conservation enters; mineral lost by improper mines may be lost forever, but the surface can usually eral Gov't should decide whether all merchantable restoration of the surface, or to permit the per- trying to avoid temporary surface injury.

er subsidence affects the surface at all depths of ain depth; what percentage of the deposit left as

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pillars, or how much filling, would prevent subsidence; whether surface occur vertically above the excavation, or at an angle from the vertical; for of surface depression; effect of strata dipping at different angles. If sub pected, the ratio between surface area affected and area of excavation sh mined, so that surface structures may be removed or protected; the sur have to be bought or condemned. Effect of faults and joints in overlying r studied; also the possibility that subsidence might tap and divert a ground

Forms assumed underground are: flexures of the rock form (Fig 547), small faults, irregular cavities, and domes of variable commonly sharp. In elongated stopes, cavities usually have an unless affected by faults or fracturing (creeps) may result from the crushing of ground around the excavation.

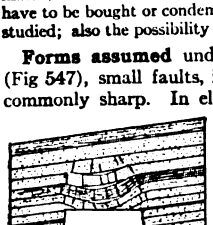

Fig 547. Subsidence over a Drift

Even if the back or roof does not bre effects of squeezes, or of insufficient suppo the surface, causing: (*a*) CRACKS, with or location or sinking (Fig 548); (*b*) movem FAULTS; (*c*) fracturing producing a crushe area; (*d*) broad shallow basins; (*e*) small cular pot holes; (*f*) large irregular cavit binations of the above.

Nature of damage may be internal (within the mine), or external; eit loss of life. INTERNAL DAMAGES are due to: (*a*) falls of ground; (*b*) sque injuring or destroying the workings; (*c*) caving above workings, causing ing, stoppage of ventilation, mixing barren rock with ore, or ruining the s blasts, destroying timbering and workings, and initiating further subsidenc insecure ground; (*e*) inrush of bodies of water due to subsidence reaching causing temporary abandonment, or partial or complete loss of the mine, as at Alaska-Treadwell mine (Art 69) in 1917; (*f*) inrushes of sand and mud, through tapping of pot holes, or soft, watery strata (this may occur by the action of water subsequent to subsidence); (*g*) tapping ground-water or underground channels, through dislocation of water-bearing strata; (*h*) destruction of overlying beds of coal, through crushing caused by working a lower seam first. EXTERNAL DAMAGES are due to: (*a*) destructive vibrations (small earthquakes) set up by sudden dislocations; (*b*) landslides (rare), as at Frank, Alberta; (*c*) cracking, dislocation, or actual sinking of the surface, injur roads, reservoirs, sewers, water pipes, or gas lines. Bridges have been tw way lines so disturbed as to require removal; (*d*) temporary or permane of agricultural lands; (*e*) ponding and stagnation of drainage lines; (*f*) water-bearing strata, dissipating the ground-water and cutting off the local (*g*) accumulation of water in basin-like depressions, or change of stream resultant damage from floods. (See Sec 23, Art 16.)

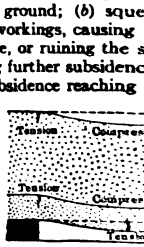

Fig 548. Tensi Edge of a

113. Conditions Affecting Subsidence

Depth of workings. A large working near the surface will cau unless the ground is protected. The depth at which the surface is varies with local conditions; it has ranged from 300 to 2 400 ft. (subsidence has occurred from workings at 340 to 924 ft; Richards siders that caved stopes below 1 000 ft will not affect the surface; of depth means increase in weight of strata, with greater liability t

Kind of excavation. Longwall mining produces subsidence and-pillar work, or narrow vertical stopes, would have no effect. clined excavation is less liable to cause extended subsidence than Other factors are the number, size, and position of supports. Fay

allest where the method of mining is such as
on of the roof.

s of its shape or position, may determine the
of subsidence. Over a thick coal seam, sub-
rface; over a thin seam, there may be only

es, based on mining a 13-ft seam at the Commentry

	Max height, zone of subsidence, ft
thout filling	2 628
th filling	1 050
filling	657
ng	327

vation, PROMPTLY SUPPORTED, is less liable to
, where unsupported strata are under strain
ing WITHOUT SUPPORTS is more apt to cause
e latter case, the overlying strata may adjust
ith little transmission of the movement; in
ure may result. Richardson (466) says col-
rapidly advanced, there will be little damage
t 1 200 ft, the effect is negligible.

ine working is usually surrounded by, or con-
t removed. This material is generally weaker
iable, soft, or yielding, the overlying load may
ge, resulting in subsidence.

occurs also in metal mines, as at Bisbee, Ariz, where
es of altered, putty-like rock, readily flowing under
y hard, oxidize on exposure to air and water, or
d lose their strength. This is true in sulphide de-
, or where the country or gangue rock is kaolinized
gration of walls due to oxidation of sulphides has
on (466) notes that pillars containing much pyrite
vatersrand, deteriorate by oxidation in 7 to 8 years,
ll of the Main Reef.

re most affected by subsidence, their physical
amount, height, and direction of subsidence.
ariable that no general rules can be laid down.

d, and of regular thickness, composition and hard-
pports. But, as such conditions are rare in rocks
the application of the principles of mechanics would
bsidence might be computed, but the presence of a
iate the result. In considering whether subsidence
le area, direction and intensity, the overlying rocks
ratigraphic column platted. CHARACTER OF ROCKS.
ncreases in volume more than plastic rock, an exca-
ubsidence is retarded; elastic rocks transmit pressure
k hard roof rock falls in large blocks, while soft lami-
. A thick hard layer, as a bed of compact sandstone,
bove the excavation, will check subsidence and help
vering of surface alluvium, subsidence spreads out
eath (Fig 549). In Westphalian mines, subsidence
ds above the coal; in the 1 200-ft bed of marl in the
lreich (465) observed vertical subsidence over an
sliding of the adjacent marl. JOINTS in the over-
d, in caving stopes, such ground will not follow its
nce started in highly jointed rocks may enlarge and

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continue to the surface. FAULTS AND FRACTURES, if numerous, break t mosaic of blocks, without cohesion; it is as though a girder in a buildi into pieces. Presence of gouge in the faults accelerates subsidence, for it cant; this effect is increased by tures are parallel in dip and stri subsidence will be elongated in t the breaks and its width deter width of the fractured zone. toward the excavation restrict their vicinity, but deflect it so t effects will not be vertically ab The caving along a single faul great, but parallel faults may a between them to move and affec as has occurred in the May Cons Rand. If adjacent faults dip so over the excavation, their effect if they form an inverted V, they

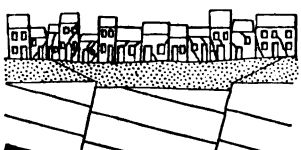


Fig 549. Influence of Faults and Alluvium on Subsidence

supported wedge, ready to drop. This is one of the most dangerous condit ence. In general, faults tend to produce sudden caves. CLEAVAGE (schistosi to occur in the wall rocks of ore deposits than in those of coal seams. If it cohesion is absent in two directions and very small in the third, due to lac of the individual plates. If cleavage is normal to the excavation, the effect on subsidence is much greater than if parallel. FOLDS both aid and hinder subsidence; in case of an anticline above an excavation, the strata, if comprising one or more competent members, may form a supporting arch for the rock above them. If it consist of soft rocks, folding is apt to shear and crush them, which may aid, rather than hinder, subsidence. A syncline over an excavation would aid subsidence, for its nose is apt to be fractured by the bending. BEDDING. Thick, bedded rocks are usually more capable of acting as a unit under stresses, and will support more load than thin beds; their presence would retard or prevent subsidence. Sedimentary rocks overlying ore deposits may be thick or thin bedded; those immediately over coal seams are usually thin be beds, usually thick. DIP OF STRATA overlying coal seams is parallel to th an integral part of the rock series; hence all subsidence breaks across the be discussed in Art 114. In ore deposits, as the bedding may lie at any angle to subsidence is more liable to extend farther vertically when it follows up the d has to break across strata. COMPRESSIVE STRENGTH of the rocks above an ex mines whether subsidence will begin, and if so when. Subsidence may not stone at a given depth beneath the surface, when it would occur in weaker Laboratory tests of rocks can not be safely used to calculate the load they they show the relative strengths. Bunting (468) says tests show that sands manent sets for the smallest loads, while granite and limestone are almost p The average moduli of rupture, in lb per sq in, from a number of tests, are: fi 2 700; slate, 7 736; sandstone, 806; granite, 1 681 (465). For tests on coa

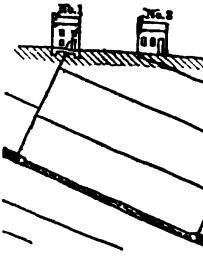


Fig 550. Surface Damag

Underlying rocks sustain the weight transmitted to them through pil excavations. Coal seams are often underlaid by clay which, due to pressu through a pillar, may be forced out into the working and allow the ground side (Table 81, Art 115).

Water-bearing strata above an excavation may be drained, and if shrink, causing surface subsidence. Clays, marls, and clayey sands are produce these results.

Surface alteration may so disintegrate the material overlying an exca that its strength is greatly reduced, and it might allow subsidence to extend

and Principles of Subsidence

are subject to two kinds of distortion due to the with-
oval of mineral: local caving in stopes, and a general
xtension of the local movement may produce general
ng forces are complicated. For the conditions influ-
Subsidence may sometimes be treated as a matter of
nall excavations, but formulas are not discussed here.
experience, comparison of observed facts, and experi-
ons. The observations and experiments of H. Fayol
urate data and reliable conclusions, and are generally
ary of conflicting opinions as to the nature and effects

ment: (*a*) movement is transmitted to the surface
b) the surface is not affected when the workings exceed

) subsidence extends to surface without sensible dim-
eaker as they extend upwards.
subsidence and mine working: (*a*) subsidence takes
limited to an area bounded by perimeter of the work-
; (*c*) subsidence can not be referred to the excavation,
ormal to the beds, but only by lines at an angle of 45°
ose of the ground, or by some similar angle.
of filling effectually protects the surface; (*b*) filling
:) subsidence is greater with filling than without it.

ement. Fayol (467) concluded that, in strati-
imited by a sort of dome, which has for its base
e amplitude of the movements diminishes away
Shape of dome depends upon position of the

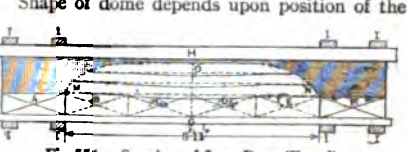
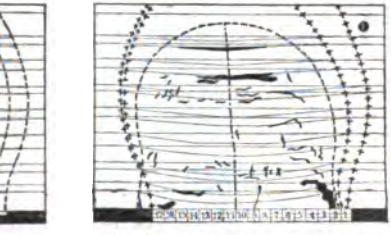


Fig 551. Sagging of Iron Bars (Fayol)

lay, sand, plaster, or paper; these materials were sup-
be removed, simulating removal of coal. The layers
pposite which lines were drawn on a glass front to the



Removal **Fig 553.** Dome of Subsidence from Removal of 3 Rows of Blocks (Fayol)

subsidence, and the deflection of the paper sheets was
In Fig 551, iron bars 5 mm thick, one above another,
was placed on top and, by stays *I* and bolts, the whol

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was secured to table *G*. Blocks *B* to *E* were removed over a length of 4 ft bars noted; the area of subsidence was a flat dome. In other experiment long, beds of earth, sand, clay, and plaster formed the layers. Wood b were numbered 1 to 17 (Fig 552). By removing them in order, excava

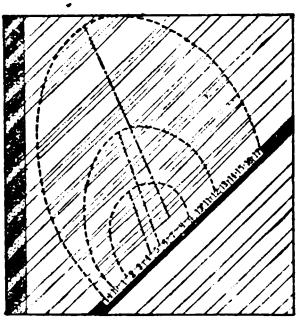


Fig 554. Dome of Subsidence in Inclined Strata, from Removal of 1 Row of Blocks (Fayol)

Fig 555. Dome of S clined Strata, from Rows of Blocks (Fa

and domes of subsidence resulted. In Fig 553, 3 layers of blocks ar of crosses show domes caused by removing the 2nd and 3rd layer, re slices of coal. Fig 554, 555 show experiments on inclined strata. strata, the axis of the dome approaches the horiz. Fig 556, 557 show

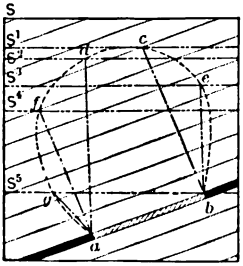


Fig 556. Dome of Subsidence in Inclined Strata, showing Angles of Lines Connecting Edges of Excavation and Surface Area for Different Surface Levels (Fayol)

the surface above the excavation dete or presence, and extent of, surface e to S_5.

Wachsmann stated (470) that when the first overlying strata collapse, t

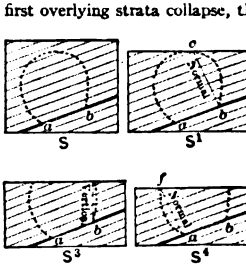


Fig 557. Extent of Surface Sub upon Depth of Excavat

and crack if the excavation is large enough, while the uppermost s cracking (Fig 558). Hausse (471) distinguishes a MAIN BREAK and (Fig 559); but this has not generally been noted; the dome theory ex ences more readily, and agrees with the observed facts.

Positions of surface subsidence and excavation. Gon and others held that the outer limits of the zone of subsidence strata normal to the bedding planes, formulating the LAW (Fig 560). Others say (465) that they will lie between the normal (Fig 561). and even at different angles.

nnect the limits of surface subsidence with the exca-
nd cut the strata at angles depending upon depth of
These lines may coincide with the vertical when the
S_4, or between normal and vertical at S_3; hence, the
depth, rather than of direction of fracture.

ce shown
orated by

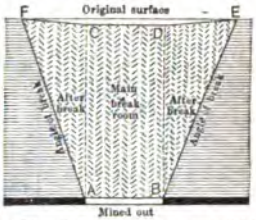
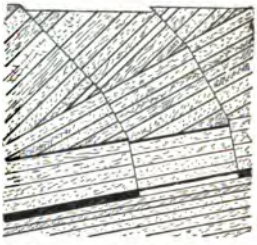


Fig 559. "Main Break" and "After Break" (Hausse)



ock and of ann)

Fig 560. Fracture Normal to Bedding; "Law of the Normal" (Callon)

on the Rand the surface is unaffected by caving
y be affected at smaller depths (466).

mines also indicate
65). The surface
ation at a depth of
al and may be due
dreich (465) and
sidence follows at
rs are left. Fayol
zone of subsidence,
the betls dip less
ite, does not exceed
(2) where the area
t twice the breadth
ick, up to 4 times
1 6 ft. The writer
e when excavations
e character of the
lts, dikes, or folds,

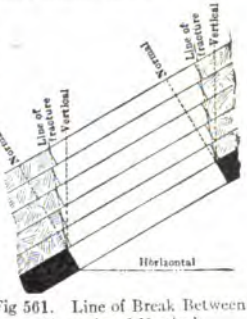


Fig 561. Line of Break Between Normal and Vertical

ement decreases with the depth of the exca-

s usually from 30 to 50% of the height of excavation.
llars or filling, or the increased volume due to break-

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ing of the roof, occupy part of the excavation; the breaking of some of the zone of subsidence helps to swell the volume; the elasticity of rocks

Table 76. Surface Subsidence

Authority	Location	Depth of excavation, ft	Surface subsidence, ft	Thickness of excav'n, ft	Surface subsidence, %
Fayol.....	Commentry Mines	321* (Filled)	.....	....	20
"	"	189 (Filled)	.....	....	10
Dixon.....	Bent Coll'y	650 (Not filled)	3.76	5.5	68
Snow.....	So Kirkby Coll'y	2 108	3.47	4.75	73
Fayol.....	Montrambert	.....	.....	6.8	76
"	"	.350†	.....	....	25–30
"	"	57	3.25	4.25	....
Hay.....	Shirebrook Coll'y	1 500–1 700	1.3–1.7	5.0	30
Kay.....	.....	360	.....	5.0	70
"	.....	990	.....	3.5	64
Mengel.....	Saxony	600–2 400	.....	....	50
.....	Bully Greenay	.....	.....	29.36	46
.....	France	432	.....	44.4	30
O'Donahue	.....	600	.....	4.0	75
"	.....	2 400	.....	4.0	25
Fayol.....	France	(Filled)	.....	7.5	19
"	"	2 600 (Not filled)	0	13	0
"	"	1 040 (Not filled)	0	13	0
Gresley....	England	390	.....	70	40

* Dip of beds, 34°. † Dip of beds, 35°.

face coincides with the top of the dome (Fig 557) its area will be smaller th area; a few recorded instances are in agreement with this. The area also on the character of rock; soft strata or alluvium at the surface increas rocks will not. Over long, moderately wide stopes in metal mines the su trough shaped, somewhat wider than the excavation.

Advancing subsidence. In many instances movement oc and above, a working face; hence, surface structures must be p the excavation advances under them.

Fayol observed that, in the Commentry mines (367), movement fir certain horiz distance ahead of the working face, and that this distance constant. Dixon (472) showed that a wave of disturbance preceded by 65 ft, but the max subsidence followed it regularly at an aver distance of the face was 650 ft, dip of bed, 3°. Levelings by Snow (474) showed pressed a rapidly advancing longwall face by 433 ft; total subsidenc In back the face. At Shirebrook colliery, surface cracks appeared above t 5 ft seam at 1 500 ft depth. Richardson found that with workings at 3 Kleinfontein mine, Rand, the surface was affected 25 ft back from the o on either side of the excavation; over the Treasury-Jumpers mine it wa north of the excavation, 100 ft east, and 200 ft west.

Time. Any data indicating (a) duration and period of gre of subsidence, and (b) time it takes to extend to the surface, w Scattered observations show that (a) varies from 1 to 50 years, weeks to several years.

Control of Subsidence

total expansion of the subsiding area equals the excavation be filled wholly or in part by natural ce may be controlled or prevented. If the con- steps may be taken underground and on surface.

localized chiefly in a caving upwards, continuing until s the overlying strata. Artificial support is necessary n the excavation and the top of the arch. Timbering eing to PREVENT THE STARTING OF CAVING.

rous deposits, which usually have considerable vertical alt, subsidence is more extensive and timbering is in- : pillars, cribs, and filling (Art 59, 93, 110). In coal y of artificial support depends on the method of min- ompared to cost of filling, and value of the surface, or surface is not owned by the mine.

Stoek (465) summarize rules from different author- s in flat coal seams, as follows: MERIVALE. S = of side of pillar, yd; D = depth of shaft, fathoms.

50-ft shaft is yd per 75 ft aw a line en- ave shaft-pil- l coal beyond . WARDLE. creasing with um be 120 ft q ft for every GHES. Allow oth; hence, a llar of 300 ft t, make pillar 1 ft for every adius of pillar ft, t = thick- EERING (Lon- w a minimum

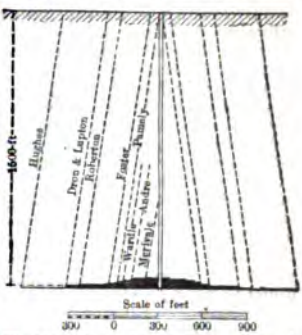

Fig 562. Sizes of Shaft Pillars, According to Different Formulas (Knox)

ts, $R = 60 + (D \div 10)\sqrt{t \div 3}$, where R = radius ness of seam, all in ft. ROBERTON, E. H. R = + 6) + 2 Dt, where R = radius of pillar, D = depth haft, t = thickness of seam, all in ft. Central Coal in rule, Ill. Allow 100 sq ft of pillar for each ft lepth; if bottom is soft, increase by half; if coal is ker than 5 to 6 ft, the pillar should be larger.

The diversity of opinion regarding shaft-pillars is shown phically in Fig 562. Fig 563 shows the effect of a shaft- ar (467); depth of seam is 453 ft, thickness 60 ft, and on des of the pillar 25 ft of the thickness has been removed. face was affected to points C, D, and horiz masonry at D' became curved. To protect surface structures, Scotch ineers leave pillars ½ to ⅔ larger than the floor plan of Connellsville, Pa, for depths of 150 to 300 ft, a margin of a building. If the tract is large, 50 to 60% of the coal nder being left in pillars.

positions of surface structure and pillar are shown in

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Fig 564 (474). Richardson (466) suggests "construction of a cross-sectio surface object and underlying orebodies; from each end of the object dra

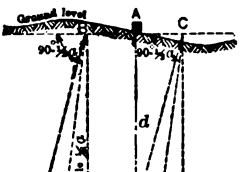

Fig 564. Position of Pillar with Respect to Surface Object (O'Donahue)

the reefs toward the rise side and making horiz plane equal to the angle of fracture inclosed within these lines is the pillar re angle of fracture is taken as halfway bet cal and normal to the planes of stratifica

Effectiveness of filling. It redu vertical movement, but will not prev nary filling is compressible (see test never fills excavations completely. W settles, the resistance of filling, wea creases rapidly, and soon stops the m

Belgian engineers believed the most c gives no guarantee against damage to sur it only lessens the sinking (467). In th mines, careful filling greatly reduces verti but has no great effect upon its lateral ex French engineer says: "In working by a permits the roof to fall, the movement of strata gradually diminishes and tain level; with filled longwall, it is almost independent of the depth; le pillars can alone insure safety of the surface" (467).

Fayol's observations at Commentry mines (467) are: (*a*) Certain sea ft thick, difficult to keep open without filling, have hardly required timb time they were filled. (*b*) For soft rock, formerly used, a hard incon was substituted; the haulageways at once became steady, the roof sett less, and cost of timbering decreased. (*c*) Some filling was done imperf slabbed off and required much timber. Filling was then done carefully; n left. Soon the pressure was hardly felt at the face, and all the timbers c Making reasonable assumptions, Fayol showed by calculation that in the s in a filled excavation 39 in high, subsidence would reach a height of 65 sank without breaking; if the roof broke, it would reach a height of 541 f would be reduced to 262 ft. See also the case stated under Size of excav

Summary of opinions: (*a*) filling greatly diminishes subsidence; (*b*) it vent it; (*c*) minimizing open spaces by careful packing of filling is impor figures are based on mine gob which undergoes fairly high compression; sa materials compress but little (see tests below), but their use is costly and n ticable. In narrow excavations filling limits, and usually prevents, subs

Effectiveness of pillars. In general, pillars are the only m tecting shafts, important workings, or surface objects of value. and English coal fields, they are found to prevent surface dam 50% of the coal is left.

Fayol stated respecting room work that, if enough pillars be left, the affected; the mesh of the network of pillars and working places should be workings are shallower. As depth increases, the workings can be enlarged in proportion to the area of pillars, provided the different zones of subsidence are kept distinct from one another. The structure and character of the rocks may modify the above statements, but, in properly spaced and proportioned workings, the disturbance may be limited to the volume within the 565, where the odd numbers represent rooms with their domes of subsid If pillar 2 be removed, these affect a larger area. Also, a small pillar may protect a surface object, because the zones of subsidence on either side overla

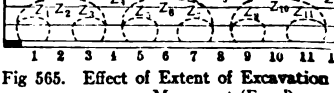

Fig 565. Effect of Extent of Excavation Movement (Fayol)

workings at depth of 20 to 40 ft, an outline of each pillar
ardson states that, on the Rand, when a pillar has been
side often causes more damage than if no pillar existed.
the only effective prevention of subsidence for coal
ally sufficient. Timbering serves in small stopes of
ds the force exerted by the portion of the roof below
d that in coal seams timbering prevents the over-
ding before general subsidence occurs; hence it may
urface effects. Properly spaced pillars of sufficient

ble 77. **Safe Unit Stresses for Stone** (Douglas)

	Crushing, lb per sq in	Shear, lb per sq in	Tension, lb per sq in
ranite.....	1 020	200	150
mestone.....	800	150	125
ndstone.....	700	150	75

d friability; laboratory tests do not apply to

s for
ta

| pillar
hing
of |
|---|
| 000 lb |
| 8.1 |
| 6.3 |
| 4.5 |
| 2.6 |
| 0.7 |
| 8.9 |
| 7.0 |
| 5.2 |
| 3.3 |
| 1.4 |
| 9.6 |
| 7.7 |

Empirical formulas are necessary, aided by an approx estimate of strengths of the rocks based on tests. For moduli of rupture, see Art 113. An aver of numerous tests of the Rand quartzite gives a mean ultimate crushing strength of 7 521 lb per sq in (466). Since tested cubes are weaker than the same area in a wide bed, Richardson thinks 10 000 lb is a probable figure; or, at great depths, even 15 000. Taking the sp gr of quartzite as 2.83, the above figures give 8 183 and 12 275 ft, respectively, as the depths at which quartzite will fail under its own weight (Table 78); for inclined strata, multiply the figures in the table by the cosine of the dip. The ultimate strength of trap rock forming pillars in the Lake Superior copper mines is 16 666 lb per sq in (465). Tests of building stone (475) have little value in subsidence; rocks of the coal measures are unsuitable for building. Moisture lowers the crushing strength of sandstone
shing strength of Illinois coal gave an aver maximum
ittsburg seam, 3 155 lb (465). An average of 45 tests
different sized
ith pressure ap-
to the bedding,
b per sq in (see
inous coal gave
e bearing power
ngth of the clay

g is greater for
Though sands
hey are almost

Table 79. Crushing Tests, North Carolina Sandstones (475)

Absorption, %	Conditions	Crushing strength, lb per sq in
4.2	Dry	10 736
4.2	Wet	6 399
3.71	Dry	11 741
3.71	Wet	6 174

point of the grains. Voids of broken rock are
in size; as pieces are in contact in few places,
er and allow considerable compression.

a pressure less than the crushing strength of a single
be filled with sand, its compressibility is lessened.

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Mixtures of large pieces of sandstone and shale, commonly used for filling in have about 40% voids. Fayol states that in workings from 300 to 900 f filling is compressed 30%, leaving a volume about 12% larger than the orig in place (467). The Co vestigating the Frank sl concluded that under av the settlement would be thickness of bed, if ordina used; an inappreciable a granulated slag; 10 to 15 sandy clay, and ashes; with dry packing (476).

Table 80. Crushing Tests, Pennsylvania Anthracite Coal

Samples from Anthracite Fields	Height of sample, in	Aver per sq in		Number of tests
		First crack	Max load	
Northern	1	3 022	6 241	122
	2	2 025	4 087	116
	4	1 875	2 854	113
Eastern Middle	1	4 996	7 417	7
	2	3 343	3 857	6
	4	3 413	3 821	7
Western Middle	1	3 001	8 631	3
	2	788	3 499	3
	4	1 440	2 447	3
Southern	1	1 124	3 814	12
	2	1 099	2 377	12
	4	988	1 809	12

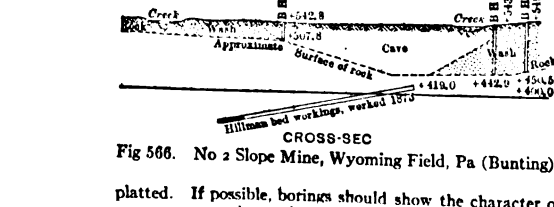
Water should be k partly subsided area loosen sands or clays surface movement, re further underground st

Workings extending u strata or creeks may c ward movement of water loosen the rock and start Streams flowing over ground should be diverted, to prevent inrush of water and sands in event of With such precautionary measures, the accident in No 2 slope, Wyoming would have been averted (Fig 566) (468). Water in mine workings should to drains, to prevent undermining or disintegration of pillars; if allowed to run through filled areas, it carries away the finer materials and reduces the volume of fill. In salt mines, the water menace is more serious. DISINTEGRATION OF MINE PILLARS due to oxidation, decomposition, slacking, or solution, can be prevented by covering them with concrete or plaster.

Table 81. Bearing Power of Rocks, Clay s

Safe bearing power, tons per sq ft, for different materials	M
Rock (hardest), thick layers native bed	200
Rock equal to best ashlar masonry	25
Rock equal to best brick masonry	15
Rock equal to poor brick masonry	5
Clay in thick beds, always dry	6
Clay in thick beds, moderately dry	4
Clay in soft beds	1
Gravel and coarse sand, well cemented	8
Sand, dry, compact, well cemented	4
Sand, clean, dry	2
Quicksand, alluvial soils	0

Protection against effects of subsidence is essential. Resulting AIR BLASTS may be subdued by bulkheads or filling, and important workings protected from shock by pillars; loose ground should be bered; all underground machinery should be in protected areas. In shallow GROUND SHOULD BE advance of minin from surface will sudden drops in be holes, or old stream In subaqueous mi holes should be ke vance, to detect wa nels. Levels take bottom of the w thickness of mud workings should be



CROSS-SEC

Fig 566. No 2 Slope Mine, Wyoming Field, Pa (Bunting)

platted. If possible, borings should show the character of the subaqueous str SURFACE AREA where subsidence might occur should be outlined in advance of the All inhabitants should be moved, machinery or other valuable objects transfer

ng Material

umes *	
Grains 15 to 20 mm (.59 to .787 in)	Mixed grains and dust
225	216
224	229
310	214
223	202

ed on unbroken volume of 100.

Table 83. Compressibility of Crushed Materials (Fayol)

Kind of rock	Rocks previously crushed, occupy space indicated, under pressure *			
	Press 1 422 lb per sq in	Press 2 844 lb per sq in	Press 7 110 lb per sq in	Press 14 220 lb per sq in
Clay	100	90	75	70
Shale	128	116	110	97
S'dstone	136	125	120	105
Coal	130	125	118	109

pporting Strength of Dry Filling (477)

tificial supports	Approx depth, ft, of column of coal-measure rock, 1 ft sq, necessary to compress artificial support					
pression	1	3	5	10	20	30
ary construction	....	10	12	36	125	*306
well constructed	....	46	75	146	292	*512
aver construction	....	8	68	182	270	*419
one through 1¾-in						
	....	....	20	53	124	*298
, filled with sand	....	....	21	53	186	*465
nd rock, 45% voids	....	48	66	121	351	*492
sandstone, 50% voids	12	27	45	117	434	0615
dstone, 40% voids	....	44	74	177	619	1 310
adstone, 40% voids						
	....	46	77	325	6 000	68 860
hes, 64% voids	....	13	25	70	143	332
nd	12	40	70	442	1 715	6 640
ushed in with water	111	522	891	2 310	....	c8 860
flushed in	32	118	190	472	1 822	5 905
sand and gravel; 5						
	117	1 092	(c)	....	....	....
	1.0	1.0	1.0	1	1	†
	3.5	4.4	4.7	5	4	†
	3.6	9.0	(d)	(d)	(d)	†(d)

5% settlement. (c) 20¾% settlement. (d) Worthless, s under continuous load equal to 600 ft of rock. * Free rative.

ransmission, and pipe lines laid in more secure places. A ıtcrops, thickness of soils and faults, fracture zones, and o. No inhabited buildings nor objects of value should be nes for some distance from the prescribed area of subsidence long the strike, lying entirely without the area, but dipping ly (see house No 2, Fig 550); the hanging-wall side is more If the fault crosses the area, the foot wall is more dangerous. well packed. SPECIAL CONSTRUCTION should be used for over an area where subsidence has taken place or may occur. s should determine extent and location of workings, gobbed s should go to bedrock; concrete columns have been used ould be adapted from the type prescribed for earthquake

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regions. Existing buildings may be reinforced, so that if they tilt they wi
If pipe lines must cross the area, lead joints may be used with advantage.

Agricultural lands can be restored when damaged. Holes and cracks n
and surfaced with soil. Depressions may be drained if sufficient grade
otherwise they cannot be used unless filled, for storm water would keep the

PLACER MINING METHODS

Introductory. Classification: (*a*) Surface methods which are
of open-cut work (Art 95–101); and of dredging (Art 126). These a
important and varied, since most placers occur at or near the surfac
derground methods for deep or buried placers (Art 116), and kno
Mining or Drifting (Art 127).

Placer mining includes the work of excavating and transporting placer
of recovering their contained gold or other valuable mineral, which is usu
concentrating or "washing" the gravel in running water. All the common
excavating and handling earth and rock (Sec 3, 5) are utilized where suitabl
placer mining, as sometimes used, excludes hydraulicking (Art 122), dredgi
mining; general term "alluvial mining" then includes all methods of mining
Exploration or prospecting of placers, to determine their value and yard
exploitation, is done by drill holes or shafts (Sec 25).

116. Placer Deposits

Definition. Placers are deposits of sand, gravel, or other alluvium, con
ticles of valuable minerals in workable amounts. Native gold is the mos
placer mineral; platinum and cassiterite (stream tin) also occur in gravels
sionally, native Ag, Bi and Cu, amalgam, palladium, cinnabar, monazite, di
other gems; some phosphate deposits (Art 99) may be classed as placers.
"gravel deposit," "gold-bearing gravel," and "alluvial deposit" are used lo
of "placer" or "placer deposit." The following paragraphs deal chiefl
placers; their bearing on other placers is obvious.

Geology. "Three conditions operate to form placers: (*a*) Oc
gold in bedrock to which erosion has had access; (*b*) separation o
bedrock by weathering or abrasion; (*c*) transport, sorting, and de
auriferous material derived from erosion. Erosion, while operati
cases, is not absolutely essential, as residual placers may be form
weathering in place of auriferous bedrock" (377). The primary
PLACER GOLD is almost always in auriferous veins, stringers or othe
These primary deposits were not necessarily rich; they may have be
eroded, or their remnants may not be workable.

Gravel deposits are often concentrations of enormous volumes of rock, in
may have existed in stringers too small to mine. Thus, in the Klondike,
bodies have been found in connection with the placers; the prevailing cou
schist, containing numerous unworkable quartz stringers sometimes showin
produce the present placers, it is estimated that 136 cu miles of rock, averag
2¢ in gold per ton, have been eroded and concentrated. Some gold may
from solution in residual placers; this agency is unimportant in the depositi
formation of nuggets in stream gravels (307, 378).

Weathering and erosion. Deep secular decay of gold-bearing rocks
formation of most important placers; it can occur only in base-leveled regi
graphic maturity. Under such conditions, rocks break down into clay and fi
which are removed by wind and slow-moving water. This effects surface c
of heavy and resistant minerals, and sometimes forms commercially import
deposits, as the aeolian placers of Australia. A subsequent uplift which
the streams will cause rapid removal of the residual mantle and further conc
its heavy minerals along water courses. Placers formed by the concentr
of running water are called SORTED PLACERS. Rapid erosion of fresh ro

placers. A geological history involving several cycles
he formation of rich placers, as deposits of one period
of a later cycle. Many rich placers in Alaska, Cali-
ed; they are called RESORTED PLACERS (377, 379, 307).

and resorted placers is irregular; normally, but not
ted on bedrock or within a few ft of it; coarse gold is
wer 20 ft of a deposit, but, except in minute quanti-
ormly through a great thickness of gravel. In small
whole width of the stream bottom; in larger streams,
ts on bedrock, but most of it is usually concentrated
s the "paystreak," "pay-lead," "channel," "run of
ustralia). Paystreaks often follow devious courses,
at of the present stream; Fig 567 shows a case in
ut off suddenly; they may or may not occupy the
ey may form at any eleva-
stratum of clay or other
a "false bedrock." Some
hus formed and overlying
tions of fine gold occur in
is checked (Sec 2, Art 21).
vanced to explain the for-
reaks. J. B. Tyrell holds
y the bottoms of the origi-
early period of stream ac-
ecular decay and is quickly
gold concentrated in such
ited in the bottom layer of
ream. As the stream wid-
would be covered. A later
a lower elevation; depend-

Fig 567

t might descend into it or remain in its original position
Smyth points out the fact that the whole contents of
allow concentration of heavy minerals; such motion
vels are shallow and floods occur. Lindgren ascribes
(*a*) partly to a natural "jig-like" movement in mod-
during long conditions of fair balance between loading
stream, (*b*) partly to a slow forward and downward
n gravels, which would allow heavy minerals to work
hat heavier gold particles entering a stream bed from
d out into gravel flats of streams of gentle grade (the
d plains), but settle on the marginal bedrock of the
lens, it covers the accessions of gold along its margin
esting on bedrock and lying under barren or lean allu-
s no essential disagreement between these statements;
at different times.

old; glaciers dissipate the gold which they pick up
ce glacial gravels have little economic value unless
ost-glacial streams or have derived their gold from

courses varies between wide limits. Lindgren (307)
ditions for concentration of gold exist in streams of
. If sediment is deposited in an overloaded stream,
s; conditions for formation of rich placers are also
rapid, unless the gold supply is unusually abundant.
vada have grades of 50 to 100 ft per mile; many in
t per mile. Depressed or elevated ancient river beds
nal grades much modified.

cal geology, regional geological history, and physio-
aid in placer mining; it furnishes an hypothesis on
ation, and mining. Further data on the geology of

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placers are given below and in Art 117; see also Bib (307, 377, 378, 379, ; Geol Surv Bull 328, 337, 410, 498, 533, 534, and 592.

Placers may be classified according to their origin into resi and resorted placers, as above. From the mining standpoint th grouping based largely on form is more useful: (*a*) RESIDUAL PLAC when formed directly over outcrops or on gentle slopes below them times called "eluvial" placers. Residual placers are relatively u their valuable particles are not rounded by abrasion; concentrati is superficial, the richest part being at or close to the surface (379). SIDE PLACERS occur on valley slopes; are not in well defined chann somewhat sorted by water; a transitional type between residual gravels. (*c*) GULCH OR CREEK PLACERS are gravel deposits in, a and at the level of small streams (Fig 568); they are usually shall placer sometimes denotes gravel in gulches which are dry or carry i

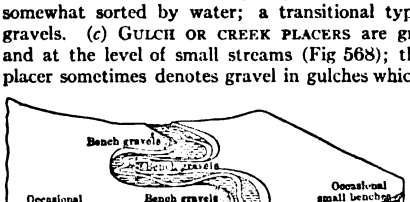

Fig 568. Placer Deposits (U S Geol Surv)

streams. (*d*) BEN (terrace gravels) ar of old stream grave moved by subsequ action which has cu to bedrock. They on the flanks of pre (Fig 568), or as rer previous drainage course of which ha relation to existin

The White Channel, in the Klondike, is of the latter type. (*e*) RIVER-B occur in bars and gravel flats adjacent to large streams of gentle grad surface often lies a few ft above normal water level. Fine gold is d existing streams in bars at points where the velocity of the water is chec Art 21). (*f*) GRAVEL-PLAIN PLACERS are formed in flood plains and term is not strictly defined. The wide valley at Oroville, Cal, formed b ing of the Feather river, is a gravel plain; also some of the tundra pla coastal plain at Nome, Alaska (383). (*g*) BEACH PLACERS (marine p formed by the concentrating action of waves on a sloping beach.

Gold in beach placers may come from a sea bluff of auriferous gravel, broke concentrated by the surf; some beaches are probably enriched by gold brough by nearby streams. Nome beach, a famous example, is 50 to 75 yd wide, slopes abuts an escarpment of gravel, muck and tundra 10 to 20 ft high (Fig 569); workable placers occupied 20 miles of beach. Fine gold is concentrated near the bottom of lenticular masses of garnet and magnetite sand, which are 5 to 9 ft thick and normally rest on a clay false bedrock. The lenses are rarely more than 100 ft long, and are not uniformly enriched over the width of the beach. The richest pay was usually in lenses 5 to 6 ft wide by 2 to 6 in thick

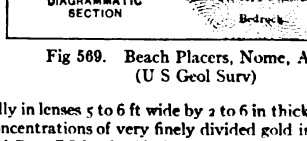

Fig 569. Beach Placers, Nome, A (U S Geol Surv)

Beach placers, consisting of surface concentrations of very finely divided gold i and garnet sands, occur on Kodiak and Popoff Islands, Alaska, on the coast of elsewhere. They are constantly shifting; in places, beaches are reworked annu storms. Beach deposits must usually be mined by simple hand methods (Art 11 disastrous attempts to work them on a larger scale failed to recognise that whil thin layers may be rich, the aver value of any considerable depth of sand i Wave action sorts beach material into equal-falling grains (Sec 28, Art 17); wh

magnetite and garnet sand, it is difficult or impossible
ning appliances (382).

ers" or "deep leads," Australia) exist in many
osits may be a buried placer, for example:

ading of streams, may deeply bury the accumulated
In the Fairbanks district, Alaska, deep gravels, laid
water-courses which occupied the present valleys, are
uvium. Bedrock of the old streams was a little steeper
ttoms. Most of these channels are centrally located
of the present valleys; present streams occupy asym-
ch they follow closely. The gravels are covered with
humus, fine sand, clay and silt), derived partly from
part by sluggish streams. Near the headwaters of
with those of the present streams (383).
the Sierra Nevada, Cal, were covered by deposits of
reccias in places 1 500 ft thick. The region was then
eroded the present canyons, some of which are 2 000
rest as more or less connected remnants on the sum-
s (307). Near Nome, 7 buried beach placers have
indicating periods of subsidence or elevation which
e. The richest is the Third Beach, lying 20 to 124 ft
present beach, which it roughly parallels; it has been
s like those of the present beach (384).

to auriferous alluvials of various kinds, occurring in
for "dry-washers," see Bib (452, 409).

teristics of Placer Gravels

ne sand to boulders of several tons. Numerous
nethods like dredging (Art 126); they increase
) or other methods, as they must be broken by
icks. Size of gravel affects the amount that
with a given volume of water, and hence the
tal. In general, the duty of water (Art 122) is
may be reduced by very fine gravel which packs

us meanings in placer mining; thus, gravel containing
for dredging and large for shoveling-in (Art 120).
and water-worn, sub-angular, or angular. The duty
es with the angularity of the stones.

nting material. Some gravels consist entirely
and; others, of stones embedded in a clay matrix.
o solidified by chemical action as to cement the
TED GRAVELS.

excavated and washed, but it may be difficult to hold
material. Deposits containing much sand are apt to
yers are usually barren. Stiff clay always increases
reduces output, increases repairs, and prevents free
break up readily in water, others ("sluice robbers")
es and pick up gold or amalgam. Cemented gravel
to a greater degree; it prohibits dredging if much is
blasting prior to washing. A few California drift mines
stamp mills.

bility of different methods of mining; see individual

old is retained on soft creviced surfaces, where
ooth surfaces. In schists and slates (especially

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if they strike parallel to a stream course), clay, clayey sandstone gold may work down several feet; it also settles into minute crevi rock; it has been found in solution cavities in limestone to a depth of

In a new locality, the character of bedrock may be inferred from surroun evidence from shafts or boreholes is more reliable than that from exposures in in which decomposed rock would be eroded. Large boulders of adjacent co the gravel usually indicate hard bedrock. Bedrocks of schist, granite, an stone and porphyry, are usually decomposed to a depth sufficient to all In one South American hydraulic mine, a granite bedrock is rotted to a 10 ft; though barren, this material must be washed to recover the gold lyi limestone and slate formations, bedrock is almost always hard and block deep crevices. Creviced bedrock increases costs, as it must be taken up t contained gold. In hydraulic mining, hard bedrock increases cost of bed Bedrock of regular contour is desirable for all placer methods (385).

Character of gold. Placer gold is of all sizes, from large nugget specks called "colors." The size of colors is indicated by the num 1¢ worth of gold; some gold in the Snake River, Idaho, runs 2 000 Gold usually occurs in flattened and rounded grains; angular piec tallized gold, occasionally found, indicate nearness to the primary d

Fine gold is sometimes in thin scales, difficult to save; scaly gold and other forms may be coated with a film of SiO_2, MnO_2 or limonite, which hir mation. Fragments of quartz often adhere to gold, or form part of nugg commonly associated with particles of magnetite and ilmenite ("black sa (ruby sand), zircon and other heavy rock minerals (307). If present in la black sand may pack in riffles of sluices (Art 124) and interfere with gold

Lead in form of shot occurs in placer districts where much hunting has be caught by gold-saving devices and is separated at the time of clean-up; pays to melt such metal and sell it as base bullion.

Placer gold varies in fineness from 500 to 999; it is always alloyed sometimes copper is present; it is usually purer than that in veins district; its purity increases with the distance transported and w creasing size of grains.

Distribution of values may be the determining factor in c mining method. For example, all the gold in a deposit 100 ft d concentrated on bedrock in a 4 or 5-ft paystreak. A high-cost drifting (Art 127) may give more profit than cheaper open-cut wor the handling of 95 ft of barren material.

Frozen gravels of Alaska and Siberia must often be thawed by artificial they can be worked (Art 126, 127).

Buried timber occurs in many placers in the tropics; it is a serious hind may prohibit dredging. It may be piped out of the bank in hydraulic m into pieces for hoisting by derricks. Standing timber increases cost of o by the cost of clearing, which, per cu yd, varies inversely with the depth of

Swell of gravel upon excavation is usually 20 to 30%, up to 50 pact clayey gravel.

Swell may vary in different parts of same deposit; its determination is all sampling work, to avoid errors in estimating values. Several excavatic made and the resulting gravel measured loose in a wooden box holding 1 or percentage of swell is computed from the volume of excavation.

Values in gravel are expressed in cents or dollars per cu yd, or sq yd of bedrock. Drift mines often report values per ton.

Value per sq ft is the most convenient basis in placers where most of t thin paystreaks. If 2 or more paystreaks overlie each other, the total val is obtained by adding the values per sq ft of the separate paystreaks; th than estimates per cu yd. Value per sq ft × depth in ft ÷ 27 = value per

, Rocker, and Long Tom

ısed by prospectors in searching for placers, by miners e, and by engineers for recovering gold from samples

a circular dish with sloping sides; top diam, 10, to 2⅜ in; side slopes, 35° to 40°. Fig 570 shows erican pan, which a it is larger.

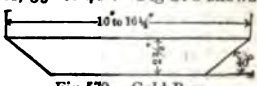

Fig 570. Gold Pan

ugh to stand rough oth, bright, and free cared for, pans of nents, and are cheap. Agateware does not rust, but ight, do not rust, but lack stiffness. Pans of copper, or re sometimes used for fine gold which will amalgamate; ated with mercury.

an of gravel is placed in water, the gravel thoroughly ak up lumps of clay, and the larger stones are picked is then given a shaking or gyratory motion, which face and allows heavy particles to settle; at intervals material washed off. These processes alternate until sand is left, which is dried and the gold separated by ck sand), or mercury is added to collect the gold as here is little or no loss of gold (Sec 31, Art 5).

back-breaking work; as the only tools required are a poor-man's method and is a common temporary expe- indispensable for testing gravel when prospecting and oncentrates in large-scale sampling and mining opera-

. An experienced man can pan carefully about hr; cemented gravel or sticky clay reduces this e percentage of coarse gravel, which need not be handles more than 1 cu yd aver gravel per 10 hr.

d of gravel varies with the swell (Art 117) and e amount put into the pan.

ne value of gravel is computed from the number of d recovered; a small error in the number of pans per n the estimate of value. The content of the ordinary stum of a cone, is 321 cu in. If the swell of the gravel + 1.20 = 267 cu in of gravel in place; hence there are cu ft, or 176 pans in 1 cu yd of gravel in place. If each there will be only 133 pans per cu yd. The number yd in different districts is usually between 130 and 150. compute values on the basis of 189 pans per cu yd; a No 2 round-point shovel is considered a panful (386). aluelessunless made by a skilled panner, who is also cal conditions.

wood or iron; its diam varies locally from 16 to 30 in. nerica (and in southwest U S for dry concentration of re employed by natives of India, Sumatra, Nigeria, etc.

ec 25, shows one design. Gravel is placed on ver it from a dipper, and at the same time the otion. The water and undersize of the screen ttom of the rocker, and are discharged over the emaining on the screen are washed clean, they s repeated. Most of the gold is retained on the

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apron; more is caught on the bottom, which may have cross ups are made at intervals, depending on richness and character

For saving fine gold, the material on the floor of the rocker must be ke and spread evenly; clayey gravel, or that containing much fine black behind riffles and tail-piece and requires frequent clean-ups. Before material back of the tail-piece is removed, dumped into the screen a apron is then lifted out and its contents washed into a pan for final conc men are sometimes expert enough to use the apron itself as a vanning pling work, the rocker is thoroughly cleaned-up after each sample has the tailing must often be re-rocked to save fine gold. On large-scale s gravel, it pays to break up the clay in a puddling box ahead of the being shoveled directly onto the screen. Wooden or galvanized iron pa for carrying samples to rockers.

Construction of rockers. Design and proportions vary wi

Table 85. Dimensions of Rockers in Different Regi

Example No	Length of bottom, ft	Width of bottom, in	Height of tail-piece, in (*a*)	Screen	
				Size, in	Diam of holes, in
1	5	14	1.5	18 by 18.25	0.25
2	5	18	1.5	24 by 21	0.25
3	6	19 (*c*) to 15 (*d*)	.....	.....	0.37 to 0.5
4	4	18	0.75	16 by 16	0.5
5	4	16 (*c*) to 15 (*d*)	1.75	20 by 21	0.5
6	4	15.5	0.75	23.5 by 15 5	0.37
7	4	14	.....	13 by 13	.....
8	4	20	0.75 to 1	12 by 24	0.25

(*a*) Height at center. Tail-piece is called "lower end-piece" in Fi Any desired slope is obtained by blocking up the frame on which the roc upper end of rocker. (*d*) At tail-piece.

Example 1. Rocker shown by Fig 5, Sec 25; recommended by Knox for sampling, with the comment that most rockers are too high and sh poor gold savers. Ex 2. A Cal rocker, found by author to be heavy a piece unnecessarily high. Ex 3. A large rocker used by miners (388 W. H. Storms (387). Ex 5. A very satisfactory rocker, used by the a work. Ex 6. From D. Waterman, *Min & Sci Pr*, Feb 20, 1909. S. O. Andros (389). A galvanized iron chute is used instead of an ap is entirely by cross riffles on the bottom. Ex 8. A Cal rocker; screen the upper part of apron, which is 18 in long; this forces all undersize pass over the apron; reduction in screen area is compensated by the un

Rockers should have tight joints, with corners strengthened by galva angles, placed outside; 2 or 3 light tie-bolts (Fig 5, Sec 25) aid in prev or swelling. Rockers are sometimes built to knock-down for easy Bottom board should be in 1 piece, free from knots and cracks, and will not "rough up" when scraped in cleaning-up. If good lumber i the floor may be covered with canvas fastened by quarter-round strips t Canvas makes an excellent surface for saving fine gold. Rocker bott ered with blanket, which is taken up at intervals and washed in a tub; fine gold savers, but are undesirable in sampling work because of the dela Aprons are of canvas, or rubber sheeting with canvas backing; their stretched tight; a sag is left at the bottom. High rockers are sometime 3 aprons inclined in opposite directions and superposed, so that the taili onto the upper end of the next; 1 apron is sufficient and more convenient Some Chinese rockers are apronless; to save the fine gold they require Screens are of galvanized or black sheet iron with round holes; 18 gage iro for small gravel; 10 gage better for coarse; Table 85 gives usual size holes, which are slightly countersunk on under side to prevent clogging excellent screen construction. Screen is picked up and dumped by the

ened to the screen box. Very heavy rockers are undesir- and, like those having a large roll, are difficult to rock

full 0.75 in, is ers, if not too orter ones for

usted to char- causes pack- le is required arrying much

found that a ater per day, n dry gravel; in loose sand gging settling e rocker, and h; this is sat- ain too much cleaning-up.

ay be oper- ; they spell lling gravel the duty of 5 cu yd of hr (390). Duty on sampling work varies widely rrangements for feeding gravel and water.

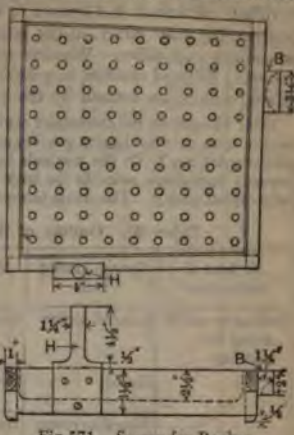
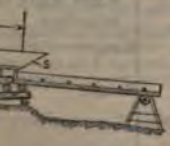


Fig 571. Screen for Rocker

rocker is a prospector's tool, or is used prior to larger- ll rich deposits in regions of scant water supply. It n prospecting and examination.



Long Tom is an open box *L*, Fig 572, having at its lower end an inclined screen *S*, punched usually with 0.5-in holes. Dimensions vary; Fig 572 represents early California practice (391).

after Bowie)

Running water is carried to the head end by a small flume *F*. Gravel is shoveled into the Tom or into the flume *F*; it is shoveled over in the Tom and large stones forked out; the fines are worked through the screen, and with the water fall t 123), set on a flatter grade than the Tom. The gold without aid of mercury. Capacity depends largely on shoveled into it; Wilson says that 2 men (1 shoveling an wash 6 cu yd of ordinary gravel, or 3 to 4 cu yd of At times the Tom is operated by 4 men; 2 shoveling-in, ng fine tailing away from the end of riffle box. Toms where running water and grade are available, a simple requires less labor. A modified Tom was used in beach ing devices of other countries, see Bib (392).

ns and Outline of Methods

applied to many forms of placer mining. A or trough, through which gravel is carried by a

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stream of water. Stones and light sands pass through and run to lower end; gold and other heavy minerals settle to the bottom a in riffles. A RIFFLE is a groove or interstice, or a cleat or block s produce the same effect, in the bottom of a sluice (1). Art 123, on sluices and riffles. Wooden sluices (Fig 591) are sometime SLUICES; inclined ditches, in gravel or bedrock, are GROUND-SLUIC ground-sluices are formed by the natural irregularities of their bo

Table 86 outlines the commoner combinations of excavating and transp used in open cuts for digging gravel and getting it into sluices. (For und ods, see Art 127.) Of the methods outlined, dredging and hydraulickin low-cost operations, suitable for large-scale mining of low-grade gravels large capital outlay. The others (excepting at times ground-sluicing) have

Table 86. Outline of Surface Methods of Placer Min

Method of excavation	Gravel loaded into	Method of transporting gravel to sluice	Rem
Running water aided by picking	Ground-sluice	Gravity	Called G sluicing
Pick and shovel	Sluice	Shoveling	Called Sl in
	Wheelbarrows; cars on tracks	Hand-tramming	
	Cars on tracks	Hand-tram to an incline and hoist to sluice on surface	Used wh bedrock topogra are flat
	Buckets or stone skips	Hand-tram on trucks to derrick or cableway, which hoists to sluice on surface	
Plow and scraper	Scraper	Scraper	.....
Power scraper	Scraper	Scraper	Scraper hoisted incline elevate
Steam shovel	Cars on tracks	Animal or mechanical haulage	.....
	Washing plant	Steam-shovel dipper	.....
Drag-line excavator	Scraper-bucket	Scraper-bucket	.....
Water under pressure	Ground-sluice	Running water	Called H licking
Mechanical excavator mounted on a boat with screens, sluices, etc			Called D

NOTE. — Work with elevators is not included (see Art 125). For v "River Mining," such as wing-damming, fluming, etc, see 9th Ann Rep, eralogist, 1889, p. 262.

high operating costs and require richer gravels to yield a profit. Work shovel or plow and scraper does not demand large initial expenditure for ment and is characteristic of small operations in rich gravel, or of mi of very cheap labor (397). The field of use for the power scraper and vator is stated in Art 121; steam shovels have not been a success.

Relative costs of different methods are indicated by Table 87

); they apply to difficult working conditions. ewhere depends on wages, cost of supplies, etc.

lining in Alaska, 1905. C. W. Purington (390)

	Location	Capacity, cu yd per 24 hr	Thickness of deposit, ft	Thickness of gravel worked, ft	Cost per cu yd
.....	Ip	150	9	9 (*a*)	$0.17
.....	Sp	173	4	4 (*a*)	0.10
.....	Ip	250	7.5	6 (*c*)	0.07
.....	Sc (*e*)	54	5.6	3.7	2.01
.....	Ip (*f*)	63	8.6	3.5	2.39
.....	Sp (*g*)	145	6.6	3.3	1.87
.....	Ip	184	8	6	1.25 (*h*)
cable					
.....	Ip(*i*)	162	17.5	4.5	2.14
ncline					
.....	Ip	450	14	5	2.43
skid-	Ip	233	15	9	1.75
cking	Sp	550	15	11	0.91
.....	Ip	105	20	10 (*c*)	0.60
.....	Sp (*j*)	200	5	5	0.46
.....	Ip (*e*)	92	15	8.7	0.49
: and	Ip	800	22	22	1.46
.....	Sp	1 000	30	27	0.52

redging, see Art 122, 125, 126. Ip = Interior Province. Sc = South Coast Province. (*a*) "Muck" or fine (*b*) Booming with self-dumping water gate (Art 120). Includes stripping top dirt, no pumping. (*e*) Aver operations. (*g*) Aver of 10 operations. (*h*) Tailings (*i*) Aver of 8 operations. (*j*) Aver of 5 operations. tailings.

ioveling-in, Horse-scraping into Sluices

shows typical ground-sluicing in shallow creek steep enough to allow it. Sluice *M* is set at the worked and a shallow trench *T* is dug, preferably is deepened to bedrock by turning a stream of picking up the bottom, the material being washed ng bedrock the trench is widened by picking and n. Small plank or earth dams *D*, erected where gainst the bank to aid in undercutting it and so l use forces a stream to do considerable work night shift. Coarse gold remains on bedrock, ce riffles. As the bank recedes, exposed bedrock and scrapers; the concentrates may be cleaned len placed at intervals along the face fork out on cleaned bedrock, as at *B*.

riations. In Alaska, it is often used for cheaply stripuck (Art 117). Tundra and brush are grubbed and be done as in Fig 573, or the water may be led through vening ridges eventually cave; spring floods are thus treams of water may be caused to trickle down the face nels; the ground between caves is blasted or pried off.

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In this way, in 1911, the Granville Mining Co, Dominion Creek, Yukon, muck 18 ft thick, the bank receding 4 to 7 in per day; est'd cost of work, 10¢ per cu yd of muck (393). Such work generally stops on reaching gr sluicing uses water as an excavating agent where pressure water for hydr available. (The term at times der shoveling into ground-sluices.)

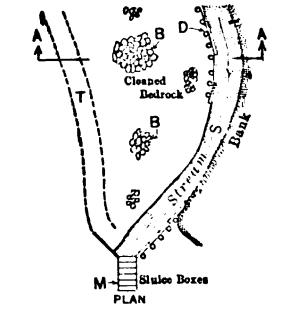
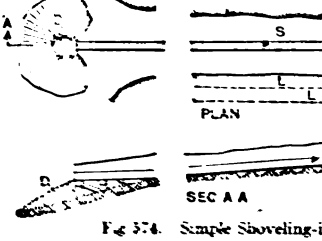


Fig 573. Ground-sluicing (diagrammatic)

Requirements for ground-sluici gravel, rarely more than 6 to 8 ft d cient grade for the available wat loosened soil. Data covering diffe are lacking. The Granville Co (se muck on grades of 17 to 25 ft per r gravel are much steeper. (*c*) Plentif again data are lacking. Longridge (it takes about 6 times as much wat terial in a ground-sluice as to do the box-sluice. Scanty water supply r be supplemented by pumping bac pools at end of sluice. (*d*) Dumpr at lower end of the sluice may be p natural grade of bedrock or surface are sometimes employed to move ta end of the sluice; worked-out area as dumps.

Duty of water and labor. Mead 2 men can move 20 to 30 cu yd ma day by ground-sluicing (395). A na Swaziland handles up to 10 cu yd of per shift, where a good stream of able (396, 397). In Colombia, in gravel 5 to 15 ft deep, a stream of 135 r 122) moved only 50 cu yd per day (385). Purington gives following data on s (50 to 75% ice) in Alaska. On Anvil Cr, Nome, 400 miner's in of wate high; grade, 4.5 in to 12 ft; duty of water, 10 cu yd per in per 24 hr. Anvil Cr, a 4.5-ft bank of muck was stripped with 100 in of water; grade duty of water, 3 cu yd per in per 24 hr. On Crooked Cr, near Council, 5 bank, 4 ft high; grade, 4.5 in to 12 ft; duty of water, 3.45 cu yd per in p

Booming ("hushing") is applicable where the water supply is inadequ work. Water is impounded above the diggings; and by releasing it at inte gravel through the ground-sluices. Booming is also used for strippin booming usually have gates opening automatically when the reservoir is fu gate is common, controlled by a lever carrying an open box at its outer en voir when full overflows into the box; the added weight lifts the gate, and empties itself. Gates must be so arranged that the dam will not be injured by the rushing water. Booming strips light overburden cheaply. In 1904 on American Cr, Alaska, an area 200 ft long by 20 ft wide was stripped of muck and gravel 5 ft thick in 3 weeks; costs including dam and gate did not exceed 2 cts per cu yd (394). See also (396, 397).



Shoveling-in (*i. e.* into sluices) has its simplest form in shallow creek gravels where bedrock grade is steeper than is required for sluices (Fig 574) or bedrock with the lower end raised and flattened to give headro

Fig 574. Simple Shoveling-i

D. Avoid dumpmen as obstacles as far as possible by extending lower

e cuts *T*, or more often in longitudinal cuts *L*. Only he sluice; large stones are piled on cleaned bedrock, at the head of the sluice, but require use of barrows arrow pits; with longitudinal cuts the sluice must be about 12 ft from it, otherwise, shoveling cost becomes ing may sometimes be avoided by feeding the main adiating or lateral sluices.

sluice in flumes, pipes or canvas hose. On Seward is commonly 12 to 14 in diam, of 12 or 14-oz duck, nd flexible, but begins to rot after about 4 mo service eek bed, water is dammed above the diggings.

are rare. The required grade must often be created e sluice, or by mounting it on trestles (Fig 589). The and quickly; it is often the only means of obtaining itate shoveling, but are too costly if deep or in hard rock. and top gravels are first removed by ground-sluicing; an 5 or 6 ft thick, is then worked by shoveling-in. Bar- where grades do not allow this to be ground-sluiced it rs; only the pay gravel is sluiced.

, clean-ups (see Art 123, 124).

ion is a self-draining pit (Fig 574). Where grades are both above and below the area worked; they are built ee Sec 43, Art 16).

y bedrock drains, on grades as flat as 1.5 in per 100 ft; the proper distance downstream to reach bedrock at n continue on bedrock grade. Purington (390) gives e, which is fairly typical. Drains are usually 2 ft sq, ft intervals by posts and caps. After the first cut in ered with logs and moss laid on the caps. For small s of 1-in plank are adequate. A perforated standpipe the lower end of the pit; drains may require flushing

Table 88. Duty of Labor Shoveling into Sluices in Alaska. Purington (390)

Location	Depth of gravel, ft	Height of lift, ft	Cu yd per man per 10 hr
Klondike (*a*)	2	9	3.5
Birch Cr (*b*)	4.41	(*c*)	5.0
American Cr	.....	.....	2.75 (*d*)
Fairbanks (*e*)	3.0	.....	7.5
Nome (*f*)	5 to 7	.....	5.76
Solomon River (*g*)	3.0	.....	3.75
Council Dist (*h*)	3.5	.....	6.63

(*a*) 2 men; platform used. (*b*) Aver of 12 operations. (*c*) Height of lift not over 6 ft. (*d*) Large boulders interfered with work. At one point, a 5-ft bank was shoveled at rate of 4 cu yd per man-shift. (*e*) Aver of 3 operations; lifts less than 5 ft; some bedrock taken up. (*f*) Aver figures; Pioneer Co, on Anvil Creek, 3-ft bank, obtained a duty of 9 cu yd. (*g*) High lift and irregular bedrock. (*h*) Aver figures, including one case where, on a limestone bedrock, with a double lift, duty was 3.5 cu yd; another where, with 8-hr shifts, 3-ft bank and 5-ft lift, duty was 12 cu yd.

eling-in is adapted to shallow gravels. It is simple, e for plant and can be carried on in remote districts al excavators would be prohibitive. In Alaska, where

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labor and supply costs are high, the cost of plant, including dams, drai
ditch, and a string of 10 sluice boxes (Art 123), is from $500 to $2 000 (39
has the great advantage of allowing careful cleaning of bedrock. Oper
widely with wages and duty of labor; only rich gravels can be worked

Cars and barrows for loading sluices. Fig 575 shows radia
small mine cars (Sec 11). To use this method the head of sluice
near the level of
must have a fairly r
Barrows may be use
on rougher bedrock
runways.

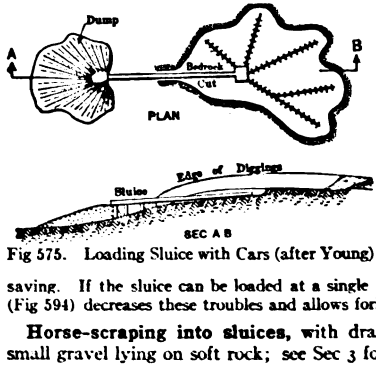

Fig 575. Loading Sluice with Cars (after Young)

Inclined barrow runv
an elevated sluice, are
angles to the sluice,
longitudinal cuts (*L*, Fi
of barrow and car wo
An objection to them is
large amounts at one t
the flow in the sluice
clogs and is not conduc
saving. If the sluice can be loaded at a single point (Fig 575), a dum
(Fig 594) decreases these troubles and allows forking out of large stones

Horse-scraping into sluices, with drag or wheel scrapers,
small gravel lying on soft rock; see Sec 3 for details and cost of

Purington gives following data for Alaska (390): A 2-horse wheel scr
small gravel on soft schist bedrock, handles 30 to 40 cu yd per 10 hr at
the cost of shoveling-in; 1 plow loosens gravel for 4 scrapers. The scrape
a hatchway in a platform built over the head of the sluice, inclines lea
form from both sides. Teams generally travel an elliptical track, passin
sluice to remove a load of tailings, then entering the pit, scraping up the pa
it to the sluice. Strong points in favor of scrapers are their low first cost a
in many cases power scrapers (Art 121) have proved cheaper in Alask
high cost of maintaining horses (390).

121. Use of Derricks, Cars and Inclines, and Mechanica

Derricks may be used in shallow open cuts in the following wa
is shoveled into barrows, wheeled to a fixed point and dumped i
skip or bucket; b gravel is shoveled into stone skips, or bucke
small trucks running on rails and pushed by hand to within reach
boom; c gravel is shoveled into buckets which are slid along
derrick tackle (see example below). Plan a requires greate
lateral handling; miners spend much time in wheeling and du
at picking and shoveling. Plan c is the most efficient in this
matter, but decreases its operative as follows:

Example. M
deep
of
Bedrock was a very
into 11-cu ft
were
on bedro
beyon
and pushed
while being pu
were
and dumped into b
over 12 m diam are
Over output, 500
cost of wood per 24 hr

ick is moved ahead 110 ft and re-erected in about 18 hr. ve the ground, and sluice is 132 ft long. Tailings are

ig 576. Derrick Working, Council District, Alaska (after Purington)

cking (Art 122), the pay gravel, 6 ft deep, was trammed an elevated sluice and dumped by hand. Sluice tailings

Cars and Incline (after Purington)

th 2-horse scrapers. Each car was manned by 2 men, ther tramming to the incline. With 6 22.5-cu ft cars,

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250 to 300 loads were handled per 10 hr. Crew outside the pit: 1 man to tend dump box, 1 hoistman who also fired the boiler; coal consumed,

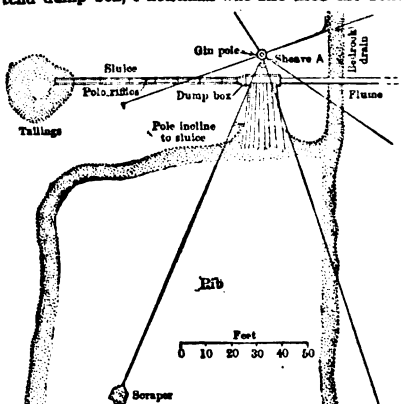


Fig 578. Power Scraper

This method lacks power-scraping or the tramming lab drawback where w

Power scrape 6) are well adap and elevating gra low placers; they to hard or block soft, even bedroc times be cleaned but final hand cl ally necessary. S vation of a tha gravel exposes to the underlying f adjacent areas a scraped and tha scrapers are also for removing taili of sluices. Large l are generally usec or wheel scrapers and too difficult loading when dra

In the Klondike, toothed slip-scrapers, holding 0.33 to 0.5 cu yd, ope 30-hp double-drum hoists, handle an aver of 250 cu yd of tailings per fireman, hoistman, and 1 or 2 men to fill, guide, and dump. Scrapers work through horiz distances of 100 to 300 ft and heights of 20 to 50 ft. Aver cost of plant, $3 500 (390).

H. I. Ellis (399) and J. F. Katz (400) give following data on power-scraping in Fairbanks District, Alaska: Bagley scrapers, bottomless or of the wheel type, have replaced other types; in 1914, 15 were in use, most of them operated by 3-drum hoists with cylinders from 7 by 10 in to 10 by 12 in. Creek gradients are flat, requiring elevated sluices to secure dumproom for tailings. Fig 578, 579 show 2 layouts; scrapers may be hauled up an incline, or may dump into cars or skips which are hoisted to the sluice; a large saving in fuel is claimed for the latter method. The 3-drum hoist with 2 tail lines, *T*, Fig 579, allows the scraper to load at any point. In Fig 578, the scraper is hauled by a continuous rope on a single drum; the lead of the pulling rope is changed by shifting sheave *B*. This rig operated a plow and scraper on alternate shifts. At a fairly typical mine (Fig 579), 3 to 4 ft of overlaid by 4 ft of barren gravel and 6 ft of muck. After the muck w

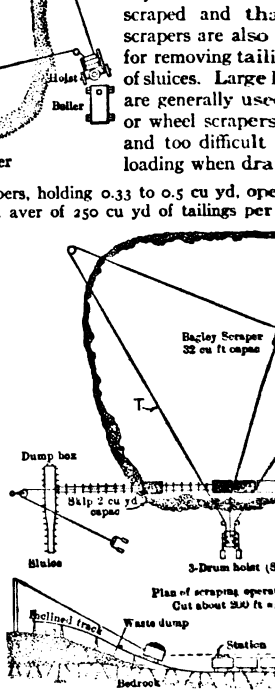
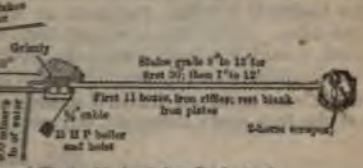


Fig 579. Power Scraper and Au

of the waste gravel, much of which was sent through
ame) at considerable expense, instead of being piled at
claim was diverted past the pit by allowing it to cut
of the pit was 40 000 sq ft, or about half a normal
ing removed, the scraper stripped about 1 000 sq ft of
ver crew per shift: 1 engineer @ $9.50; 1 fireman and
ers, $6.50 each; several extra men cleaned bedrock.
re burned per shift; wear and tear and depreciation,
r shift, $89. Erecting the plant and stripping over-
st season operated; it was expected to reduce this to
foreseen difficulties encountered in the starting opera-
g and mining would be 42¢ per cu yd handled or about
high costs are due to high labor and fuel. First cost
ound like that described is about $12 000.
el to shallow creek gravels; it is preferable to any
here the size of placer warrants initial cost of scraper
referable to drag-line excavators.

ws a typical layout for open-cut work. Gravel
se or locomotive to the foot of an incline, where



ovel Work, Alaska (after Purington)

nto a fixed sluice. The shovel sometimes loads
ng a trommel, sluice, gold-saving tables, etc,
the shovel track or on one alongside. Long-
lly best; banks less than 12 ft high require too
high have been mined in 1 slice. For data on
nd Sec 3, 5; for applications to placer mining,

n a commercial success in placer mining, though it
icking is prohibited by lack of water and grade. Its
lue to one or more of the following causes: (*a*) at-
rd bedrock which can not be dug by the shovel and
xpense; (*c*) it lacks mobility, as compared with hy-
s drawback to any placer-mining device and which
o provide transporting and washing facilities in pro-
therefore idle much of the time; (*e*) gravel is deliv-
nittently and in large amounts; the sluice or other
then underloaded, resulting in a loss of gold. Stor-
ead of the sluice obviate this difficulty, but entail
st and operating costs; (*f*) trouble with the disposal
lants. Obviously some of these difficulties have been
tem (*c*), probably the most important, is inherent in

and Sec 3) has had limited use in placer mining.
rawbacks of the steam shovel and cannot dig as hard
an irregular rock surface. But it is more mobile and
it has a wider digging radius, hence less frequent
on the surface and dumps at a considerable elevation
uices may be obtained without a separate elevating
le for a steam shovel standing on bedrock; (*c*) the

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operations of loading, tramming and elevating cars, common to many s
ations (Fig 580) are rarely necessary; the long boom of the excavator
its own transporting and elevating. Drag-line excavators can also min
of loose gravel which would not warrant installation of a dredge. Fo
tions and detail, see Bib (385, 403). Crew for a drag-line excavator lo
a sluice consists of an operator, a fireman and 2 to 4 men on the track

122. Hydraulic Mining or Hydraulicking (See also A

Term "Hydraulicking" is applied to the excavation of gravel ban
water thrown under pressure from nozzles. The method was invent
1852, by E. E. Mattison; see Bib (391) for history of its development.

General plan of work. Water is impounded in reservo
from streams, and conducted in ditches, flumes, and pipes to ar
above the placer and thence to the working face through pi
terminate in GIANTS or MONITORS (Fig 586), by which the direct
controlled. The gravel is broken down partly by direct impa
and partly by undercutting banks and forcing them to cave.
away from the face (usually in bedrock ditches), carrying the g
a sluice leading to a dump; the giants are often used to "pip
into the sluice. As the face is washed away, the bedrock cuts
advanced toward it. Some cemented gravel banks require
they can be disintegrated by water (see Bank blasting, below).

Details of water supply and methods of opening and advancing worki
ment of pipe lines, giants, sluices, etc, vary with local conditions; t

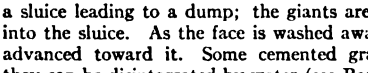


Fig 581. Simple Layout for Hydraulicking (after Young)

cuts indicate diversity
also Fig 609). Fig 58
ditions of grade and
582, layout for a bench
a mine on a small str
bedrock grade (2.35%
to provide dumproom
to be elevated; the pit
a dump as work proce

Tunnels are sometim
sluices to lower grou
gravity dump; also for opening placers, where local conditions prevent
on bedrock. Thus, in Fig 584, tunnel T is driven on sluice-grade to
of the bedrock, and connected with the surface by 1 or more shaft
hydraulicked into the shaft, the timbers and protecting lining of whic
the excavation is deepened. On reaching bedrock, the gravel bank is hy
sluice in the tunnel. Bowie
states that 300-ft shafts in Cal
were readily operated in this
way; their cross-sec was 3 by
3 ft to 4.5 by 9 ft; tunnels
were 7.5 to 8 ft high and 2 to
3 ft wider than the sluice, to
give room for construction and
cleaning-up (391).

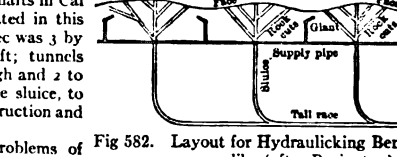


Fig 582. Layout for Hydraulicking Ber
dike (after Purington)

The principal problems of hydraulic mining are connected with water supply, sluice grades, and dumproom; the method appears sin
limitations. Some small-scale hydraulicking is successful, but as a rul
large outputs; it requires large initial investment, and skilled management

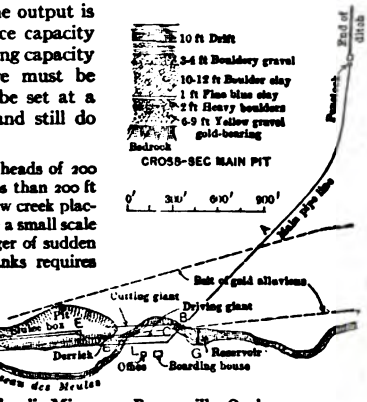
Size and tenor of deposit must be such as will amortize the inve
for its exploitation and pay the profit desired. The minimum gold
yield a profit varies with locality from 4 to 50¢ or more per cu y
capital invested, rate of working, life of property, working costs an

bedrock (see Art 117).

ple for the required output. The speed of work-
eturn, and depends largely on amount of water
below), which is related to the pressure; a small
n breaks down as much gravel as a larger amount

e output is
ce capacity
ng capacity
e must be
be set at a
nd still do

heads of 200
s than 200 ft
w creek plac-
a small scale
er of sudden
nks requires



draulic Mine near Beauceville, Quebec

ants may throw streams from a considerable distance.
irements for heavy pipe, bracing, and anchorage become
npts made to pump water for hydraulicking have nearly
) (compare Art 99).

pted to most cases is 4 to 5%. Cost and difficulty
le flattens, until a point is reached where hydrau-
23). Grades may sometimes be increased artifi-
t 125).

able at the end of the sluice; it is computed from
le for the swell of excavated gravel.

al stream which will carry away tailing, provides ideal
y well situated as regards dumproom. Large dumps are
built up on sloping ground by gradu-
ally extending lower end of the sluice
(Fig 581); branch sluices are often
installed and the dump widened
(fanned-out) as well as lengthened.
In either case, slope of the ground
must be steeper than the sluice grade.
Tailing must sometimes be impounded
mining or farming land. This condition may prohibit
ity or cost of installing and maintaining DEBRIS DAMS.
Hydraulicking in the Sacramento and San Joaquin drain-
in 1893, when the Caminetti Act was passed, creating a
ydraulic mining and prevent further silting-up of these
d for types and construction of debris dams, see Bib (405).

ened by Tunnel

s, flumes, trestles, sluices, riffles, sluice linings, etc. For
sually required at the start, located preferably near the
above or below where the water is diverted. Timber is
s needed. Cheap timber materially reduces costs (385).

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Working season is often limited by climate or water su seasons vary from 100 to 120 days. Where climate permits co water is rarely available the year around, even with reservoirs flood waters and melting snow. Length of season, proper scal annual returns, and capital required, are closely related.

Summary. Requirements for successful large-scale work a fied completely. Adequate amounts of gravel, gold, water ar feasible working season are absolute essentials. Lack of natur and dumproom generally prohibits extensive hydraulicking, l times be met by use of elevators (Art 125); the latter add an capital cost which may or may not be prohibitive.

Miscellaneous data on equipment. WATER SUPPLY is usually item; see Sec 38 for general data on rainfall, runoff, reservoirs, ditches, Flumes are preferable to ditches in all doubtful ground; both should turnouts and sand gates. Distributing reservoirs are required near avoid wastage when giants are turned off; their number, size, and im with the length of ditch and number of shut downs that occur. Almo expenditure is justified for preventing breaks in the supply system; c is essential to large output and low costs. A PRESSURE BOX (penstoc upper end of pipe leading to giants, to catch floating leaves and sticks, se and keep air out of the pipe; for details, see Bib (391, 406). A single able if it will supply all the giants; it must be carefully anchored and d in as direct a line as possible. Air valves are necessary to prevent colla it is emptied. Each branch leading to a giant should have a gate.

Operation of giants. Banks are usually broken by undercutting an ing faces should be kept square and advantage taken of corners. Na concave faces are avoided, as workmen are constantly menaced by Water should be turned away from a bank that is about to cave, othe caved material may reach the giants and men. Water cuts fastest whe a bank at an oblique angle. Banks shaped as in Fig 582, with giants nose, allow effective use of water and reduce danger from caves, as the l at right angles to the face and not toward the giant. Some Alaska creek gravels 15 to 50 ft deep by setting the giant on top of the bank stream, but the duty of water thus obtained does not indicate a high With 1 giant, part of the bank is caved, then with a larger nozzle the falle to the sluices. With 2 or more giants, their time is apportioned be sluicing. BOOSTER GIANTS may be used to drive gravel across flat bed

Number of giants depends on size of working face, character of gra amount of water. Practice tends towards use of a small number of la than a larger number of small ones; the former cut faster and save l placed to command as much of the face as possible, to work advanta bined attack on one point, and as near the face as is safe (406). For be 2 giants and 2 working faces are required, so that hydraulicking ma face while boulders are being cleared away from the other (394).

Height of face for economical work is normally limited to a max higher banks are usually worked in benches. Conditions allowing high water pressure, compact gravel that does not "run" when caved, and Where the bottom layer of a high face is tough, the upper gravel ma enough to allow the giants to be moved closer to the face and exert m

Bank-water (bank-head or bye-water) is a term applied to strea pit in ditches, not under pressure. Bank-water is allowed to run ov in disintegrating clayey gravel, thawing frozen ground, moving gravel maintaining a steady flow through it.

Bedrock cuts or ditches are a continual and sometimes serious so most hydraulic mines; in some early Cal mines, they reached 60 ft de case shown in Fig 582, 4 men were constantly employed in extending When much clay is present, more branching bedrock cuts are requ

bedrock also influences spacing of cuts. Drainage of pits es; occasional bedrock drains are required.

ped to and through the sluice are removed by derricks or derricks must be located behind the giants, they must d are cumbersome to move. Cableways do not interfere d reach sidewise. Derricks and cableways may be oper- either is usually preferable to blasting where boulders are rades are flat (406). Sledging, if feasible, is generally the lders if done by cheap labor and without delaying piping; sually costs more than blockholing. R. H. Ernest, Round ves following data on mudcapping 839 rhyolite boulders, 4 cu ft, in June, 1916. 402 lb of 40% gelatine dynamite der. Cost per boulder was 25.7¢, of which explosives cost

in cemented gravels before they can be broken down by ls (Fig 585) are usually driven at the foot of high banks. osive and drift E is tamped solid Only enough explosive is used to es several cross drifts D are driven. arge blasts in Cal (391). In banks ift E should be 66% of height of ls on extent of ground to be moved. (black powder, Judson powder, or o lb per 1 000 cu ft of gravel cov- sts require proportionately more 1 000 cu ft gravel. A blast will area of ground covered by the drifts. In some cases, where nted, the top gravel is hydraulicked, and small shafts are sunk to bedrock, where the charge is placed in small chambers. Unit costs of blasting increase rapidly as height of bank decreases. Well-drilling machines (Sec 9) might now be advantageously used in blasting high banks.

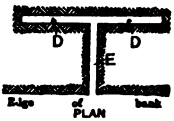

Fig 585. Bank Blasting

anks of Cemented
nd Mountain Min-
(R. H. Ernest)

Hand churn-rills (a)	Jack-hamers
996	718
8 to 22	3 to 6
966	104
.08	6.9
; 210	700
650 (b)	260 (c)
0.124	0.371
$0.036	$0.082
0.030	0.122
$0.066	$0.204 (d)

by giants with noz- (a) ⅞-in oct steel. 00 lb 40% gelatine hat 2 000 cu yd was . (c) 40% gelatine of air, etc.

Miscellany. CLEANING BEDROCK is sometimes done by giants; hard creviced bedrock must be cleaned by hand at high cost. LIGHTING is necessary at night, since work is continuous; modern equipment usually includes a small hydroelectric plant. LABOR. Skilled men required are: nozzle tenders, carpenters, blacksmith; sometimes a machinist and derrick winchman. For SLUICES, RIFFLES, etc, see Art 123, 124.

Giants (monitors). Fig 586 shows a modern, double-jointed giant, connected to pipe line by a slip or a flanged joint. It can swing horizontally through a full through a wide vertical angle about joint M. Weight ed by lever S and weighted box P; lug L is bolted to a ned and braced to bedrock; 1 or more guide vanes are nt rotary motion of the jet which destroys its solidity. . A vertical king bolt K, to take the thrust as the water top of the bolt may have a ball-bearing to reduce friction h heads of 600 to 700 ft, at La Grange, Cal, heavy lugs at

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joints *J* and *M* protect operators from injury in case elbows or joints fail (Boue
Some makers replace the king-bolt by a special ball-bearing joint at *J*, to avo
tion to the flow, but the king-bolt is a good safety precaution, and, with we
pressure boxes, should give no trouble by catching stringy matter.

Table 90 gives data on Hendy giants; other makers build giants of similar s

Table 90. Data on Hydraulic Giants (Joshua Hendy Mfg Co, San Franc

Size number	Diam of pipe inlet, in	Diam of butts with nozzle detached, in	Effective head, ft	Size of nozzle and flow, cu ft per min under given head†		Wt of heaviest part, lb	Shipping wt, lb	List
								Double-jointed giant
0	5	3	100	1″ 24	1½″ 53	100	330	$ 84
			150	1″ 29	1½″ 65			
			200	1″ 33	1½″ 75			
1	7	4	100	2″ 94	3″ 180	120	390	110
			200	2″ 133	3″ 260			
			300	2″ 163	3″ 320			
			400	2″ 188	3″ 385			
2	9	5	100	3″ 200	4″ 340	150	520	137
			200	3″ 235	4″ 500			
			300	3″ 340	4″ 615			
			400	3″ 400	4″ 716			
3	11	6	100	3″ 222	4″ 360	210	890	204
			200	3″ 300	4″ 515			
			300	3″ 368	4″ 635			
			400	3″ 425	4″ 736			
4	11	7	100	4″ 380	6″ 800	225	1 075	252
			200	4″ 535	6″ 1 150			
			300	4″ 655	6″ 1 420			
			400	4″ 756	6″ 1 650			
5	13	8	100	5″ 590	6″ 810	335	1 475	352
			200	5″ 835	6″ 1 160			
			300	5″ 1 000	6″ 1 430			
			400	5″ 1 180	6″ 1 660			
6	15	9	100	6″ 825	7″ 1 120	520	1 850	452
			200	6″ 1 175	7″ 1 600			
			300	6″ 1 445	7″ 1 990			
			400	6″ 1 675	7″ 2 260			
7	15	10	100	6″ 850	7″ 1 140	520	2 100	536
			200	6″ 1 200	7″ 1 615			
			300	6″ 1 470	7″ 2 000			
			400	6″ 1 700	7″ 2 280			
8	18	10	100	7″ 1 160	8″ 1 510	600	2 300	625
			200	7″ 1 635	8″ 2 120			
			300	7″ 2 025	8″ 2 620			
			400	7″ 2 300	8″ 2 920			
9	18	11	100	9″ 1 920	10″ 2 360	690	2 450	676
			200	9″ 2 700	10″ 3 320			
			300	9″ 3 320	10″ 4 090			
			400	9″ 3 820	10″ 4 720			

* Subject to discount and do not include price of deflectors. † Using a c
charge of 0.88 to 0.90, see under Discharge through nozzles. (See also Sec 38

Nozzles (*N*, Fig 586) of any maker are interchangeable; nozzle o
may be attached to a giant up to the max it will take. They must be
designed and highly finished, otherwise the jet is ragged and cannot b
as far without spraying. Grit in the water cuts the nozzle, and des
shape of the jet.

or those working under light heads, are pointed ed on large giants and in general where the head

Fig 586. Giant or Monitor

rted by the deflector, forces the giant in the opposite ves a sensitive control. Type (*b*) is best for heads over 300 ft, but requires a separate deflector for each size of nozzle. A single deflector of type (*a*) will take any nozzle smaller than the max that can be used on the giant.

Discharge through nozzles (Sec 38, Art 9). Approx calculations may be made by the formula, $Q = Ca\sqrt{h}$, where Q = dis- -sec area of nozzle, sq in; h = head at nozzle, ft, ff of discharge.

e above) the coeff of discharge is put by Longridge J. Garrard (397) as 0.94. Corresponding values of C ly. Makers' tables usually give full theoretical dis- e for friction.

19), unless otherwise stated, is taken here as ft per min, corresponding to its legal definition n = 1 cu ft per sec). Old records of the duty of

er's Inch at North Bloomfield and La Grange es, Cal. Bowie (391)

	Miner's 24-hr in	Grade of sluices	Height of bank, ft	Duty of miner's in, cu yd
0	710 987	.....	100	4.6
0	386 972	6.5 in per 12 ft	100	4.8
0	700 000		200	4.17
0	595 000		265	3.86
0	2 392 959	.....	.....	4.6
8	624 745	4 in per 16 ft	10– 48	1.08
4	375 155		60	1.82
2	207 010		50– 80	1.37
0	302 960		40– 50	1.52
0	203 325		10– 80	1.57
4	1 713 195	.....	.....	1.42
...	.....	8 in per 12 ft	100–200	7–10

early Cal work was estimated by State and U S army bout 3 cu yd. J. H. Hammond gives following data: ng a 120-ft bank, using 500 in water at head of 360 ft, er 12 ft and paved with rock riffles, duty of a miner's and 10 cu yd in coarse cemented gravel. By increas- and using iron riffles, duty of one inch in light grave'

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increased to 36 cu yd. These figures are exceptionally high (407). (*a*) material moved was top gravel. Sluices, 6 ft wide by 32 in deep. Blo almost entirely; rock riffles in tail sluices. (*b*) Sluices previous to 1881 by 30 in deep, paved with blocks. (*c*) French Hill claims. (*d*) Light Cla nau Claim. (*f*) Kelley Claim. (*g*) Vigno Hill. (*h*) Data from P. Bou by Joshua Hendy Iron Works, Bull No 111. Sluices, 6.33 ft wide by 47 in duty is due to steeper grades, higher banks, and improved riffles (Art 1 of water under heads of 400 to 600 ft.

Table 92. Duty of Miner's Inch, Alaskan Districts. C. W. Purington

Ex No	Location	Height of bank, ft	Character of material	Grade, in per 12 ft	Miner's in of water	Duty, cu yd per miner's in	Sluic Width, in
	Juneau:						
1	Windfall Cr	16	....	...	1 000	2.0	...
2	Gold Cr	200	A	4.5	4 000	2.0	72
3	Silver Bow Basin	80	A	4.0	2 500	2.0	50
	Atlin:						
4	McKee Cr	40	A	8.0	700	1.5	32
5	" "	85	A	8.0	1 200	3.0	32
6	Birch Cr	25	A	5.25	1 200	0.5	30
7	Spruce Cr	29	A	5.0	1 200	1.0	40
8	" "	20	A	5.5	900	0.5	48
9	Pine Cr	20	A	3.0	700	0.6	36
10	" "	60	A	5.0	3 500	0.625	60
	Dawson:						
11	Bonanza Cr	20	B	12.0	250	5.0	24
12	" "	25	C	12.0	125	4.0(?)	20
13	" "	25	B(*a*)	11.0	150	10.0	24
14	" "	35	B	12.0	200	6.5	24
15	" "	75	B	12.0	266	8.0	24
16	Eldorado Cr	60	B	11.0	160	7.0	24(?)
17	Last Chance	46	B	12.0	230	7.0	24
18	" "	25	B	14.0	120	5.0	20
19	Hunker Cr	15	B(*b*)	13.0	150	2.0	14
20	" "	8	B	12.0	125	2.6	17
	Nome:						
21	Anvil Cr	30	D	4.5	500	1.4	33
22	" "	20	E	4.5	400	10.0	...
23	" "	12	....	9.0	100	.....	16
24	Glacier Cr	20	F	6.0	760	1.32(*c*)	36
25	Dexter Cr	40	G	6.5	100	2.0(?)	16
26	Newton Cr	8	H	7.0	150	3.33	24
27	Basin Cr	20	I	8.0	250	0.6(*d*)	18
	Council:						
28	Ophir Cr	12	I	10.0	600	1.5(*d*)	24
29	" "	8	J	10.0	750	1.0(*e*)	24
30	Solomon River	5	K	10.0	600	0.8(*f*)	37

A = heavy stones. B = "White Channel"; small round gravel, fro side, small gravel, frozen. D = heavy, partly frozen gravel, much fla frozen earth or muck stripping. F = flat, semi-frozen gravel. G = limestone bedrock. H = small coastal plain gravel. I = heavy gra and clay. K = subangular gravel. (*a*) part tailings. (*b*) very little fro of the water used by an hydraulic elevator, 36 ft lift. (*d*) hydraulic elev used 60% of water. (*e*) hydraulic elevator, 28-ft lift, used 66% of water. elevator, 12-ft lift, used 50% of water. Bl = block-riffles. Ai = angl long rails. Sp = sawed pole, iron-shod. Hu = Hungarian. Pi = po P = pole. Aig = angle-iron grates. Ra = rails.

careful scrutiny; in Cal alone, the inch varied er min (391).

hydraulicking is the cu yd of gravel which can ough sluices by 1 miner's in in 24 hr. It varies of gravel and bedrock, grade of bedrock, amount h all the factors influencing sluice capacity (Art state completely all the factors involved. See d under Cost data below.

2 as follows: The high duty of the miner's inch in the ell rounded gravel containing no large stones, heavy les. Low duties at Nome (Ex 21–27) are due partly and rough, partly to use of hydraulic elevators, which water. Duties given are for both giant and elevator ommonest, but have little effect on duty because the d grades refer to bedrock sluices leading to the ele- operator's statements and comparatively short runs.

kon Gold Co. Twelve Mile River Water System (408)

ater pply, ner's in	Cu yd gravel	Cost of operating and ditch mainten- ance	Cost per cu yd, omitting deprec on main ditch (Table 98)	Duty of water, cu yd per miner's in
206	1 656 020	$140 433	20.80¢	4.50
580	2 125 750	135 710	15.50	5.40
249	2 967 750	76 760	9.37	5.40
135	2 875 952	73 054	9.70	6.60
834	3 241 641	64 397	7.60	6.02

f starting large scale hydraulicking, and the close rela- Frozen ground makes ditch construction hard; new which, however, lessen with time. Table 93 covers ring data on Bonanza Creek are illustrative:

tretch of the "White Channel," worked here, is par- above it. Bedrock is decomposed schist; most of the and the first 3 ft of bedrock. The channel is 500 to o to 20 ft on the rims, to 150 ft in deepest sections.

nanza Creek Mines, Yukon Gold Co (408)

	B	C	D	E	F
5	30	15, 16	22 to 14	16	14
10	14, 12	12	10	12	12
2	No 3, No 5	No 3	No 2	No 3	No 3
3	(*a*)	5	4	(*b*)	2
4	4, 5, 6	4, 5	3, 4	4, 5	5, 6
5	85	55	45	85 to 90	85
0	200	400	150	200	300
6	52	48	36	36, 40	40
4	30	30	24	24	30
8	8	8	8	8	8
Bl	Lm, Cm, Sp	Lm, Ai	Lm, Bl	Lm, Ai	Lm, Ai

(*a*) 2 No 3 at American Gulch; 2 No 3 and 2 No 5 at on, 4 at Monte Cristo. Lm = Longitudinal manganese Cm = Manganese steel riffles, placed crosswise. Sp = sluice; plates 0.5 in thick by 6 ft long and 1 in narrower s, placed crosswise. A = Adams Hill. B = American rked in conjunction. C = Cheechaco Hill. D = Gold te Cristo, worked in conjunction. F = Magnet Gulch

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Gravel on Bonanza Cr is frozen and has no muck covering; when thawed broken down and washed. Most of the water comes from watershed of the ' River in ditches, flumes, and pipes; total length, 70 miles (Table 94).

Table 95 is on hydraulicking cassiterite, occurring mostly in a layer of c 0.5 to 3 ft thick, on decomposed granitic bedrock and overlain by finer alluviur

Table 95. Duty of a Miner's Inch, Swaziland Tin Co, Ltd, Sout
Year ended June 30, 1915 (397)

Example No	Running time, hr	Nozzles			Aver height of bank, ft	Total cu yd gravel
		Aver diam, in	Gage press, lb	Disch, cu ft per min		
1	3 595	1.84	59	98.6	10	100 792
2	4 070	1.87	73.5	116.0	14	136 377
3	1 955	1.86	68.7	108.7	13	46 619

Table 96. Costs at No 8 Claim, North Bloomfield Co, Ca

		1874–1875	1875–1876	18
Year				
Cu yd of gravel mined		1 858 000	2 919 700	2
Yield per cu yd, ¢		3.99	6.60	
No of days worked		295(*a*)	342(*b*)	
Miner's inches of water used		386 972	700 000	
Duty of one inch		4.8	4.17	
Grade of sluices		6.5 in to 12 ft	6.5 in to 12 ft	6.5
Height of bank, ft		180	260	
Cost per cu yd	Labor	$0.0123	$0.0140	
	Explosives	0.0016	0.0035	
	Riffles (blocks)	0.0016	0.0018	
	Material	0.0030	0.0032	
	Water	0.0078	0.0075	
	General	0.0023	0.0025	
Total cost		$0.0286	$0.0325	

* See also Table 91. (*a*) Jan 1–Oct 14. (*b*) Nov 13–Oct 18. (*c*) Nov

available only during the rainy season. Natural grades are steep; a rough ti collected in ground sluices, is periodically shoveled out and cleaned up in shor

Table 97. Costs on All Claims of La Grange Co, Cal

Period, June 1, 1874, to Sept 30, 1876	Total cost	Cost per cu yd
Water	$ 17 307.62	$0.008
Labor	82 345.70	0.036
Material	21 788.35	0.010
Official	11 244.94	0.006
Contingent	3 125.80	
Taxes	1 130.41	
Total	$136 942.82	$0.060

Note. 2 275 967 cu yd of gravel were washed, yielding an aver of 10.19¢ per cu yd (see Table 91 for further data). Sluice grades were flat and pressures light; this deposit was considered to represent lowest grade gravel that could be worked at a profit under these conditions. For recent work at La Grange, see below.

Cost data. Hydrauli from 2.5¢ per cu yd under able conditions in Californ more in Alaska. Unfavo tions, as frozen gravel, or elevating tailing or for pu water, may increase cost more. Records of cost us only operating expenses, o ital charges; the latter n high; thus, a number o spent $200 000 to $500 oc ing for work; the equipme including 290 miles of ditc of pipe, cost $2 000 000 (4 included in operating cost by following examples (s 125).

California, early work. Bowie (391) gives Tables 96 and 97, showin favorable and unfavorable conditions. Wages, $2 to $3.50 per day.

om W. H. Radford (410). Record is from Nov, 1899, 24 525 miner's inches of water washed 1 251 399 cu yd d). Pay gravel to 8 ft thick lay on bedrock, covered by le broken rock, clay and soil. Height of banks, 50 to 130

Mine in Northern Cal	Total cost	Cost per cu yd
f ditch, reservoir and siphon (*a*)	$ 2 786.54	$0.00223
ing (piping)	2 401.05	0.00192
ng in bedrock cuts	1 320.53	0.00105
ering bedrock cuts	157.39	0.00012
ric lighting	598.62	0.00047
building and repairing (*b*)	1 081.20	0.00086
smithing	644.02	0.00051
ing-up	968.79	0.00077
ng pipes and giants	898.85	0.00071
ting rocks and clay	6 124.91	0.00490
ng brush	158.37	0.00012
ral expense (*c*)	3 088.69	0.00250
ies	3 015.37	0.00241
, office expense, surveying, salaries	4 267.31	0.00341
al	$27 511.64	$0.02198

Labor, $2 670.99; supplies, $115.55. (*b*) Labor, 70; supplies, $35.50. (*c*) Watching sluices and odd jobs.

ow in use; cost, including reservoirs, dams, etc, about roll, including boarding-house force. Method of opera- nozzles are used at one time; head of water, 600 ft; either o miner's in of water, emptying the distributing reservoirs bout 4 hr, while the reservoirs fill; meantime the pipe-men nted gravel. Boulders up to 2 tons sometimes go through sually blasted to save water. Cemented gravel is gen- ves the pit and little or none is found on the dumps. to the sluices about 1 000 cu yd of gravel per hr. Sluice long; see Art 124 for riffles. The great length of sluice dump; at 1 300 ft it branches into 2 parts to distribute the gulch. Each year about 50 ft of sluice is added to ary for different seasons from 2.8¢ to 3.5¢ per cu yd, as follows: maintaining ditch, 18; piping gravel, 9; sluice ers, 9; pipe lines and giants, 7.5; roads and buildings, 3.5; ; taxes, 3; teaming, 5; boarding house, 8; misc, 2.5.

ing Co, Atlin, B C; data from A. D. Hughes, Supt, inclusive (411). Main operations were in 2 pits in an ut through and exposed by Pine Cr. Grades of channel vely; early ground sluicing and drifting has filled much drock is serpentine, containing many dikes. The gravel of coarse tailing can be dumped into the creek, which he gravel) without blocking. Pit No 1 is well opened; ve creek bed; gravel comprises 10 to 20 ft of channel nd 1 to 10 ft surface alluvium; clay is tough, but not laces 500 ft from the creek. Head on giant, 140 ft. In eep. Channel bedrock averages 10 ft above the creek toward creek; gravel extends 300 to 400 ft back from Water is brought in a ditch (capacity, 10 000 miner's in) 8 ft per mile; cost, $6 285 per mile, including a dipper h which it was dug. Reservoir dam cost $8 158. Equip- house, bunkhouse, living houses, warehouse, and stable l (capac, 2 000 bd ft per day) is operated by a special ers; rough native lumber costs $30 per M bd ft at mill. hine shop containing a 16-in by 8-ft lathe, drill-press

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forge, etc, run by a Pelton wheel. A 15-kw, 110-volt, direct-current gene connected to a 4-ft Pelton wheel receiving water under 75-ft head through a furnishes power for about 100 incandescent lights and 3 arc-lamps at Pi other generator lights Pit No 2. In each pit are 2 main pipe lines, 1 500 to consisting of 30, 28, and 26-in slip-joint pipe, of No 12 and 14 gage steel. pipes to giants are mostly 16 and 18-in diam, of 14 and 16 gage steel. At p are 6 No 6 giants, 4 with 7-in and 2 with 6-in nozzles; the smaller nozzles for stacking tailing (Art 125). A No 4 giant is operated on tailing or in the p

North Columbia Mine, Atlin, B C		Pit No 1	Pit No 2
Possible running time, days per season...		184	180
Actual operating time, " " " ...		154	141
Cu yd worked per season.....		283 300	178 580
Cu yd per day for season.....		1 540	992
Cu yd per day for time run.....		1 840	1 266
Miner's in water used per day.....		4 500	4 250
Cu yd per in per day running time.....		0.41	0.30
Aver depth of ground.....		60 ft 6 in	16 ft 4 in
Aver cost per cu yd, cts	Labor.....	7.34	13.29
	Powder.....	1.40	1.96
	General operations.....	2.18	3.07
	Ditch maintenance.....	0.22	0.35
	General expense.....	0.44	0.70
	Royalties, rentals, etc.....	0.49	0.64
	Total.....	12.07	20.01

Note. 75% dynamite cost 30¢ per lb del'd at mine. No boarding house expense appears separately, as this was charged directly to the different operations each month and is therefore included in labor charges. General operation includes supplies, wages of teamsters, blacksmith, and other expense not directly chargeable to either pit. This expense was equally divided between the pits at end of season, except for such supplies as were directly charged to either pit. General expense includes traveling, office, and other incidentals, not directly connected with actual operation. Ditch maintenance includes all expense on ditches for repairs, etc; but not the wages of ditch-tender, as this was charged to No 1 pit. Royalties, rentals, etc, include Gov't tax on gold. Last 3 items were divided between the pits at end of season. Above costs do not include charges for redemption of capital or deprec of plant. Much more detail is given by Hughes, Bib (411).

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long, beginning with 26-in, No 16 gage pipe, ending with 16-in, No 14 gage. 1 150 miner's in. is used in No 6 giants with 7-in nozzles. An added 1 o flowing down the creek bed is utilized as bank water. Besides the mai 16-in line, tapering to 6 in in a length of 432 ft, is taken out of the pressure on a 25-in Cassel wheel to drive an 8 by 8-in compressor; boulders are drilled with plugger-drills. Sluices are 42 in wide and deep, 4 000 ft long, grade, 4 in to 12 ft (2.77%). The development and operation of this mine were difficult because: (*a*) The deposit is too wide to mine in a single pit; hence 2 pits were worked (Fig 588), the face of *A* being kept about 400 ft ahead of that in *B*. Hydraulicking proceeds in one, while 2 men are blockholing and blasting boulders in the

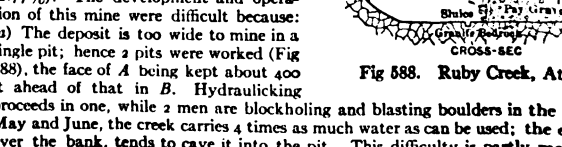

Fig 588. Ruby Creek, At

May and June, the creek carries 4 times as much water as can be used; the over the bank, tends to cave it into the pit. This difficulty is partly me sluices beginning opposite pit *B*, one being used to carry off excess wate

ut continues in *B* except for work on boulders. (*c*) As oom is limited. Sluices were first paved with block riffles, es (Art 124), and the grade flattened to 2.77%, which increased the sluice capacity. (*d*) The gravel contains re blasted in 2 yr, an aver of 1 per 2.5 cu yd of gravel lders are bulldozed, the larger blockholed and all pieces 1915, explosives cost 26¢ per boulder. In one place there is loosened by gopher blasts in the underlying gravel for 1915 (working season, about 150 days): aver men, 20;

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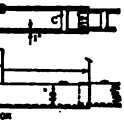
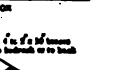
Table 98. Cost of Hydraulicking, 1914

Yukon Gold Co	Cost per cu yd
Fixed salaries	$0.001
Labor	0.023
Supplies	0.003
Shop expense (repairs)	0.001
Preliminary expense (*a*)	0.006
Bullion charges (*b*)	0.005
General charges	0.007
Deprec on distributing pipe lines	0.009
Stables	0.001
Transportation (*c*)	0.001
Main ditch operation (*d*)	0.069
Total operating cost	$0.126

(*a*) Charges during closed season. (*b*) Mint refining and government gold tax. (*c*) Skilled labor. (*d*) Water, including deprec.

r height of banks, 37.4 ft. Aver cost of 4 and of 3 opera- for hydraulicking was 65¢ and 93¢ respectively.
-scale hydraulicking on Mastodon Cr, Circle district, in ft; from 2 to 4 ft of coarse slabby bedrock is also re- water used under a 100-ft head. Equipment included cking tailing. Sluice, 30 in wide, 72 ft long, paved with cu yd per miner's 20-hr in, in partly frozen gravel. Pit be ground-sluiced before piping is necessary. Bedrock evicing necessary. Large rocks are handled in barrows.





When water is plentiful, the day shift consists of 3 nozzle men and 2 rock wheelers; night shift, 3 nozzle men and 1 rock wheeler (part time); aver washed per 2 shifts, 600 cu yd, at cost of 12¢. When water is low, 3 men work each shift; they wheel rock and make repairs while waiting for the reservoirs to fill; with this intermittent operation, aver cost was 14¢ to 17¢; with water at the lowest stage, cost rose to 21¢ per cu yd.

122. Sluices

Construction. Sluices are made in sections, usually 12 ft long; each section is called a SLUICE BOX. Fig 589 shows typical sluice box for shoveling-in and other small work (Art 120).

ices

12, 14 or 16-in boards, rough or planed on one side; the l than the other, so that adjacent boxes will telescope a few

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inches into each other. Sluices may rest on bedrock, or be set in ditches, c Sluices are braced laterally by inclined struts *B*. On the side of the sluice shoveler is a board *S*, against which the gravel is thrown, instead of being sho into the sluice; this increases the duty of labor and prevents spilling. A placed at the junction of each 2 boxes (Fig 590), thus eliminating cross-brac which obstruct top of sluice and interfere slightly with cleaning-up. Small sl be made with flush joints, like a flume (Sec 38); some operators say this reduce

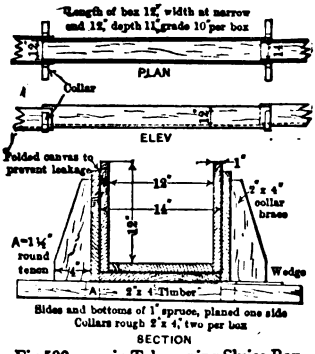

Fig 590. 14-in Telescoping Sluice Box, Fairbanks, Alaska (after Ellis)

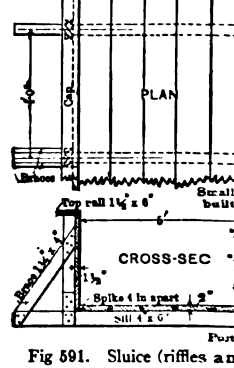

Fig 591. Sluice (riffles an omitted)

telescoping box is most used because it is easily erected and moved (414). have a short life, but are cheap and well suited to small work. Worn out b burned and the ashes rocked or sluiced to recover gold which lodges in crac

Fig 591 shows large sluices, for hydraulicking; details vary widely

4 by 4-in sills suffice for sluices up to 4 ft wide; 4 by 6-in sills and 4 by posts for wider ones. Sills are 3 to 4 ft apart; every 3rd or 4th post is supp

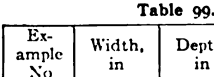

Fig 592. Tightener for Bottom Boards (after Purington)

braces. Posts are halved into sills, or fasten (post straps), or toe-nailed (Fig 591). Lumb surfaced, but for sides and bottom should be fr To prevent loss of quicksilver and amalgam, must be made tight by using tongue and groo by outside battens; close fitting alone is ofter Fig 592 shows a useful device for tightening b before spiking (390). Sills must rest on a solid foundation to prevent settle bottom of a new sluice is likely to rise by water collecting under it, the ends be weighted down (391, 406).

Table 99. Details of Sluices for Hydraulicking

Example No	Width, in	Depth, in	Dimensions of		Thick
			Sills	Posts	Bottom, in
1	24	.....	4″×6″	4″×6″	1.5
2	24	20	2″×2″	2″×2″	2.0
3	29	38.5	4″×6″×6′	4″×4″×3′ 4″	1.5
4	48	58	6″×6″×5′	6″×6″×5′	2.0
5	72	32	4″×6″×8′	4″×6″×3′ 2″	2.0
6	60	32	4″×6″×7′	4″×6″×3′ 2″	1.5
7	48	32	4″×6″×7′	4″×4″×3′ 2″	1.5
8	54	54	4″×6″	4″×4″	1.5
9	60	54	See note below		1.5
10	32	24	3″×4″	3″×4″	1.5

tlin, B C (390). Heavy gravel, 400 miner's in of water; . Block riffles used at head of sluice, followed by rails. difficult to obtain locally boards wider than 8 in. Sluice Wages, $3 per 10-hr shift and board.

nel" bench gravels, Klondike (390). 250 miner's in of r 24 hr. 12 to 14 sluice boxes usually constitute a string. and 9 in sq are used; cost, 25¢ each and last 1 season.

in, B C (390). 2 sluices, 600 and 700 ft long. Top rail, 1.5 1.25 by 2 by 4 in; side lining boards, 1.5 by 8 in by 12 ft. d for riffles; riffle strips, 1 by 3 by 28 in; braces, 1.5 by 4 in es, etc, contains about 540 bd ft of lumber and costs about M; riffle blocks, $6 per box-length.

in, Alaska (390). Sluice is in a 9 by 10-ft tunnel, 3 300 ft ; 000 cu yd gravel per 24 hr. All surfaced lumber. Riffles locks, separated by 1⅞ by 2-in riffle strips. Lining boards, by 8 in. Each box contains about 1 100 bd ft of lumber t per box, $30, of which $10 is for labor. Annual mainte- of riffle blocks (life 2 yr) and lining boards, $1 000.

ld tunnel sluice, Cal (391). 30d nails used for the bottoms, lining was of worn blocks, 3 in thick and 18 to 20 in deep. Block riffles, 20.5 in sq and 13 in deep, riffle strips, 1.25 by er of 19 blocks per 12-ft box. A flume for seepage water n deep, of 1.5-in plank, was built along one side of sluice.

n, Cal (391). Boxes, 14 ft long. Top rails, 2 by 7 in by were set in grooves between bottom planks. Side lining by 20 in sq. Braces, 1.5 by 4 in by 2 ft. 27 blocks, 17 in es in 1 box; riffle strips, 1.25 by 3 in by 5 ft. Cost per box, lumber and side lining @ $20 per M, $13; 704 bd ft blocks nails @ 5¢, $1; labor, $2 to $3 per day, $7.

ne, Cal; sluice used in 1880 (391). Boxes, 16 ft long, with p rails, 1.5 by 8 in by 16 ft; braces, 1 by 6 in by 3 ft; 36 d in each box; riffle strips, 1.75 by 2 in by 4 ft; side lining, . 420 bd ft of lumber required per 16-ft box, exclusive of er box, as follows:

No	Size	Place used	No	Size
12	10d	Braces to posts and sills.....	160	12d
40	20d	Sides to bottoms.....	76	20d
36	20d	Sides to posts.....	96	20d
50	30d	Top rails to posts and sides .	50	30d

lows: 420 ft lumber @ $30 per M, $12.60; 36 blocks @ @ 4¼¢, $0.64; labor @ $1 to $2.50, $2.50.

Trinity Co, Cal (415). 600 to 700 miner's in wash 500 cu nd braces are bolted together; sills are mounted on skids nd beveled at each end. Boxes are moved over bedrock to lock riffles are 11 in or 18 in sq by 10 in deep; riffle strips, out $20; the company has a sawmill.

ine, Trinity Co, Cal (415). Sluice is laid in an 8 by 8-ft es are of 1.5 by 6-in, 1.5 by 4-in, and 4 by 5-in pieces. Riffle 11 in to 18 by 18 in sq. Side liners of 2 by 10 and 2 by 6-in sticks, 2 by 2 in by 5 ft. 500 bd ft of lumber used per box ; 525 bd ft for blocks. Cost per box: lumber, $16; blocks, ncludes nails and labor for handling and placing.

ine, Siskiyou Co, Cal (416). Sluice is at head of an hydrau- t c-c. 12 in sq blocks used for riffles, also rails and trans-

efly at the head of hydraulic elevators (Art 125). They are stance and are easily bolted together; their cost is prohibitive for dredges (Art 126) are an exception).

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Curves should be avoided if possible; they increase friction a duce velocity of the water, and if sharp cause splashing.

Sluices are curved by making small deflections at successive joints. says a 5-in "swing" is the max permissible for a 12-ft box; if a sharper tur use a 4-in swing on a 6-ft box, or 3.5-in on a 4-ft box. Curves should be ends, and the outer edge of the boxes elevated from ⅛ to ⅜ in per ft o Grade of the sluice should also be steepened at curves; an increase of as is desirable on short turns. Bowie (391) gives further data on curves, rela and turn-out sluices. These work well on steep grades, but require car flat grades at connections with main sluice; otherwise gravel collects a

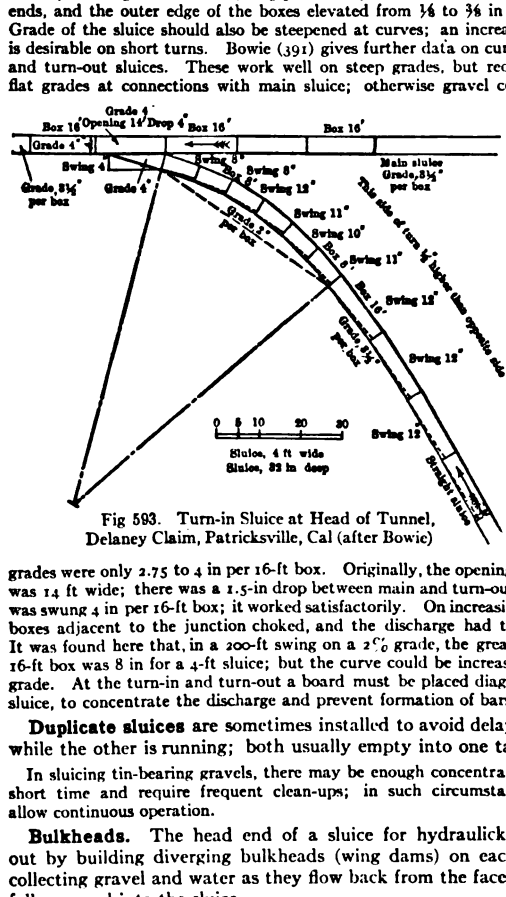

Fig 593. Turn-in Sluice at Head of Tunnel, Delaney Claim, Patricksville, Cal (after Bowie)

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grades were only 2.75 to 4 in per 16-ft box. Originally, the opening at point was 14 ft wide; there was a 1.5-in drop between main and turn-out sluices, was swung 4 in per 16-ft box; it worked satisfactorily. On increasing the sw boxes adjacent to the junction choked, and the discharge had to be wid It was found here that, in a 200-ft swing on a 2% grade, the greatest poss 16-ft box was 8 in for a 4-ft sluice; but the curve could be increased in pro grade. At the turn-in and turn-out a board must be placed diagonally ac sluice, to concentrate the discharge and prevent formation of bars.

Duplicate sluices are sometimes installed to avoid delays; one while the other is running; both usually empty into one tail sluice

In sluicing tin-bearing gravels, there may be enough concentrate to fill short time and require frequent clean-ups; in such circumstances, du allow continuous operation.

Bulkheads. The head end of a sluice for hydraulicking is u out by building diverging bulkheads (wing dams) on each side; collecting gravel and water as they flow back from the face, and ass fallen gravel into the sluice.

H. L. Mead, in 1913, stated that the sluice mouth at La Grange, Cal, is 18 ft high; a heavy bulkhead of this height is built out from both sides generally work in pairs, first cutting down the bank and then swinging aro the fallen gravel toward the sluice and adding water to it; very heavy mate be washed into the sluice. The high bulkhead and high sides of the sluice allow gravel to build up and increase the grade; this, with the above meth starts material down the sluice with a high velocity and increases its capac

ed on small sluices in Alaska as a puddling or dump box, l. In it, sticky clay and mud are broken up and large stalled for the latter purpose alone in shoveling-in, if the ice fork effectively. Mud boxes have an apron 8 or 10 ft a slope of or derrick r scrapers pron and are set on 2 ft) than le, block, h of the 6 in wide ft and in ton (390) acity and ie cost of avel and ld-saving eir opera- l variable for some

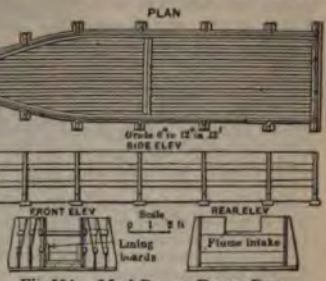

Fig 594. Mud Box or Dump Box (after Purington)

lf-dumping buckets are used, some other form of agitator r detail and description of a mechanical agitator (Siberian

y expressed in terms of the fall in inches per sluice not given, it is understood to be 12 ft. Thus, a 5 in per 12 ft. Grades should be uniform; flat r of the whole sluice; riffles and linings in steep and the flow must be stopped for replacing them. aver conditions, 6 to 7-in grades are satisfactory. p, grades as flat as 2.5 or 3 in per 12-ft box have nd earth. Flat grades reduce the duty of water of stones that can be sent through the sluice, and handling boulders. Lack of grade may be com- increasing the amount of water; conversely, steep water supply. Grades over 13 to 14 in per 12 ft d by topography, as the resulting velocity of flow rse gold.

es worked on grades of 10 to 24 in per 12 ft. Water was e sluiced (391). See Table 91.

luice grades depend largely on the kind of gravel; steep flat, sub-angular, or cemented gravel, and when much des may be flatter, and a larger duty is obtained with a ter, when sluicing a mixture of large and small gravel. s stir up fines and keep them in suspension.

l be sufficient to disintegrate the gravel and free his is accomplished in 100 to 300 ft. Crude shov- often use only 3 to 6 boxes; lengths, 36 to 72 ft to save very fine gold. DROPS (vertical falls) may aphy allows) to break up cemented gravels and

a sluice so long as the yield from the lower boxes exceeds them. Velocity of flow largely determines the minimum es; hence, a greater length than is necessary to disinte- ort sluices with drops and undercurrents (Art 124) are than long sluices without them. Very long sluices may

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be needed to transport gravel to dumps; their lower parts are called TAIL-SLU examples of lengths see Tables 94, 99 and under Cost data, Art 122.

Cross-section of sluices is proportioned to amounts of water a and to the velocity of flow, which in turn depends on the grade. For first cost and wetted perimeter (hence for minimum frictional resis max discharge for a given cross-sec) the sluice width should be about depth of stream, as in flumes (Sec 38, Art 15); it may be impossible this ratio in large sluices. Depth of water should be sufficient to the largest stones. Width of sluice is adjusted to depth and grade, quired capacity. Fine gravel containing fine gold should have ver wide sluices, on steep grades (see Undercurrents (Art 124)). The ten riffles to pack increases with depth of stream, which is usually 6 to 1

To reduce first cost, and facilitate cleaning out and replacing riffles, th sides of a sluice should not exceed 1.3 to 1.5 times the depth of stream. Fo deep cuts in firm ground, sides may be lower than for those on or above the s surface sluices clog and overflow, much of the gravel in them above point must be shoveled out (406).

Water required. The only data available are given under Duty in hydraulicking, Art 122; these data are unsatisfactory, as the cutti ity of giant water may not equal its sluicing capacity.

The water used in a small sluice for shoveling-in (Fig 589) is known as a " sl irrespective of its quantity; it varies from 30 to 60 miner's in for sluices 10 to set on grades of 6 to 8 in per 12-ft box. In British colonies this term means cu ft of water per sec (394).

Calculations of sluice grades, cross-section, velocity of flow, etc, canno accurately, because the complex relationships between the factors are not wh and few experiments have been made to determine the necessary empirical

Veloc., ft per min	
15	Begins to wear away fine clay
30	Just lifts fine sand
40	Carries sand as coarse as linseed
60	Moves fine gravel
120	Moves pebbles of 1 in diam
180	" " of egg size

Most authorities suggest so cation of Chazy's formula, 11, as a basis. For exampl (394, 388, 417).

The author's calculatio Kutter's formula (Sec 38), sluices where all necessary available, show values fro 0.04. Calculations should against actual operations under similar conditions; Tables in Art 122, 123 a Geike gives the above data on velocity of streams required to move materia ent sizes (418). Somewhat smaller velocities suffice for sluices.

Van Wagenen (417) states that a veloc of 200 ft per min is necessary to move pebbles the size of an egg, 320 ft for stones of 3 to 4 in diam, 400 ft for boulders of 6 to 8 in diam, 600 ft for boulders 12 to 18 in diam. G. K. Gilbert's experiments on stream and flume traction are also suggestive, Bib (419).

(*a*) Depth of flow, 6 to 7 in. (*b*) Depth of flow, 10 in. (*c*) Height of side, in this and following examples, is 0.5 to 0.75 width of sluice; depth of flow, 0.33 to 0.5 of clear inside depth of sluice. Duty of water (Art 122) must be known or assumed in designing a sluice for a given amount of gravel.

Table 100. Capacity of Sluic

Sluice		Grade	Miner's in of
Width, in	H'ght of side, in	per 12-ft box, in	water
10 to 12	(*a*)	6	30
12 to 14	(*b*)	9	67
72	36	6 to 7	2 000 to 3 500
48	30	3	800 to 1 500
48	30	6	2 000
36	30	2.25	600 to 1 000
36	(*c*)	.....	200 to 600
48	.....	.....	400 to 1 200
60	.....	.....	1 000 to 2 500
72	.....	.....	2 000 to 4 000
96	.....	.....	3 000 to 5 000
120	.....	.....	4 000 to 7 000

calculations: Inches of fall per 12-ft box × 0.694 = % .521 = % grade; fall in ft per mile × 0.027 = fall per er × 0.025 = cu ft per sec; cu yd of gravel per 24 hr er sec. If D = duty of 1 miner's in, in cu yd of gravel of water and gravel is uniform, then 80 ÷ D = ratio ad cu ft of gravel per sec.

rrents, and Other Gold-saving Devices

tions: (*a*) to retard material moving over them ; (*b*) to form pockets to retain gold which settles which roughly classify (Sec 28) the material in the is not well understood. The strength and shape iffle) is affected by the shape and spacing of riffles, the direction of flow, and the velocity of current. gh to prevent the riffles from filling up with heavy trong to prevent the lodgment of gold.

especially for large scale hydraulicking, are stated by hey should oppose minimum resistance to the flow, in water. (*b*) They should resist wear, to reduce cost of n, and to preserve their gold-saving capacity. The ould be considered in the design and in the choice of ufficient in number to save all the gold commercially omy of a high cost material, as manganese or nickel material, as structural steel, may be determined thus: 's cost of 2 materials *a* and *b*, and *B* the cost of transport e of the cheaper material *a* is 1 year, that of the high equal costs at the end of N years, $X + B = N(A + B)$, equality exists, or if X is greater than $N(A + B) - B$, high-grade material. If X is less than $N(A + B) - B$, erial increases approx with N. Good material saves ost of labor and rn riffles (420).

s. Fig 595 for shoveling-

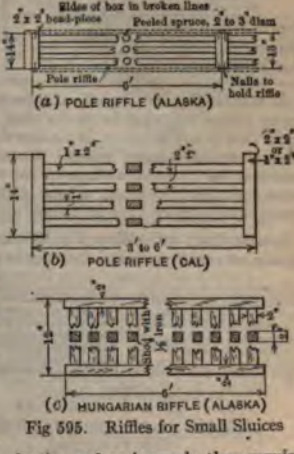

Fig 595. Riffles for Small Sluices

e in Alaska for , made of sawed ngarian") riffle ce, clogs more but is a better uice boxes, both nber and distri- perator's fancy. ails driven into he nails are not led when riffles ges may be used ome.

are of many ands descrip-

) are cheap and their life aver- d blocks (406). As the surface is rough, they require They are well adapted to tail sluices which are cleaned cult to take up and relay.

ere timber is cheap, they are often the most economical s they are quickly taken up and replaced. They make

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a smooth pavement on which stones may roll or slide, and work well. T
8 to 12 in deep, set in transverse rows separated by "riffle strips," and

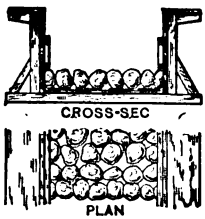

Fig 596. Cobble Riffles

that when worn they may be turned as requir
smoothest surface; the side of the square is a
of the sluice width. It is usual to set blocks in
to break joints, because longitudinal cracks e
and force earlier renewals.
At a few mines, better gold recovery is claimed for a pavement in which joints are broken as above, but longitudinal spaces are also left between the blocks of each row (390). Blocks are held in place by the side lining and riffle strips. Each strip is nailed to a row of blocks with headless wire nails, which are not driven home but project from 0.5 to 0.75 in. The adjacent blocks are driven against these nails until they rest solidly against the riffle strip. For dimensions see examples following Table 99.

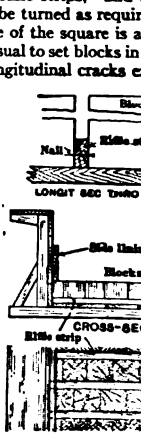

Fig 597. Bloc

The objection to block riffles is their rapid wear under heavy service; their life depends on quality of wood, sluice grade, character and quantity of gravel, and amount of water. Long-grained wood, which "brooms up," is best; hard wood is not desirable. On a given grade, the larger the ratio of water to gravel, the less the wear (391). Blocks worn to a thickness of 4 or 5 in are discarded, or used for lining the sides of sluices. Bowie gives followi early California mines (391) (see also Table 96, 97).

Locality	Width of sluice	Grade, in per 12 ft	Depth of blocks, in	Life of miner's
North Bloomfield.....	6	6.5	13	175 000
Manzanita mine.....	5	7.0	13	100 000
La Grange.....	4	3.0	8	100 000

At Manzanita mine, and also at French Corral mine where similar figures w poorer timber was used for blocks than at North Bloomfield. P. Bouery, i states that in 6-ft sluices at La Grange the life of 13-in blocks was 45 days to in the higher boxes lasted longest. On increasing the duty of water 40%, the decreased 60%, so that a clean-up was necessary every 17 days. Conseque expense, and limited supply of pine blocks, led to use of manganese steel rif Similar substitutions have been made at several large mines since 1910 (421

Longitudinal rail riffles (Fig 598) are of 20 to 40-lb RR rails, in leng usually set upside down and spaced 3 to 5 in or more apart by wooden or C-I

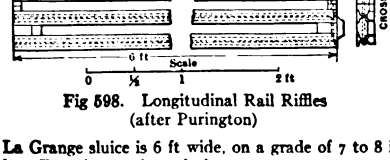

Fig 598. Longitudinal Rail Riffles (after Purington)

a given grade, rails will fines and more boulder (406); they are largely upper boxes of sluices, fairly well and are easi

Transverse rail riffle experiments at La Gra under Block riffles), pub (420), give valuable d in general for large hyd

La Grange sluice is 6 ft wide, on a grade of 7 to 8 in per 12 ft; depth of wa in. Experiment showed that transverse were superior to longitudinal riff or a sand particle may remain in contact with and wear the web, flange longit rail throughout its entire length, but wears only the top of a tra Longit rails 8 in apart wore faster than those spaced 5 in. 40-lb longit rails,

oo miner's in of water; 40-lb transverse rails, spaced 5-in spacing (c to c) proved best; different spacing ades or depths of water, but tween 4 and 6 in; spacing less greater than 6 in interferes rail was selected for the first considered too light; 55-lb in proportion to its head; he most economical. Rails were set on a series of 2 blocks (Fig 599); rails alone do not form sufficiently might wear the sluice bottom. This system proved more efficient for catching gold than the previous riffles, and the gravel never packed hard between the 7 spacing blocks had to be set between each pair of (Fig 600). Wear is greater in center than at sides ice, and rails bend before wearing out, unless rigidly rted. By using 7 spacing blocks rails could be used they were 62.5% worn, instead of 37.5%, as was the with 3 or 4 spacers. Ends of worn rails, utilized to ct the sides of sluices (Fig 601), were held in place by bent over the flange of the top rail, and by a 2 by 6-in . Side rails did not increase the duty of water mater- but decreased cost of replacing the wooden blocks rly used; they last 5 to 7 years; duty, about 30 n cu yd of gravel.

Fig 599

nd alloy steels, showed manganese steel to be the riffle were developed, aiming to lengthen their life bows a manganese steel all the requisite quali- low wear, good setting, f gold. Some of them ter and from 12 to 15 were discarded. For bution of gold in sluices,

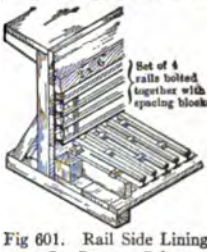

Fig 601. Rail Side Lining, La Grange, Cal

with iron straps, sim- y 4-in or larger scant- not available. They gold-saving, but have There are many forms,

e) are of the Hungarian type. They have been used (Art 125), on tables and tail sluices of dredges (Art) and sometimes for small opencut work (Table 92) hydraulicking (Table 94). Size of angles varies from by 1½ by ¼ in for small sized material, up to 2½ by by ⅜ in. Fig 603 shows one method of holding les in place. They are set with the vertical leg on the tream side; clear space between them, 0.25 to 3 in. se spacing gives a weak eddy and the "dead water" ce is said to be a good fine gold-saver, assuming that current velocity allows such gold to settle on the es. T-iron may be used instead of angles.

ast-iron grate riffle (Fig 604) has done good work in ll sluices on Seward Peninsula, Alaska. It is light, t the long dimension of the slots is either transverse thought best (390).

ffected important economics at several large mines, s for tail-sluices. They reduce friction, compared ts, and are especially useful on limited grades; se

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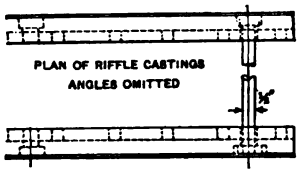
Ruby Creek, Art 122. The steel contains 0.8 to 1.2% carbon; in some c
that the outside skin of the plate is more resistant to wear than the in
are commonly 0.5 in thick, of
sluice and of various lengths.
modes of supporting plates 12 f
space and a 0.5-in drop is left
plate. Supports *T* are 6 to 8-in
sawed flat on 2 sides and taperec
upper end to 4 in at downstrea
are held down by 3 by 10-in lining
made a good riffle and increased t
ity about 40%; a
boulders 30 in long c
as against 20-in pie
riffle (412). At V
plates 20 by 30 in a
in cross joists; the
apart; no drop is us
Quesnelle Co, B C,
sq in a 6-ft sluice, se
in space and a 0.5-in
ber of boxes at hea
paved with manga
riffles which extrac
gold (421). See T



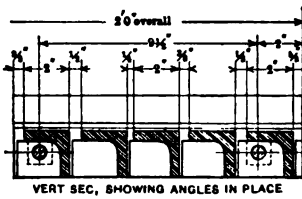


Fig 603. Riffle Castings for 2 by 2-in Angles, 5/16 or 3/8 in thick

For further data on riffles for handling fine material, see Undercurrer
Dredging (Art 126).

Side linings are required to protect the sides of large sluices; also they often furnish a means of holding riffles in place.

Plank liners are common (Fig 597), 1-in for smaller sluices, 2 to 3-in for wide sluices with a large head of water. Where block riffles are used, worn blocks are placed on the sides; they serve for tail sluices, but are inconvenient where frequent clean-ups are necessary. See Fig 601 for rail linings. Height of side linings need not be more than 2 or 3 in above normal level of the stream of water and gravel.

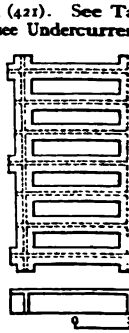


Fig. 604 (after

Undercurrents are intended to save fine gold which will not settle in the sluice. Fig 606 shows the North Bloomfield undercurrent, fairly typical for large Cal sluices; it is practically a wide sluice, set on a heavy grade at one side of and below main sluice.

Bowie gives following details: Across the main sluice, at some point can be made, is placed a grizzly, over which coarse material and some of th
the undersize falls into a spill box and runs
a distributing sluice (grade 2 to 3%) to th
proper. The latter is a shallow box, 20 to
40 to 50 ft long, divided into sections by
tions for convenience in placing riffles and
and to allow better control of the distributi
Undercurrents are paved with small woode
bles, or pole riffles shod with iron; grades rec
riffles are 14, 16 and 12 in per 12 ft, resp
pending on location of the undercurrent,
water are discharged directly or led back to the main sluice, which extends
at which the undercurrent is erected. Width of undercurrent, 8 to 10 ×
sluice. A wide undercurrent costs slightly more to clean up than a narrow o
more efficient; at French Corral, with a 5-ft sluice, the yield of the first unde
wide) was 20% of yield of all; 10 ft added width increased the yield to 27%

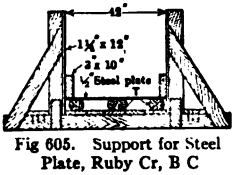
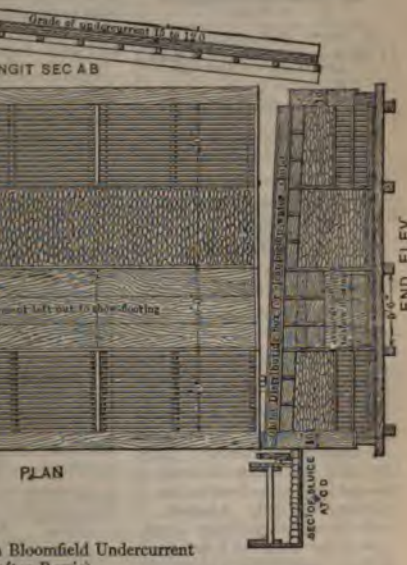


Fig 605. Support for Steel Plate, Ruby Cr, B C

eral type vary widely in detail. Grizzlies in sluices lb rail, 0.75-in round or octagonal steel, or steel flats; a. In large sluices, grizzlies are usually of V-shaped upported that they will rock under the impact of ngs vary in width to 2 in. Grizzlies are placed on eper than the sluice grade. Any device that will grizzly is desirable, as it gives sand and gravel a ercurrent.

, or of wood shod with iron, are also used for under- carpet tufted with chicken wire, cocoa matting with



Bloomfield Undercurrent
after Bowie)

e employed at different mines. The ideal location for is near the dump, so that large material need not be his is not always feasible, and several other undercur- g a sluice. A minimum fall of about 5 ft is required s to the sluice; the sluice grade can therefore be steep- which the undercurrent water reënters. For further , 424). Though the above undercurrents remove large some sand and gold pass over the grizzly with oversize. partly overcome by a different type of undercurrent. troduced at Fairbanks, Alaska, in 1914. It greatly compared with the old type, but it was necessary to e its efficiency decreased greatly thereafter. 2 boxes yd of gravel per day) were fitted with this device; one or cocoa matting being washed in a tub. On a claim is undercurrent recovered 20% of the entire clean-up, a profit (414). Fig 607 is a modified Caribou riffle or

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undercurrent (Fig 608), regarding which Purington says (390): Hungarian riff
or iron, carpets, plush or blankets, are placed in the sluice bottom under the scre
ing on character of the fine gold. This device affords cheapness, flexibility, and
but does not obviate the need of puddling clayey gravel treated in small sluices

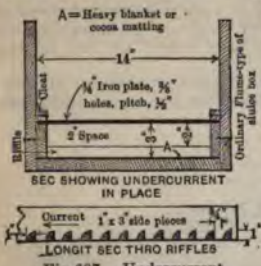

Fig 607. Undercurrent (after Ellis)

box, Art 123). Experiments with punche
New Zealand showed that holes less than
give too small a discharge and are apt to cl
holes passed too much water and material.
sults were obtained in treating marine grav
ing fine shingle (425). The proper size of h
locality is a matter of simple experiment.
River, Idaho, a different form of Caribou ri
dercurrent has been used, in which the heavi
of the concentrates is drawn off continuou
holes in the sluice bottom. This allows th
of black sand containing valuable minerals
gold which require further treatment (426).

Operators disagree as to the use and
undercurrents. Where the gold is fine t
of evidence is in their favor, if they are
designed and cleaned up often enough to do efficient work. At La Ge
after riffles like those in Fig 602 had been installed, undercurrents were
as they clogged and the gold recovered did not pay cost of cleaning up

Use of mercury in riffles and undercurrents is quite general; an
easier to handle in clean-ups than fine gold; some gold, otherwise lost,
saved by amalgamation. Where much coarse gold occurs, mercury i
in a few boxes at the head of the sluice, to avoid the trouble and cost of retorting coarse gold amalgam.

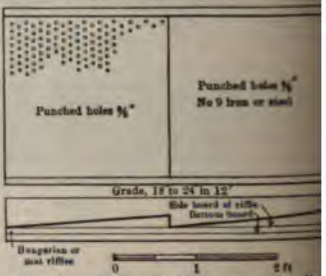

Fig 608. Caribou Undercurrent (after Pu

Sluicing begins with a small head of water, until all cracks and leaks are stopped. The water is then shut off and the riffles in the upper part of sluice and undercurrents are charged with mercury. More is added from time to time, as needed to keep a clean surface of mercury exposed. Amounts of mercury used vary widely; a common initial charge for large sluices is 2 to 3 flasks (76.5 lb each); for large undercurrents, 80 to 160 lb. Mercury must be clean and in charging all or spattering should be avoided; otherwise minute globules are formed which Loss of mercury is inevitable; it varies from 4 or 5% of the total amount us under poor conditions, averaging perhaps 10 to 15%; loss is least in well-l sluices, provided with undercurrents (391, 394).

Cleaning-up consists in removing the riffles and collecting the amalgam. Interval between clean-ups is made as long as possible, the delay they cause; often the lower parts of a sluice are cleaned once in a season. Clean-ups are required when worn riffles must be and when much gold has collected, as in the first few boxes of a sluice rich gravel; danger of theft often influences the frequency of clean-up

Cleaning-up begins at the sluice head, by removing a few riffles and turning flow of water. The concentrates are worked over with shovels or scoops, a slowly down the sluice bottom, the gold and amalgam lags behind the black and are scooped up into pans or buckets. The process is repeated in the section. See Bib (414, 391, 398).

luices. Most of the gold is caught near the
that an average of 80% of the total yield of
e first 200 ft of length, according to results at
In Alaskan shoveling-in or small open-cut work,
oxes below it retain most of the gold.

periments by Bouery, at La Grange, Cal, in 1910, to
different sizes; the tests lasted 15 days (420). Value

f Gold in Sluice Boxes, La Grange, Cal

ing test on gold recovered in different boxes

	Mesh, per cent				
−10+50	−50+100	−100+150	−150+200	−200
50.7	1.4	0.4	0.3	1.4
78.8	2.2	0.9	0.3	0.8
76.1	11.6	2.6	0.9	2.3
58.1	28.3	3.2	1.3	4.3
18.6	72.6	1.2	4.0	0.8
36.8	18.8	29.9	7.6	6.9

3, 75, 108, 101, 100, 53, 46, 43 oz in these boxes.

e head boxes to $6 in box 136. Head boxes did not
re often blocked by boulders and sand (Bulkheads,
t accurate determination of the point beyond which
onstruction and maintenance of added sluice length.

not be accurately determined. The gold con-
t known exactly, and there is no practical way
during operation.

placers, as at Oroville, Cal, ample opportunity is
omputed from churn-drill exploration (Sec 25) with
mpirical factors have been developed for discounting
represent a combination of errors in sampling and
d can not be used in other localities. Drill and shaft
ers often give results widely different from recoveries
simple open-cut methods (447).

skan shoveling-in recovers 80 to 90% of the total
es place the recovery in hydraulic mining with
ents at 60 to 85%; obviously it must vary with
that of the gold-saving devices.

al ranging from large stones to fine sand. The veloc-
nt to sweep the coarse material along. Gold is sep-
te by the classifying action of irregular eddies set up
uired to prevent riffles from packing. These condi-
low of water in hydraulicking and intermittent clean-
e flaky or spongy gold and are responsible for losses.
if properly cared for, but no device is known that
hrough a stationary grizzly on the sluice bottom and
versize on. It is important to note that nearly all
e a very low gold content, and that in spite of losses
robably make nearly the max profit, in view of any
nowledge of the art. Recoveries compare favorably
treating low grade ores (as the porphyry coppers) by
materials can be closely sized, classified and treated
ssion does not apply to small gravel containing very
er, Idaho, and in certain beach sands), nor to gravels
s metallic gold, which present problems still unsolved
information, see Bib (390, 427, 424, 382).

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125. Elevators

Elevators are employed to secure artificial dumproom at placers lacking ties for the disposal of tailing, and where conditions prohibit dredging.

Hydraulic elevators (Fig 609–611) are often used in connecti draulic mining. The excavated gravel, with the giant water and conducted in ground- or box-sluices to the foo vator, which delivers it to a sluice on the surfa the dump (385, 394, 390, 397). The surface slu short, with undercurre OF USE is i lacking s and dum having a cheap wate sure. De

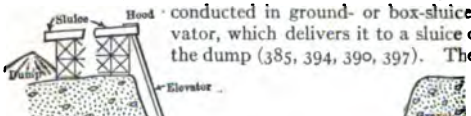

Fig 609. Elevator Pit (diagrammatic vert sec)

worked range from a few ft to 90 ft; depths of 20 to 25 ft are favor allow considerable gravel to be mined from 1 set-up of the eleva excessive lift. Boulders (or buried stumps and timber) too large to the elevator increase the operating cost, as they must be blasted o derricks; if too numerous, they may prohibit elevator work. The h vator is inefficient and wasteful, but useful where the necessary water is available. Its CONSTRUCTION and OPERATION are shown by Fig 610.

1 Flange of press pipe
2 Swivel elbows
4 Flanges
6 Nozzle butt
9 Ground section
10 Entrance section
11 Throat section
12 Outer section
13 Upcast pipe flange

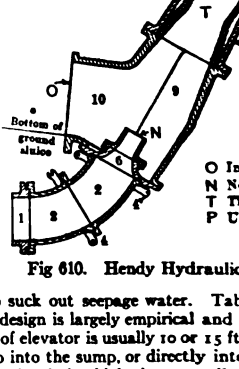

Fig 610. Hendy Hydraulic

Water under pressure is discharged upwards through nozzle *N*; in passing through the restricted throat area, *T*, it sucks in water, gravel, and air through intake opening *O*. The force of the jet elevates the aërated column of water and gravel through upcast pipe *P*, the upper end of which projects through the bottom of the first box of the surface sluice. To prevent spattering, this box is covered and is provided with a liner ("hood") which diverts the discharge laterally. The hood and throat take the most severe wear, and are usually of manganese steel; special steels may also be used for other parts. The upcast may be a riveted or welded steel pipe, with slip or flanged joints, depending chiefly on height of lift. The Evans elevator has an auxiliary suction opening on each side of the main intake, to allow air to enter the elevator if the main intake clogs; they may also be extended by pipes to low places on bedrock and used to suck out seepage water. Tab size and weight of elevators; data for their design is largely empirical and published by the makers. The SUMP at foot of elevator is usually 10 or 15 ft 4 ft deep (385). Bedrock sluices may dump into the sump, or directly int intake; the latter plan is best, as the suction head should be kept small. usually has a slope of 60° to 70°; the makers of the Evans elevator state best at 80°. Elevators are sometimes set vertically, or on slopes as flat as

Water required. The elevator nozzle water will lift from 0.5 to 1 × its of outside water. Hence, 50 to 66% of the pressure water available is as elevator, and the rest to the giants; the higher figure is conservative (no

ic Elevators (From makers' catalogs)

f	Hendy elevators (*e*)		
Nozzle, in	Diam of upcast, in	Approx shipping wt, lb (*d*)	Wt of heaviest piece, lb
2–2.5	8	2 200	350
2.5–3			
3–4.5	10	2 700	450
3.25–5	12	3 200	650
4.5–5.5	14	3 750	825
5–6	16	4 600	1 000
1.5–3	18	5 800	1 200
2.5–4	20	7 600	1 450
2.5–5	24	9 600	1 800
6–8	30	14 000	2 400

lso made by the N Y Engineering Co, N Y City. McCone, San Francisco, Cal. (*b*) By Union Iron weight from 2 550 to 7 500 lb. Some sectional eleva- ndy Iron Works Co, San Francisco. Size of throat iout upraise pipe and suction attachment.

of water between elevator and giants is controlled Flow through nozzles, Art 122).

mical operation is about 17% of the effective head at oportionately more effective than low ones; the lift is

103. Water Capacity of Hendy Elevators*

'otal water *a*), miner's in	Suitable for use with 1 giant No — (*b*)	Approx width of sluices (*c*), in		
		Bedrock sluice		Surface, all wood (*f*)
		Steel (*d*)	Wood (*e*)	
133– 266	1	10–12	12–18	24–30
216– 430	1	10–12	12–18	30–36
300– 630	2	12–14	14–18	30–36
466– 866	3	12–16	16–24	32–42
750–1 150	4	16–24	20–30	40–60
866–1 400	4	18–26	24–36	42–60
070–1 800	5 or 6	24–36	30–42	48–66
630–2 530	7	36–42	48–60	60–72
530–4 000	Two No 6	48–60	54–66	72–84

catalog of Joshua Hendy Iron Works Co, San Fran- Total water = giant, seepage and elevator water. or handling gravel can be estimated if duty of s known. For lifts of 10 to 100 ft. (*b*) See Table 90, c) Depth of small bedrock sluices = ⅔ the width; of ⅓ the width. Depth of surface sluices = ⅔ the No riffles. (*e*) Wood block riffles. (*f*) Allowance ffles.

throat diam. To prevent serious clogs, a grizzly is rs are spaced at least 1 in closer than the throat diam.

gravel forms at most 5% (usually 2–3%) of levator; hence, in estimating capacities, only

city in a given case varies with the ratio of pressure olume, regularity of flow of gravel and water to the

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elevator, and also with the giant water duty. The latter is a major factor in de
ing the size of elevator. If there is much seepage water, which reduces the
available for elevating giant water, a WATER LIFTER (practically a small elevat
a suction pipe instead of an open intake) is often installed to drain the pit.

Table 104. Summary of Hydraulic Elevator Work, Wild Goose C

Line No*			1908	1909	
1		Area, sq ft	306 480	465 300	320
2		Aver depth worked, ft	10.7	8.7	
3		Material handled, cu yd	121 960	150 637	95
	Operating data	Number of pits working	12(*a*)	19(*a*)	
6		Piping: Number of days	123.8	186.2	11
7		Sq ft per day	2480	2 500	2
8		Cu yd per day	986	809	
11		Cleaning bedrock, days	81.5	84.0	2
14		Time lost, days	79.9	73.9(*b*)	1
15		Total running time, days	285.2	285.6	9
16		Sluiced per day, sq ft (*m*)	1 075	1 630	3
17		Sluiced per day, cu yd (*m*)	428	528	1
22	Water used	Elevator water, in-days	63 082	74 791	54
36		Giant water, in-days	18 368	26 788(*l*)	27
40		Pump water, in-days	6 000	4 200	
41		Other water, in-days	.....	3 188	3
42		Total water, in-days	90 570(*f*)	113 814(*g*)	87
43	Water duty	Sq ft per in-day	3.4	4.1	
44		Cu yd per in-day	1.35	1.32	1
45		Giant water, sq ft per in-day	16.7	17.4	1
46		" " cu yd per in-day	6.6	5.6	
	Costs, cts per sq ft	Labor and mess:			
47		Digging sump, erecting elevators	.....	1.39	0
48		Ground sluicing	.....	0.03	0
49		Piping	.....	3.00	2
50		Cleaning bedrock	.....	1.61	0
51		Dismantling	.....	0.20	0
52		Total	7.42(*i*)	6.23	5
53		Stable	0.65	0.41	0
54		Merchandise	0.35(*j*)	0.37	0
55		Water (*k*)	2.70	1.75	1
56		Main pipe installation	1.34	1.23	0
57		Turning creek	(*i*)	0.18	0
58		General work	0.57	0.79	0
59		Prospecting	.....	0.55	0
60		Melting and transporting bullion	0.25	0.28	0
61		Management and office	3.47	1.67	1
62		Winter stable	0.62	0.22	0
63		Deprec on equipment	1.04	0.79	0
64		Working cost, cts per sq ft	18.41	14.47	11
65		" " " " cu yd	46.20	44.8	39

* Numbers correspond with same lines in Table 105. (*a*) 2 more pits started, 1
completed. (*b*) Short water, 15.5 days; no water, 53 days; other delays, 5.4
(*c*) Excluding time digging sumps. (*d*) Included in line 41. (*e*) Includes wa
pump and for handling tailing. (*f*) Includes 3 120 in-days not distributed abov
Includes 4 847 in-days not distributed above. (*h*) Includes 2 450 in-days not
uted above. (*i*) Includes labor and mess for turning creek. (*j*) Includes equ
0.097¢. (*k*) Water charge, cts per in-day: 1908, 9.46¢; 1909, 7.52¢; 1910,
(*l*) 68.6% of total. (*m*) Total running time.

is expressed by: $E = H\,(62.4\,W + S) \div 62.4$ fic; H = height of lift, ft; H_1 = effective head n of giant, seepage, and bank water; S = wt of er gravel normally weighs about 3 000 lb per cu rough elevator nozzle, cu ft per min (397). Effi- to 20%. Longridge cites 12 elevators in N Z, ft with water under heads of 200 to 448 ft, the gravel) was 20 to 33% (394).

ir Cr, Council District, Alaska. W. H. Lanagan and Tables 104, 105, which are from detailed records.

4 Hydraulic Elevator Pits, Wild Goose Co

	Pit No 1	Pit No 2	Pit No 3	Pit No 4
.....	23 800	67 200	24 100	16 800
.....	6.4	8.5	16.0	11.1
.....	5 640	21 100	14 320	6 010
el, bedrock..	(a)	(b)	(c)	(d)
.....	(e)	None	None	None
ta				
.....	8.0	20.0	16.0	7.5
.....	2 980	3 840	1 510	2 240
.....	705	1 055	896	802
od.....	(a)	Giant	(f)	(g)
.....	13 175	.....	.....	11 600
.....	5.0	.....	19.0	6.0
.....	176	.....	.....	99
.....	5.9	.....	.....	6.1
.....	None	1(h)	None	2(i)
s (j).....	14	22	36	17
(w).....	1 700	3 055	670	1 120
e).....	403	960	398	400
.....	(l)	.....	(m)	(n)
zzle, in.....	3.5	3.5	3.5	4.0
.....	13.5	19.5	34.7	15.1
(o).....	302	300	280	360
).....	4 075	5 850	9 720	5 440
.....	.....	215*	.....	197
.....	22	20	27	28.5
ated.....	.....	4.7	5.0	0.5
(o).....	.....	210	203	196
).....	.....	987	1 015	98
rated.....	4.5	13.5	8.0	6.8
(o).....	150	145	141	139
).....	675	1 957	1 128	945
ated.....	1.8	5.7	6.0	0.4
ches (o).....	96	90	90	89
(p).....	173	513	540	36
ys (p).....	848	3 457	2 683	1 079
.....	2	2	None	2
.....	4.2	2.2	.....	1.0
(o).....	80	80	.....	80
).....	336	176	.....	80
.....	420(q)	.....	380(r)	59(n)
.....	5 679	9 483	14 383(t)	7 358(s)

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Table 105. Work in 4 Hydraulic Elevator Pits, Wild Goose Co (Co

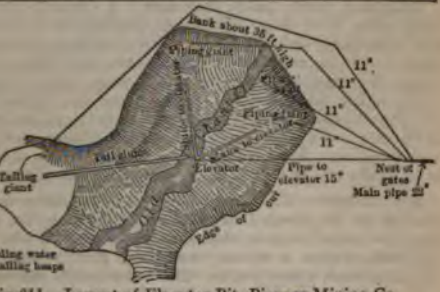
Line		Pit No 1	Pit No 2	Pit No
	Duty of water			
43	Sq ft per in-day	4.2	7.1	1.7
44	Cu yd per in-day	1.0	2.2	1.0
45	Giant water, sq ft per in-day	28.1	19.4	9.0
46	" " cu yd per in-day	6.7	6.1	5.3
	Costs, cts per sq ft			
	Labor and mess:			
47	Digging sumps, erecting elevators	0.90	0.75	1.17
48	Ground sluicing	.....	.....	.....
49	Piping	2.26	1.57	6.24
50	Cleaning bedrock	2.83	Giant	12.00(c)
51	Dismantling	0.16	0.07	0.14
52	Total	6.15	2.39	19.55
53	Stable	0.17	0.08	0.10
54	Merchandise	0.79	0.05	.....
55	Water @ 7.52¢ per miner's in	1.79	1.06	3.98
56	Main pipe installation	1.72	0.41	2.12
57	Turning creek	0.06	.....	.....
58	General work	0.79	0.79	0.79
59	Prospecting	0.93	.....	0.16
60	Melting and transporting bullion	0.32	0.29	0.55
61	Management and office	1.67	0.65	5.30
62	Winter stable	0.09	0.03	0.04
63	Deprec on equipment	0.88	0.49	2.23
64	Working cost, cts per sq ft	15.36	6.24	34.82
65	" " " " cu yd	64.7	19.9	58.6

* Estimated. (*a*) No frost; 1 to 2 ft soil overburden; small, clean gr 5 ft thick; bedrock, flat, slabby mica schist, too hard and irregular to clean 6 in to 1 ft taken up by hand. (*b*) Very little frost; 3.5 ft muck overbur small gravel; soft schist bedrock, taken up for about 2 ft; gold on bedroc frost; soil overburden, 4 ft deep; gravel, 8 to 11 ft deep; very irregular calca and lime bedrock, 4 200 cu yd taken up. (*d*) No frost; overburden is soil small, clean gravel; irregular calcareous schist and limestone bedrock, wit 3 to 6 ft deep. (*e*) Est'd flow, 90 in. (*f*) Cleaned by shoveling; aver depth taken up; some pot-holes 10 ft deep. (*g*) All bedrock cleaned by hand. (*h*) (*i*) Dam broke. (*j*) Exclusive of time building sump, taking 4 to 10 days of bedrock. (*l*) Not much trouble. (*m*) 2-in giant used 1 day, using (*n*) 2-in giant for 0.7 day, using 59 in-days. (*o*) 1 miner's in = 1.5 cu ft pe Number of days times rate in miner's in; 1 in-day = 2 160 cu ft. (*q*) Slu days = 420 in-days. (*r*) Sluice-box, 3.8 days = 380 in-days. (*s*) Includes not distributed above, probably bank water. (*t*) Includes 1 600 in-days not above, probably bank water. (*u*) At another pit with a mica schist and lin 3 days needed to clean 5 357 sq ft; duty 185 sq ft and 5.7 cu yd per man taken up for about 1 ft; part shoveled and part piped. (*v*) Campbell ele throat, 15-in upcast. (*u*) Per day total running time.

Pioneer Mining Co, Anvil Cr, Nome, Alaska. Data by C. W. Puringto 29 days' operation in 1912 (428). Gravel is coarse, subangular, unfrozen, medium size boulders; it occupies a bench channel roughly parallel to Anvil C Depth of gravel, 25–45 ft. Bedrock is hard laminated schist; no muck over gold occurs in upper 10 ft of gravel; bedrock gravel is rich. Crew, 18 me more than half of them employed in cleaning bedrock. Bedrock is uneven an with brushes after piping is finished, some of the dirt being wheeled to the sl rows. A team hauls boulders out of the pit; the giant water will not drive st in diam into the bedrock sluice leading to the elevator. Only 1 piping giant is a time. With an unriffled steel bedrock sluice, the face can be carried 150 to from the foot of the elevator in any direction. The record in Tables 106, 107 excavation of about 1 acre, averaging 25.5 ft deep. A giant stacks tailing at e

tor Work, Pioneer Mining Co, Anvil Cr, Nome, Alaska

	Hydraulic elevator	Piping giant	Tailing giant	Bedrock sluice
.....	39 789	39 789	21 047 (*a*)	.....
.....	1 626	.....	1 626	1 626
.....	41 415	39 789	22 673	1 626
.....	518	519.5	274	.....
.....	152	.....	16	152
.....	3	1.5	111 (*b*)	.....
.....	670	519.5	290	152
.....	8 325 (*e*)	3 214 (*f*)	1 231 (*g*)	.....
.....	2 225	.....	70	668
in (24-hr)	10 550	3 214	1 301	668
le, ft	179	149	163	.....
ozzle, ft ...	160	134	137	.....
.....	4.78	12.38	17.1	.....
.....	0.73	.....	23	2.43
r = 2.63...	.....	.....	.....	.....
rd	67.1	20.4	8.3	4.2



ig 611. Layout of Elevator Pit, Pioneer Mining Co

Nozzle 4⅜ in; throat, 10.5 in; upcast, 15 in; lift, 26.5 ft in in clear; sides, from 2 to 4 ft; length, 136 ft. Grade: 3 in, first or upper 12 ft; 4 in, second 12 ft; 5 in third 12 ft; 5 in for each 10 ft thereafter. Riffles, Oregon pine blocks. Piping giant: nozzle, 3 and 3 1/16 in (only 1 nozzle used at a time under various heads). Tailing giant: nozzle, 2 9/16 in. Pipe lines: Main, supplying all water, 22 in, 1 429 ft; branch for elevator, 15 in, 307.5 ft; branch, for piping giant No 1, 11 in, 284 ft; branch, for piping giant No 2, 11 in, 236 ft; branch, for piping giant No 3, 11 in, 139 ft; branch for tailing giant, 11 in, 889 ft. Bedrock sluice-boxes, 24 in wide, iron, without riffles.

ioneer Mining Co (*a*)

	Cost per day	Cost per cu yd
	$ 15.00	$0.010
	179.77	0.126
	18.18	0.013
	56.90	0.039
	41.73	0.029
	$311.58	$0.217
	$ 38.30	$0.026
	9.09	0.006
	$358.97	$0.249

supplies, teams, handling boulders, boarding house costs tration charges. (*b*) Freight, tools, carbide, quicksilver, (*c*) Exclusive of ditch. An arbitrary charge based on ,000.

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Little Creek, Nome district, affords an interesting comparison wi Gravel is frozen, but bedrock is softer and flatter. A little more than hal are required to operate pits similar to those on Anvil Cr, but the combine and elevator water is only 1.09 cu yd.

River Bend Mine, Siskiyou Co, Cal; data by J. H. Theller, covering 6 tion in 1912 and 1913 (416). An old channel is worked, about 100 ft deep. Gravel is heavy, well rounded and contains boulders to 1 000 ll larger than 10.5-in diam is bulldozed. Gravel is hydraulicked with 2 No 2 using 350 in of water under a head of 90 to 100 ft. A Campbell ele with 3.5-in nozzle, 10.5-in throat, and 14-in upraise pipe at an angle of 70°. uses 375 in of water under an effective head of 325 ft; lift, 46 ft vert sump is 6 ft square at the collar, tapering to 4 ft at bottom. An aver of washed per day; duty of giant water, 1.19 cu yd per in. Seepage wate Working costs per cu yd, excluding administrative charges, are: foreman 2 freight, supplies, etc, 0.2¢; boarding-house, 1.4¢; maintenance and de deprec of plant and ditches, 1.5¢; total, 8¢ per cu yd.

Logan Mine, Waldo, Ore. From 15 000 to 30 000 cu yd of gravel wash with a total water supply of 1 600 miner's in; 4 giants are used, 2 in th stacking tailing. A 20-in hydraulic elevator, with 2 lifts, raises the gra large boulders and gravel is easily washed. Operating costs said to be a 4¢ per cu yd; this is a low cost, obtainable only under very favorable co

Swaziland, South Africa; data from J. J. Garrard (397). Tin-bearing aging about 18 ft deep, are worked. Records for 12 months' operation that 4 elevators, with an aver lift of 23.3 ft, took an aver of 73% of the w leaving 27% for the giants. Best month's work showed 60% elevator water; the worst, 84% elevator to 16% giant water; effic of elevator, 1(water supply was ample, its inefficient use was allowable. Working c including administrative charges, were about 12.5¢ per cu yd, covering at the rate of 466 cu yd per day. 61 native laborers were employed per s and 1 elevator; labor cost, 8.9¢ per cu yd. Elevator nozzle, 3.5 in; head on elevator water, 127 ft; consumption of nozzle water, 330 cu ft p

Box elevator. Knox and Haley (385) describe a home-made eleva cessfully by the North Fork Salmon River Mining Co, Cal. It consist lined box about 18 in sq, sloping about 50°, with a 5-in jet at the bott steel striking-plate at the top, immediately over the sluice. Height of lif operating head, about 250 ft. The water supply was excellent; about were required for the elevator. Boulders to 11 or 12 in diam were handled throat, everything being lifted directly by the jet. Capacity, about 1 00

Ruble or grizzly elevator consists of an inclined chute at an angle and 100 ft in length, including a 10-ft apron to make connection with apron fits closely between the walls of the main elevator chute; the latter long. The chute and apron are lined with 0.25-in steel side and ⅜-in Chute is about 8 ft wide, its walls tapering from 12 ft in height at the bo 4 ft at the top. For the first 20 ft of the incline, the bottom is solid. 70 ft consists of transverse grizzly-bars, 2.5 in apart, made of 2 by 6-in ti with 0.5-in steel straps. Underneath the grizzly is a steel-lined false b from upper end of the elevator to a sluice box, set at right angles to th directly under the lower end of the grizzly. The sluice, about 60 ft long on light trestles, is paved with Hungarian and pole riffles, consisting of 2 b shod with steel. If desired, the sluice can be covered and locked up. supported on 3 heavy stringers resting on trestle bents, which are moun skids, blocked up level from bedrock. It is moved with a winch and cabl

Operation: The elevator giant has its axis in a vertical plane passing thro line of the elevator and about 80 ft away from it. Wings, about 10 ft high scrap timber, are built out on either side, 1 wing extending to the bank; they on portable frames. The cutting giant works behind the elevator giant fallen gravel along the bank to the elevator wing, where it is picked up b giant and washed in small quantities at a time up the solid portion of the is taken to "boil out" the fines over the lower portion of the grizzly; o would be washed over the elevator top. When the heavy stones are c

d of the machine. The fine tailing is piped from the end
nt about once an hour. An extra tailing giant is set up
ater is not being used by the other giants. When the
the grizzly, a platform is laid on it extending practically
can be piled much higher than the machine itself. These
lly by the tailing giant. If the water pressure is high,
be worked with 1 set-up of the elevator. The elevator
when the driving limit of the cutting giant is reached;
ve. This serious delay can be obviated only by using 2
e water from one to another. At one property, this
water under a 450-ft head. The gravel was heavy, with
ons; depth of bank, 20 to 25 ft. Capacity of the elevator,
100 000 cu yd of gravel were washed in 4 months; total
naintenance, was 6¢ per cu yd. First cost of elevator,
om a railway. For further details, see Bib (385, 452).
tor are: low first cost, the fact that it can be made on
handling heavy boulders. It is adapted to flat placers
50 ft deep, where water is available for hydraulicking.
uneven bedrock, owing to difficulty in moving.

ing tailing is possible where dumproom is ample
and where the water will run off by gravity (406).
ining Co, Atlin, B C, stacks coarser tailing to heights
rking under heads of 110 and 140 ft (411). Where
king giant can often be operated intermittently.
under Hydraulic elevators, and Bib (415).

LUICE used in connection with giants in California, Ore-
tailing at the end of a sluice. On Mastodon Cr, Circle,
hydraulicked with 2 No 1 giants into a sluice delivering
s the tailing was without difficulty driven up an inclined
o-2 giant under a head of 100 ft was used for stacking;
for hydraulicking and stacking was 2.5 cu yd per miner's
of bedrock were raised by the giant. It is stated that this
ld be required by an hydraulic elevator, and that it is
r small installations (413). INCLINES and giants have
EDING SLUICES which must be elevated a short distance
-lined back-stop is built directly behind the sluice, and
it forming an incline leading to bedrock. Material is
st the back-stop, from which it drops into the sluice.

used extensively in Swaziland, South Africa, for
rd gives the following data (397):

enewable shoes on the impellers and renewable linings,
tors, running at 485 r p m, and deliver water and gravel
ump is mounted on a pontoon resting on bedrock in the
nd washed to a sump, whence it is pumped to an elevated
p is sometimes 300 ft long; delivery pipe slopes 45°.
, 1915, 3 gravel pumps ran 70.9% of the time that power
cu yd of gravel at an aver rate of 24 cu yd per hr.
o 230 cu ft per min; aver height of lift, 38.6 ft. Aver
ps was 42.2¢, of which 23¢ was for power (cost 0.733¢ per
for repairs. This gives a cost of about 1.8¢ per cu yd.
he 3 pumps, 27.35%. Normal crew for each pit (1 giant
165 natives per day of 3 shifts. Total costs (including
cu yd. (These results are remarkable. The pumps
water power than hydraulic elevators working under
er lifts. In America centrifugal pumps for elevating sand
excessive expense for renewals and repairs. Author.)

ly consist of an endless chain-bucket excavator, sup-
to an elevated sluice. The buckets handle the gravel;

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a centrifugal pump is required for the water. These elevators have bee
mines to raise gravel mined by hydraulicking, but have failed because
and operating costs; (*b*) pump troubles; (*c*) complicated machinery, r
of a machine shop; (*d*) lack of mobility. For details, see Bib (385, 40

126. Gold Dredging

Introductory. A placer dredge is a scow carrying a mechanical
screening and washing plant, for mining subaqueous gravels. The
widely employed for INLAND DREDGING in California and elsewhere. T
on an artificial pond, usually supplied with water from an outside source;
supply is desirable, though pumping is often necessary. The dredge di
the pond and fills in the opposite side with washed tailing, thus carryir
it as it advances. Under suitable conditions the method is profitable fo
placers, otherwise unworkable. The modern efficient gold dredge has
since 1901; it may not yet have reached its limit of capacity and effect
The elevator bucket dredge is the only type described here, as it has p
seded other forms. Gold dredging is a subject of great detail; for a go
see (438, 437, 385); (409) contains a brief bibliography.

Requirements and limitations. Deposits suited to dredg
sive river bar and gravel-plain placers (Art 116), occurring chi
cally old districts. Requirements as to bedrock, character and
effect of boulders, cemented gravel and buried or standing tir
limits of dredging depth, water for dredge ponds, etc, are sum
117 and Sec 25, Art 7. See also under Cost and operating data,
considerations affecting choice of method and operation and de
are: CHARACTER OF GOLD. Very fine flaky gold, like that on th
Idaho, can not be profitably recovered by the present da
devices. Coarse gold sometimes modifies the form of the appa
GRADIENTS. Dredging is inherently applicable to large flat d
lack of grade prevents hydraulicking; small topographic irre
flat surface or bedrock gradients affect only the height of bank
water level. Some dredging has been done on grades as steep a
ing dams to maintain a pond (431). FROZEN GRAVEL must be
it can be dredged. Shallow gravels may be thawed by solar he
of removing muck or other covering; deep gravels require s
(Tables 121–123). FLOODS. Dredging in beds of torrential s
carious, due to danger of being wrecked. CLIMATE determines
working season and hence the annual output and return on inv
dredge of a given capacity. The minimum gold content of w
is higher where the season is short than where operation is cont
climate increases costs, because the dredge must be heated and
protected from accumulations of ice. See Bib (408, 432, 409)
of successful winter work. TRANSPORT FACILITIES are more in
for other forms of placer mining. The weight of the machinery
for small dredges; attempts to sectionalize a plant for mule
have not been successful. The high first cost of a dredge whe
are lacking militates against the construction of large boats in
(385, 433). LABOR. A small crew can handle a large yardage
except a few roustabouts are skilled men. Cheap POWER is esse
YARDAGE of gravel must be sufficient to amortize the initial in
yield the desired profit. A choice often exists between use of
several, of different sizes, with corresponding alternatives in firs
return, and life of property (Sec 25, Art 14). Closer estimates o
possible on well-explored dredging ground than on any other fo
property. An attempt is usually made to adjust the rate of

rty, to the LIFE OF THE DREDGE. This is practically machinery is repaired or renewed as necessary. The ıtled after working out an area, and installed else- a new hull. Steel hulls resist digging strains longer s not yet known. Wooden hulls last about 10 years.

Cal, states: The life of wooden hulls seems to be affected built, those erected in summer outliving others. Forced l is required to prevent dry rot.

on Life of Wooden Hulls, Natomas Co, Cal

work	Condition of hull, July, 1917
905	Sunk May, 1912. Rebuilt on steel hull
905	Dismantled Nov, 1914. Hull not bad, but dredge was run down
905	Sunk June, 1913. Raised and repaired in 6 mo. Dismantled Apr, 1917
908	Dismant, Feb, 1916; poor condition
908	In operation; fair condition
908	In operation; liable to go any time
908	In operation. Drydocked for repair May, 1913; should last 2 years
906	In operation; liable to go any time
908	In operation. May last 2 years

apacity is essential to low working costs; hence the con- redge must be planned to minimize delays.

veloped in New Zealand and Cal, are shown diagram-) SINGLE-LIFT SLUICE DREDGE. Buckets discharge ojecting astern. (*b*) DOUBLE-LIFT SLUICE DREDGE. grizzly; the oversize is undersize, with or with- raised by a centrifugal o a sluice on an auxil- -LIFT SLUICE DREDGE. nmel, the undersize of an auxiliary pontoon. ITH STACKER. Buckets oversize is piled behind (stacker); undersize is bles (not shown in Fig scharges astern through SLUICE DREDGE. Same coarse-mesh trommel at t stacker dumping the

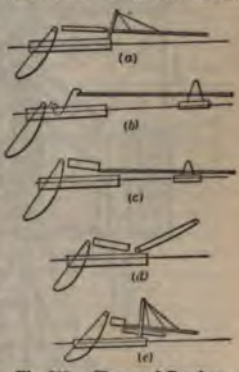

Fig 612. Types of Dredges

ically obsolete and are not otes deal chiefly with type n use, known as the CALI- his type, but smaller than the California and differing from , is the NEW ZEALAND DREDGE.

nly called "flume" or FLUME-TYPE. Type (*e*) has been n the Seward Peninsula, Alaska. Summary of conditions on) (434): Dredging is confined to narrow but relatively a; aver depth, about 8 ft; working season, only 4 months; e. In general, the gravel is loose and easily dug, and the th and large table area of the Cal dredge are not required. draught, and inexpensive, and has been successful.

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SIDE ELEV
Rear gantry
Main gantry
Main drive
Upper tumbler
Hull
Ladder hoist
Pilot house and winch room
4-cu. electric hoist
Front gantry
Bridge
Water line
Tail sheave
Digging depth 65' x 65' x 85'
170 to end of stacker
Stacker ladder, 175 ft. c-c tumblers
45°
to crown of deck 31'9"
to crown of deck
travelling crane
40'-10"
110'-0"
170'
60'
15'0" to side of hull

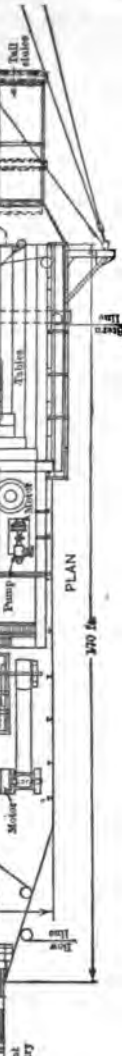

Fig 614. California Type of Gold Dredge, built by Bucyrus Co, Milwaukee, Wis (Plan)

Other forms sometimes used are the dipper, clam-shell bucket, and suction dredges; for data, see Bib (409, 435). In Australia, gravel pumps (Art 125), with or without a centrifugal pump for supplying giant water, are called pump-dredges. This work is not strictly dredging, though the pontoon carrying the pumps usually floats from place to place. For details, see Bib (436).

Construction and mechanical equipment. (For dimensions and weights, see Tables 111 *et seq.*) Fig 613, 614 show a recent California-type dredge.

Hull is of wood or steel, generally wood, the choice depending on size, difficulty of dredging and consequent strength required, relative first cost and life, and locality. Use of steel has been increasing, especially for large boats. R. G. Smith furnishes the following on Cal practice (Aug, 1917): Steel hulls have shown considerable pitting from action of the muddy water (439), and require repainting. Bitumastic enamel was used on the latest boats. A combination wood and steel hull is now being built for a new dredge; all loads are carried by steel, but the hull is of wood, reinforced on outside corners by steel plates; it is cheaper and apparently as good or better than the all-steel hull.

The bow end is divided, to form a well for the bucket chain and ladder. Depth of hull depends largely on the required displacement, the beam partly on the desired gold-saving area, the size of well on the dimensions of the digging ladder (which depend in turn on the depth of digging), and height of main gantry. Distance from well to stern is determined by length of screen and arrangement of machinery. Hull is designed to ride on an even keel when digging at aver depth, with a free board not less than 0.25 of its total depth (440).

Gantries (gauntries) (Fig 613). Front gantry carries the tackle for raising and lowering the dig-

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ging ladder. Main gantry supports the bucket drive and upper end of d
height depends on headroom required for the screen, gold-saving tables
Rear gantry carries the stacker and tackle for handling the spuds.

Bucket line is supported by the DIGGING LADDER, consisting of 2 hea
plate girders; lattice girders are used only for easy digging. In the
dredge, the bucket line is close-connected (Fig 613), a form able to excav
and dig a much greater yardage than the open-connected or bucket-a
tion, characteristic of the New Zealand dredge. The bucket line is c
on the ladder; for deep digging, an idler wheel reduces the sag of the line

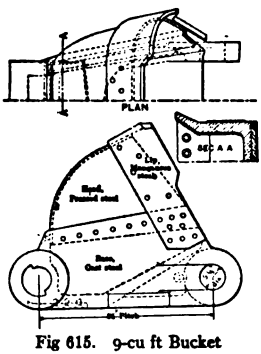

Fig 615. 9-cu ft Bucket (Cranston)

The UPPER TUMBLER, which drives t
hexagonal; LOWER TUMBLER, usually
times round. BUCKETS vary in capa
cu ft. Fig 615 shows a 3-piece bucke
towards 2-piece buckets, in which t
tegral part of the base. Some 1-piece
an attempt being made to proportion
the lip will last as long as the base a
of the bucket varies; some engineer
arched digging face because of its stre
that a flat face with sharply rounded
digger (440). To dump freely cla
require special shapes (441).

Main drive consists of a gear on e
tumbler shaft, connected with a gear
gantry. These gears may be belt-dri
motor, or from a countershaft on the
also drives the ladder hoist, or gear
motor. Each plan has its advocates;
(440, 442). In either plan, there is som
friction between the source of power and tumbler shaft. The DUMP HO
gravel from the buckets as they turn over upper tumbler and delivers it

Screens. The trommel (Sec 28, Art 16), used almost entirely instead o
screen of early Cal dredges, is especially advantageous in disintegrat
clayey gravel. It has powerful spray pipes. Ordinary mesh of screens,

Stacker is a belt conveyer on a lattice girder truss (Fig 613), belt-d
head pulley; gear drives are unsatisfactory (440). The stacker belt re
oversize and deposits it on the dump behind the dredge. Height of dum
swell of the gravel, height of bank carried above water level, relative digg
radii, relative amounts of sand and stones in the gravel, and method of oper
In the Cal dredge, fine tailing is deposited on bedrock and coarse tailing
thus greatly increasing the space occupied by tailing as compared with plac
angle of repose of coarse gravel tailing is close to 40°; of sand above wate
of sand below water level, 40° to 45°. Conveyer belts will not work abov
23°, partly because rounded stones begin to roll at about that slope wh
fully loaded. All of the above factors affect the length of stacker required
such a distance back of the boat that it will not interfere with operation. A
to swing laterally, as well as up and down, may facilitate tailing disposal.
or more stackers on a single dredge, see below under Resoiling. Stackers
dredges consist of pin-connected pan or tray elevators; they will work
than belt conveyers, but are more troublesome and require frequent rep

Anchorage, etc. Spuds (Fig 613) hold the dredge in position and act
For stepping the boat ahead to a new position, 2 spuds are required; b
steel construction, so that either may be used for digging if the othe
lines have some advantages (438) over spuds for very easy digging, bu
in the U S. When at work, the dredge is swung across the cut by wire
connected to the lower end of the digging ladder, and running thence
and back to a winch on the main deck. This winch also drives drums
stacker ladder and spuds, and for operating stern lines; the latter contr
of the boat when shifting it laterally with the spuds up. For details of wi
hoist, see makers' catalogs and Bib (440).

; steam is feasible but clumsy; internal-combustion successful for small dredges in Alaska. For fuel con- lectric power is not already available and 2 or more ocality, the feasibility of a central power station should le electric motors, see Bib (442).

ipes or jets in the screens, for gold-saving tables, for umps are needed; all are centrifugals, taking water ect-connected where electric power is used. Equip- orking head, 60 lb or more) for spray pipes, and a small shing down decks, cleaning-up, and priming larger ings without the use of a sand pump to elevate fines pump furnishes water for a small giant on the bow ank ahead of it (438).

y strains and wear on much of the machinery, high- ors are required. High carbon steel, manganese steel l alloys are all used. For further detail, see Bib (440,

ype. The former is more lightly constructed, has f bucket, consumes less power and costs much less. ricts. Knox and Haley state that the New Zealand of small acreage, or where the gravel is moderately or where capital is limited. Its chief application is are poor (385). It is generally built in Great Britain.

practically an undercurrent (Art 124), which h water of the trommel, with added water as

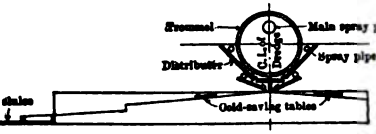

Fig 616. Diagrammatic Cross-sec thro' Trommel and Gold-saving Tables (*after Janin*)

are 2 to 3 ft wide, set on grades of 1¼ to 1½ in widely and in recent dredges has been greatly ge handled.

discharges astern through tail sluices. Table g several tail sluices, some of them projecting main tables, a small sluice or table catches hey descend into the well; considerable recov- alled the "save-all").

ss nearly standardized than other parts of the dredge. ithout mercury, have been tried; at present, small few dredges are equipped with Neill jigs, followed by a lgamator. The Richards pulsating riffle has also been) in Cal is indicated by the following, from R. G. Smith, Tables are 30 in wide, all having Hungarian riffles le with a 1.25-in space, of wood shod with iron straps. nk of tables. Transverse tables have a grade of 1.25 in per ft. Table area approximates 500 sq ft per cu m of saving fine gold has not yet been solved, under iling in many dredge ponds. For further detail and , 446, 409, 447, 390, 408); also tables below.

ne position of the spud, a dredge excavates a slice lly about 200 ft wide and extending from the sur- ess of the slice is from 6 ft up, depending on t[he]

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character of gravel and size of dredge. Digging begins at the s bucket ladder raised; the dredge is swung across the face, the ladd 6 ft or more, and the dredge swung back, these operations being bedrock is reached, After bedrock has been scraped the dre ahead, by means of the 2 spuds and the bow lines, and begins an

Narrow deposits are worked to their full width in one operation; wide several successive cuts. In the latter case, the method depends on the n and is often modified by local conditions. A single dredge excavates a wide, extending through the property, then turns around and comes Tailing should be so stacked that it will not pile up on the sides of the rehandling when the adjacent ground is worked; this influences width of of cuts taken simultaneously in one direction (390, 438).

Clean-ups are made as described in Art 124. Yardage dug is u monthly; for a method of measurement, see Bib (448).

Resoiling. The typical California dredge destroys the la tural purposes. Flume-type dredges also dissipate a loam cove posit coarse and fines together, leaving a fairly even surface at a nal level. The problem of re-soiling dredged areas cheaply has attention. R. G. Smith comments as follows on recent expe Folsom district, Cal; see also Bib (449):

First trial was made by equipping a dredge with 2 short stackers dump at points between the stern and ends of the tail sluices. The lower end of arranged to segregate large stones, and dump them through a chute ne second dredge was fitted up similarly, but with 2 additional stackers (4 in

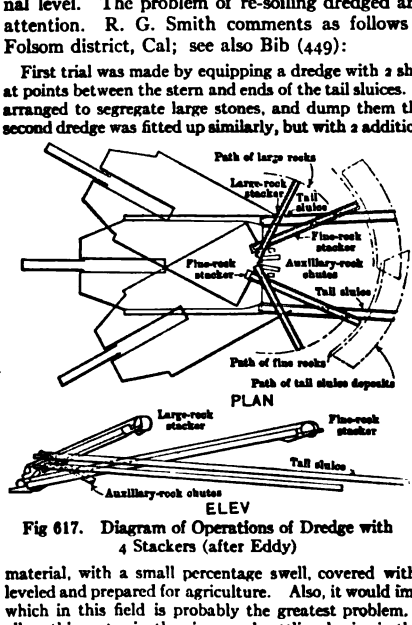

Fig 617. Diagram of Operations of Dredge with 4 Stackers (after Eddy)

large rocks (Fig procedure filled tailings, gave a f tribution, and di dumps. But, du action of the wate no fines were left the soil was all i dredge pond. T ing (Aug, 1917) a 60 ft deep, with device to dewate posit them with t eventually, it m conveyers to dep fines on top of th are also under wa dredge with a dump hopper, a carry the topsoil and deposit it on this is successfu fairly uniform dis material, with a small percentage swell, covered with the original soil leveled and prepared for agriculture. Also, it would improve the muddy which in this field is probably the greatest problem. The Debris Co allow this water in the river, and settling basins in the old tailings are The problems of disposing of muddy water and leaving present farming condition are the controlling factors in many available dredging fields.

Cost data. Operating costs vary widely with character and size of and size of dredges, and with locality; the latter determines the length o and cost of labor, supplies, power and transport. Cost of repairs an during the dredge's life, being small at first, then increasing to a fair and often decreasing toward the end of an operation when only essen made. Discrepancies are found in comparing operating costs of similar

ons, due to differences in accounting. Short time records are not reliable criteria; example, note the yearly variations in cost of the same dredge (Table 110). perating costs in the U S range from 3 to 10¢ per cu yd. Under very favorable con- ns, costs on individual dredges are slightly below 3¢; on many Cal dredges, between d 6¢. Bib (435) gives costs with 3 to 7.5-cu ft dredges in Cal, in 1910. A dredge in o, in 1914, handled 4 670 000 cu yd at a total cost of 2.77¢. At Breckenridge, Colo, dges of the Tonopah Placers Co handled 3 242 240 cu yd during 1915 at 6.1¢. mirs estimates that costs in Alaska usually vary from 20 to 30¢, although lower fig- are sometimes claimed. He reports other costs as follows (409). Pato dredge, mbia: 8-ft dredge for year ended Sept 30, 1915, handled 1 420 000 cu yd at 16¢, ding general expense which, in 1914, was 4.8¢. Gumaos dredge, Philippine Islands: dredge, from Oct 1, 1912, to Dec 21, 1914, handled 1 850 000 cu yd at about 12¢. iled costs of American dredging are given in tables below. A dredge crew consists winchman and an oiler per shift, with 1 or more roustabouts on day shift; 1 dredge er may look after several boats; blacksmiths, machinists, teamsters, and other labor, depend on locality and nearness to custom shops; 3 8-hr shifts are usual.

rthern California. Data from J. F. Newsom. Table 109 gives the operating d of a 5-cu ft dredge (wood hull), built in 1908 for $65 000. It worked out the prop- on which it was erected, and in 1915 was dismantled and re-erected on a new hull other place for $51 242. The data are fairly typical of a 1-dredge operation. The al property was 5 miles from a R R. Ground was 12 to 20 ft deep, rather hard contained much clay in places; bedrock, mostly sandstone, somewhat irregular and red considerable scraping to clean it up. Electric power cost 1¢ per kw-hr. Second erty is 15 miles from R R. Gravel is 7 to 15 ft deep, lies on a soft lava-ash bed- which is ideal for cleaning up, is very easily dug, free from clay and boulders, but is y in places. On account of lack of depth, considerable bedrock sometimes had to ug, and low water required damming and re-pumping to secure flotation. This is of the lower cost 1-dredge operations in Cal. Electric power costs 0.75¢ per kw-hr, $150 per month. Dredge data: 59 buckets, capacity, 5 cu ft; total weight of bucket, lb; weight of lip, 166 lb. Ladder, 60 ft long; dredge designed to dig 24 ft under er. Total hp, 290; digging motor, 75 hp. Table area of original dredge, 1 700 sq ft; ebuilt dredge, 1 270 sq ft. Wages, original dredge: winchman, $3.25, oilers, $2.75, ers, $2.50. Wages, rebuilt dredge: $3.50, $3, and $2.75 respectively.

Table 109. Cost of a 1-Dredge Operation, Northern Cal (J. F. Newsom)

	Original dredge			Reconstructed dredge		
Year	1911	1913	1914(*b*)	1915(*a*)	1916	1917(*c*)
Running time, % of total (*d*)	77.0	77.1	76.3	81.4	83.5	82.0
Cu yd per day	1 398	1 994	2 166	2 268	2 189	2 418
Cu yd per hr operated	75	105	127	115.2	113.1	121
Total gravel worked, cu yd	42 531	60 917	559 626	290 646	797 984	360 358
Acres worked	1.38	2.10	21.47	15.85	38.48	.....
Labor	2.38¢	1.79¢	1.53¢	1.48¢	1.50¢	1.20¢
Power	1.97	1.32	1.07	1.06	1.12	0.98
Repairs and supplies	1.90	1.10	1.22	0.24	1.06	0.61
Taxes and insurance	0.45	0.23	0.36	0.69	0.37	0.15
Miscellaneous	0.61	0.67	0.73	0.26	0.41	0.16
Express and treatment	0.06	0.04	0.04	0.04	0.04	0.05
Salaries	1.34	0.69	0.82	0.57	0.69	0.65
General expense	0.06	0.05	0.05	0.04	0.08	0.11
Total	8.77	5.89	5.82	4.38	5.27	3.91

(*a*) Began operating Aug 24, 1915. (*b*) Shut down from latter part of Dec, 1913, to Apr 1914, on account of high water. (*c*) Aver from Jan 1 to June 1, 1917. (*d*) On days ked. Does not include long shut downs.

Folsom District, Cal. Data from R. G. Smith. Tables 110, 111, 112, cover 10 ars, selected for purposes of comparison. Dredges *A*, *B*, and *C* are of similar size and k under similar conditions. Dredges *D* and *E* were built to dig deeper and more diffi- t ground; their operating costs were somewhat *increased by* sinking and reconstruct-

Table 110. Details of Operating Costs, Dredges A to I, Folsom District,* Cal

	Year	Days operated	Running time per day, hr, min	Material handled: Thousands of cu yd	Material handled: Running time, cu yd per hr	Aver depth, ft	Cost per cu yd, cents: Operating: Labor	Operating: Supplies	Operating: Power	Operating: Water	Repairs: Labor	Repairs: Supplies	General expense	Total
Dredge *A*, Folsom dist, 8.5-cu ft buckets; wood hull	1912	364	18 : 53	1 920	280	24	0.94	0.18	0.62	0.10	0.23	1.60	0.69	4.54
	1913	364	20 : 42	2 142	284	27	0.88	0.13	0.64	0.15	0.17	1.59	0.48	4.19
	1914	363	20 : 57	1 817	239	21	1.03	0.18	0.71	0.19	0.19	0.95	0.47	3.89
	1915	363	20 : 13	2 019	275	32	1.03	0.17	0.75	.....	0.25	1.81	0.55	4.56
	1916	364	21 : 38	2 407	306	35	0.74	0.14	0.76	.....	0.12	0.52	0.54	2.82
	Totals & aver	1 818	20 : 30	2 061	277	.....	0.92	0.16	0.70	0.08	0.19	1.27	0.54	3.95
	% of total....	.....	.....	.....	.....	.....	23.3	4.1	17.7	2.0	4.8	32.2	13.7	100.0
Dredge *B*, Folsom dist, 8.5-cu ft buckets; wood hull*	1912	365(a)	19 : 24	1 988	281	29	1.00	0.22	0.64	0.10	0.19	2.29	0.67	5.31
	1913	363	20 : 23	1 726	233	28	1.11	0.23	0.75	0.08	0.23	1.27	0.87	4.74
	1914	364	19 : 42	1 617	225	37	1.13	0.15	0.75	0.14	0.21	2.29	0.51	5.33
	1915	363	21 : 04	1 880	246	31	1.13	0.20	0.71	0.06	0.17	0.82	0.61	3.70
	1916	364	21 : 35	1 413(b)	180	26	1.20	0.26	1.01	.....	0.19	1.03	0.79	4.48
	Totals & aver	1 819	20 : 24	1 725	232	.....	1.11	0.21	0.76	0.08	0.20	1.56	0.69	4.72
	% of total....	.....	.....	.....	.....	.....	23.5	4.4	16.1	1.7	4.2	33.1	14.6	100.0
Dredge *C*, Oroville dist, 7.5-cu ft buckets; wood hull	1912	364	21 : 32	1 589	203	31	0.84	0.04	0.60	.....	0.05	0.95	0.60	3.39
	1913	363	20 : 22	1 483	200	33	0.83	0.05	0.60	.....	0.10	1.66	0.55	4.11
	1914	363	20 : 40	1 509	200	35	0.87	0.06	0.60	.....	0.13	0.64	0.45	3.01
	1915	363	20 : 41	1 363	181	31	1.09	0.04	0.66	.....	0.18	0.87	0.37	3.21
	1916	364	21 : 28	1 300	166	27	1.07	0.06	0.68	.....	0.14	0.53	0.34	2.82
	Totals & aver	1 817	20 : 55	1 449	190	.....	0.93	0.05	0.63	.....	0.12	0.94	0.47	3.32
	% of total....	.....	.....	.....	.....	.....	28.0	1.5	19.0	.....	3.6	28.3	14.2	100.0
Dredge *D*, Folsom dist, 9-cu ft buckets; wood hull†	1912	364	19 : 44	1 749	251	56	0.90	0.18	0.97	0.07	0.32	2.29	0.76	5.73
	1913	217	19 : 16	1 049	251	53	0.87	0.12	0.97	0.07	0.35	1.54	0.58	4.75(c)
	1914	363	19 : 45	1 803	251	54	0.77	0.08	0.84	0.07	0.33	1.56	0.46	4.27
	1915	363	19 : 58	1 516	209	45	1.00	0.20	0.90	0.00	0.27	2.01	0.85	6.41(d)

(a) 20 days' shut-down for repairs. (b) 128 000 cu yd bedrock dug for flotation. † Sunk June, 1915; raised this hull was in poor shape, expected to sink at any time. (c) $66 000 repairs subsequent to sinking not in mantled Apr, 1917, for rebuilding.

	1915	363	19 : 15	2 211	316	61	0.87	0.23	1.49	0.10	0.34	2.17	0.47	3.07
	1916	364	18 : 58	2 338	339	56	0.83	0.29	1.56	0.12	0.36	3.45 (g)	0.53	7.13
	Totals & aver	1 817	19 : 47	2 382	331	.....	0.81	0.23	1.38	0.12	0.27	2.42	0.56	5.86
	% of total	.....	.....	.....	.....	.....	13.8	3.9	23.6	2.0	4.6	41.3	9.6	100.0
Dredge *H*, Pol-	1913	363	18 : 55	2 470	360	45	0.88	0.18	1.07	0.08	0.24	2.03	0.58	5.15
	1914	363	17 : 59	2 270	347	37	0.88	0.17	1.09	0.12	0.30	1.96	0.58	5.26
	1915	363	20 : 11	2 608	356	43	0.91	0.16	1.44	0.10	0.20	3.24 (g)	0.64	6.69
	1916	364	20 : 28	2 851	383	54	0.86	0.15	1.15	0.09	0.13	1.15	0.53	4.00
all steel hull	Totals & aver	1 453	19 : 23	2 550	362	.....	0.83	0.16	1.13	0.10	0.21	1.95	0.56	5.00
	% of total	.....	.....	.....	.....	.....	16.6	3.2	23.6	2.0	4.2	39.0	11.2	100.0
Dredge *I*, Pol-	1912	364	19 : 20	3 089	431	23	0.65	0.21	0.60	0.13	0.14	1.56	0.44	3.87
	1913	363	18 : 50	2 575	376	22	0.73	0.11	0.74	0.07	0.25	1.46	0.60	4.09
	1914	363	19 : 34	2 884	406	24	0.59	0.06	0.70	0.08	0.19	1.75	0.44	3.93
	1915	363	19 : 31	3 304	466	22	0.55	0.10	0.68	0.03	0.17	0.85	0.32	2.70
wood hull **	Totals & aver	1 453	19 : 24	2 963	421	.....	0.63	0.12	0.68	0.08	0.19	1.39	0.44	3.61
	% of total	.....	.....	.....	.....	.....	17.5	3.3	18.8	2.2	5.3	38.5	12.2	100.0
Dredge *J*, Oro-	1912	288.5	20 : 30	2 400	406	27	0.75	0.08	0.94	.....	0.04	1.45	0.41	3.95
	1913	363	19 : 43	3 090	432	27	0.78	0.13	0.91	.....	0.09	1.87	0.41	4.46 (*h*)
	1914	363	20 : 33	3 336	447	41	0.67	0.13	0.80	.....	0.08	0.65	0.30	2.80
	1915	363	20 : 57	3 169	417	45	0.62	0.06	0.95	.....	0.09	1.00	0.16	2.88
	1916	364	20 : 43	3 252	431	53	0.58	0.07	0.94	.....	0.10	1.13	0.13	2.95
	Totals & aver	1 741.5	20 : 28	3 181	427	.....	0.67	0.09	0.90	.....	0.08	1.20	0.24	3.37
	% of total	.....	.....	.....	.....	.....	19.9	2.7	26.7	.....	2.4	35.6	7.1	100.0

Dredge *H*, Pol- / Dredge *I*, Pol- / Dredge *J*, Oro-: cu ft buckets; som dist; 15-cu ft buckets; all steel hull / cu ft buckets; 13-som dist; wood hull / ville dist; 15-cu ft buckets; wood hull

of dry docking and extraordinary repairs in May, 1913, which
dge. (*f*) Incl cost of new bucket line and a 15-day repair job.
smantled for rebuilding, Feb, 1916. (*h*) Incl new bucket line.

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The large dredges *F*, *G*, and *H* are designed for the same conditions as *D* and *E* *I* has large capacity and is designed for shallow, easy digging; *J* is similar worked in easier but deeper ground than *I*.

To June, 1914, a suspense account was carried for large items, as bucket l blers, etc, these costs being distributed pro rata over the life of the parts in Thereafter, when such parts were put into use, their full cost was charged to Wages on all dredges: winchmen, 45¢ per hr; oilers, 35¢ per hr; laborers, $2.5 Operating costs do not include taxes and insurance, amounting to a yearly dredge of $2 740 to $4 970, aver $3 600 for all dredges but *I*. For the latt taxes, etc, are $5 500 to $6 760.

Table 111. Summary, 10 Dredges in Folsom Dist, Cal

Dredge	Date started *	Bucket, cu ft	Deposit: Character	Deposit: Aver depth, ft	Digging depth below water, ft	Hull	First cost †	Power, per kw-hr	Aver operating cost, cts per cu yd
A	Aug, 1908	8.5	(*a*)	28	35	*W*	$135 000	0.70	3.95(*k*)
B	July, 1908	8.5	(*b*)	29	40	*W*	135 000	0.70	4.72(*k*)
C	Mar, 1908	7.5	(*c*)	32	34	*W*	130 000	0.75	3.32(*k*)
D	Dec, 1905	9	(*d*)	51	50	*W*	160 000	0.70	5.10(*k*)
E	Mar, 1908	9	(*e*)	48	50	*W*	185 000	0.70	6.28(*k*)
F	Jan, 1913	15	(*f*)	63	60	*S*	345 000	0.70	5.65(*l*)
G	Aug, 1911	15	(*g*)	59	60	*W*	330 000	0.70	5.86(*k*)
H	Oct, 1912	15	(*h*)	45	60	*S*	365 000	0.70	5.00(*l*)
I	May, 1908	13.5	(*i*)	23	32	*W*	200 000	0.70	3.61(*l*)
J	Sept, 1911	15	(*j*)	39	50	*W*	320 000	0.75	3.37(*k*)

* For period tion, see Table 1 prox. *W* = We All steel. (*a*) I and gravel on lav very easy dig Loose sand and g considerable tops bedrock. Botto is fair size and not easy digging erable drift mi in former years. sand and gravel considerable sa very easy di Tight clay and g 5 to 6 ft of tops ft of hardpan hardness and bedrock; similar

(*g*). (*e*) Tight clay and gravel; some difficult hardpan digging; lava bedrock. (*d*), (*f*), (*g*). (*f*) Tight clay and gravel, with 8–10 ft topsoil; like (*d*) and (*g*). and (*f*). Prospect shafts stand for years with little caving. (*h*) Worked on 3 benches; much manœuvering; tight clay and gravel, some hard digging in hardpan. Both soft lava and hard bedrock. (*i*) Loose sand and gravel, with 4 to 8 ft topsoil; soft lava bedrock. Very little clay, and bank not subject to excessive caving. (*j*) Very loose sand and gravel on soft lava bedrock; no clay; considerable topsoil. (*k*) Aver of 5 years' operation. (*l*) Aver of 4 years' operation.

Table 112. Details of Dredges A to J, Folsom

Dredge	Buckets: Size, cu ft	Buckets: Number	Buckets: Weight, lb: Lips	Buckets: Weight, lb: Complete	Digging depth, ft	Horse-power: Digging motor	Horse-power: Total	Ladder length,
A	8.5	68	336	2 390	35	200	463	
B	8.5	72	336	2 390	40	200	438	
C	7.5	68	....	.....	34	150	301	
D	9	82	356	2 850*	50	200	528	
E	9	86	468	2 830†	50	200	505	1
F	15	83	558	4 430*	60	400	1 120	1
G	15	83	558	4 430*	60	400	1 136	1
H	15	83	558	4 430*	60	400	1 165	1
I	13.5	60	466	3 850†	32	300	712	
J	15	81	558	4 430*	50	300	757	1

* 2-piece bucket. † 3-piece bucket. Life of parts: Manganese steel bucket 1 yr; 2-piece buckets, aver about 3 yr; stacker belts, aver about 1 yr; screen 1 set per yr. Bow lines of dredges *F*, *G*, *H* aver 11 lines, or total of 8 800 f dredges *D*, *E*, 6 lines, or 4 400 ft per yr. All other lines: dredges *F*, *G*, *H*, 3 6 *D* and *E*, 3 150 ft per yr.

Conrey Placer Mining Co, Mont; data by H. Jennings (409). Table 113 g sions of 4 dredges at Alder Gulch and Ruby. Gravel is 25 to 55 ft deep, not compact as at Oroville and Natoma, Cal, but is more difficult than in the Yuba R not so deep. Large boulders are found, embedded in clay; bedrock is favor

inter temp of 25 to 30° F below zero is a great handicap, me. Wages per 8-hr shift are higher than in Cal; elec- Tables 114–117 give operating data.

ls of Conrey Co's Dredges, Mont (409)

	No 1	No 2	No 3	No 4
.....	1908	1908	1906	1911
.....	96 270	62 709	82 415	300 000
.....	7.5	7.5	(*g*)	16–17
.....	60	80	(*g*)	80
.....	2 594	2 594	2 900	(*h*)
.....	32.75	32.75	32.75	40
.....	16	15	15	18–22
.....	4.625	5	5	7–8
.....	168	184	184	(*i*)
.....	5	5.5	6	9
.....	96	102	130	150
.....	44	44	48	58
.....	9	9	7.92	13
.....	90	85	.....	130
.....	34	34	.....	48
.....	48	54	.....	80
.....	26 500	29 000	.....	88 000
.....	100	100	150	550
.....	30	30	10	100
.....	160	160	175	285
.....	20	20	115	250
.....	70	70	10	50
.....	380	380	460	1 235
.....	69	80	100.5 (*b*)	116
.....	30	35	45	55
deck, ft.	25.7	29.3	31	35.5
.....	8 970	8 970	8 970	(*d*)
.....	16	16	16	25
.....	5 800	5 800	5 800	(*e*)
.....	13.25	13.25	13.25	15.375
.....	169 000	225 000	297 000	(*f*)
.....	20 ft 1 in	20 ft 1 in	11 ft 10 in	29 ft 10 in
.....	4 ft 11 in	4 ft 11 in	8 ft 6 in	7 ft 5 in
.....	16	16	30	24
.....	8	8	2	8
.....	35	35	18	48.5
.....	(*j*)	(*j*)	61	98
.....	0.625	0.625	0.625	1.0
.....	0.5–0.75	0.5–0.75	4.25 by 6	0.5–0.75
.....	12.5	12.5	12	12.5
.....	20	20	10	24
.....	30	30	30	30
.....	18	18	12	(*k*)
.....	4	4	5	3
.....	3, 8 & 12″	3, 8 & 12″	3, 12 & 14″ two 10″	4, 14 & 16″
.....	6″	6″	.....	.....
.....	6 700	6 700	12 500	12 000
.....	12.5	12.5	(*m*)	12.5
.....	2	2	.....	12
.....	40	40	.....	17–44
.....	48	48	.....	48
.....	8	8	8	8
.....	7.5	7.5	25.9	18.5
.....	18	18	18	18
.....	1 231	1 231	1 264	3 000

ec of steel spuds on No 1 and 2, 3 by 2 ft; on No 4, 4.5 1 wood and 1 steel spud; No 4, 2 steel spuds. (*b*) No 4 is plate girder type; on others, latticed girder type. which dredge was designed. (*d*) 16 000 to 25 300 lb. (*e*) 18 000 to 19 800 lb. All lower tumblers, round. n drive on No 3 and 4 is gear-driven from motor; No

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1 and 2, belt-driven. (*g*) 13-cu ft bucket and link at first; changed to 9.5 connected. 43 buckets open-connected, 80 close-connected. (*h*) 4 468 t Recent buckets on all dredges are 1-piece manganese steel. (*i*) 520 to Stepped cylinder, 51 to 72-in diam. (*k*) 11 to 29.75 ft. Tables on all dredges verse riffles of 1.25 by 1.25-in angles. (*l*) Riffles same as on tables. (*m*) S 3 is 5.5 ft wide and 163.9 ft long, of which 135 ft is carried on pontoon. Gra 6%; paved with transverse riffles of 2 by 2.5-in angles.

Dredges 1, 2, and 4 are California-type, built by Marion Steam Shovel C 1 has a short hull and digging ladder, and a long stacker to allow it to wo Gulch in narrow and difficult places, and to carry and hydraulick a high ba it; No 4 is designed for cheap work in deep, low-grade gravel. No 3 is

Table 114. Normal Crews of all Dredges of Conrey Co; Cost of E

	Men	Cost		M
Dredge No 1	8	$108 000	Transformer station & pole line	
" No 2	8	128 000	Machine shop	
" No 3	10	198 000	Clean-up crew	
" No 4	10	296 000	Labor	2
Office and dwellings	4	15 000	Miscellaneous	1
			Total	8

Table 115. Operating Report, Conrey Co, Dredges No 1, 2, 3, and starting to July 31, 1915

Item			No 1	No 2	No 3	No 4
Running hr, %			71	67	65	65
Lost hr, %			29	33	35	35
Cu yd dredged, new ground			7 817 520	5 515 756	8 166 281	9 880 333
Cu yd dredged, tailings (*a*)			72 678	752 240	479 872	145 192
Cu yd dredged, gross			7 890 198	6 267 996	8 646 153	10 025 525
Aver cu yd per running hr			174	132	161	420
Aver cu yd per bucket			0.18	0.18	0.19	0.38
Buckets per min			15.75	12.98	14.15	18.62
Aver kw-hr per yd			1.55	1.89	1.73	1.36
Acres dredged			223.16	143.62	144.43	134.92
Aver depth to bedrock, ft			22.1	28.1	40.8	45.3
Av working costs, cts	Labor	Per cu yd	1.50	1.68	1.75	0.79
	Fuel		0.11	0.13	0.13	0.07
	Power		1.27	1.55	1.48	1.09
	Supplies		0.10	0.11	0.09	0.09
	Upkeep of chain		1.03	1.49	1.51	1.08
Repair cost, cts per cu yd	Digging appliances, labor		0.19	0.32	0.28	0.23
	Digging appliances, material		0.32	0.46	0.62	0.61
	Pumping appliances, labor		0.04	0.02	0.03	0.01
	Pumping appliances, material		0.07	0.03	0.04	0.01
	Gold sav'g and wash'g app, labor		0.28	0.28	0.22	0.17
	Gold sav'g and wash'g app mat'l		0.60	0.66	0.36	0.57
	Electrical appliances, labor		0.04	0.05	0.05	0.05
	Electrical appliances, material		0.03	0.04	0.06	0.04
	Heating appliances, labor		0.01	.....	0.01	.....
	Heating appliances, material		0.02	0.02	0.02	.....
	Hull and housing, labor		0.01	0.05	0.04	0.01
	Hull and housing, material		0.01	0.02	0.01	0.01
	Miscellaneous		0.03	0.03	0.05	0.02
Total operating cost			5.66	6.94	6.75	4.85
Indirect expense, exclud'g int(*b*)			....	.....	.....	.....

(*a*) The tailings dredged are not used in calculating costs. (*b*) See note (*a*)

the gold-saving tables (Table 113) are supported on
rated plates, ⅝-in holes, placed in bottom of sluice,
stead of spuds. The company owns a large, well-
its own repairs.

and Expenses for Dredges No 1, 2, 3, and 4,
Sept 1, 1906, to July 31, 1915

Operating cost per cu yd, cents						
Power	Supplies	Upkeep of chain	Repairs	Total	Indirect expense (a)	Total
1.59	0.20	1.40	1.07	5.99	3.80	9.79
1.34	0.04	0.87	1.12	4.77	2.63	7.40
1.69	0.09	1.50	1.45	6.26	1.12	7.38
1.30	0.09	1.69	1.89	7.61	1.22	8.83
1.29	0.10	1.53	1.97	6.88	1.11	7.99
1.22	0.09	1.31	1.80	6.11	1.04	7.15
1.34	0.13	1.37	2.34	6.82	1.05	7.87
1.36	0.09	1.09	1.93	5.96	0.92	6.88
1.15	0.09	0.94	1.25	4.63	0.73	5.36
1.15	0.08	0.98	1.65	5.05	0.94	5.99
1.32	0.10	1.00	1.76	5.91	1.05	6.96

d Boston, ins, taxes, maintenance of pole line and sub-
nployes, legal expenses, accidents, all office expenses,
dness excluded. (*b*) To July 31.

d Delays, Conrey Co, Dredges 1, 2, 3, and 4,
n 1 to July 31, 1915

Time, %			
No 1	No 2	No 3	No 4
1.56	1.36	0.87	1.60
3.89	6.33	2.75	3.80
3.42	3.01	2.25	2.95
0.40	0.41	0.61	0.13
.....	.....	0.48	.....
0.02	0.14	0.23	1.07
1.02	1.58	0.70	5.91
0.16	1.24	.....	0.87
3.99	3.25	3.03	3.41
0.62	3.05	2.52	2.27
0.67	2.05	0.38	0.49
1.36	3.13	1.13	1.09
.....	.....	.....	0.03
3.66	2.68	0.36	0.11
0.68	0.66	0.53	0.57
0.85	1.48	.....	0.41
0.01	0.39	0.07	0.97
0.02	0.08	0.18	0.85
.....	0.06	.....	0.02
0.09	0.15	0.02	.....
0.71	0.78	0.78	0.71
0.07	1.00	.....	0.01
0.03	0.38	.....	.....
.....	.....	42.46	.....
0.02	0.01	0.02	.....
23.25	33.22	59.37	27.27
76.75	66.78	40.63	72.73
100.00	100.00	100.00	100.00

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Table 117a. Running Time and Delays on Dredge with 6-cu ft Buckets, Derry Ranch Gold Dredging Co, Leadville, Col
(N Y Eng Co)

Cause of delay	% of total time	Cause of delay	% of total time
Digging ladder	2.29	Oiling	1.15
Bucket line	2.01	Power off	0.41
Lower tumbler	0.49	Clean-up	0.39
Upper "	None	Belts	0.28
Hopper and gearing	0.77	Boulders and ice	2.84
Winches	0.25	Spuds	0.60
Motors, wiring	0.42	Miscellaneous	0.97
Screen	0.92	Total delays	15.24
Stacker	0.56	" running time	84.76
Pumps	0.73		
Lines and sheaves	0.96	" time	100.00

Yukon Gold Co, Dawson, Y T; data from O. B. Perry and Bib (408, 4
of 8 California-type dredges on Bonanza, Eldorado and Hunker Creeks
equipped with 7.5-cu ft buckets, 3 with 5-cu ft buckets. Bedrock is frac

Table 118. Dredging Costs, Yukon Gold Co, Dawson, Y

Year	No of dredges	Working season, days	Operating time, % of total	Total output, cu yd	Yield per cu yd, ¢
1910	7	147	84.7	3 249 788	66.18
1911	9	172	81.5	4 151 249	64.35
1912	8	172	86.1	5 157 280	64.88
1913	8	184	79.6	5 133 575	65.13
1914	8	148	82.8	4 800 781	54.21

Table 119. Costs per Cu Yd, Yukon Gold Co, for years 191

	Year 1913						
	Dredge number	1	2	3	4	5	6	8
Direct cost	Fixed salaries, dollar	.0009	.0009	.0010	.0007	.0007	.0008	.0008
	Labor	.0315	.0333	.0234	.0201	.0221	.0202	.0201
	Fuel	.0008	.0017	.0007	.0001	.0008	.0005	.0013
	Shop expense (repairs)	.0015	.0023	.0011	.0037	.0017	.0027	.0021
	Material, supplies	.0169	.0211	.0200	.0150	.0151	.0241	.0147
	Power	.0313	.0318	.0266	.0222	.0213	.0216	.0222
	Total	.0829	.0911	.0728	.0618	.0617	.0699	.0612
Indirect cost	Preliminary	.0422	.0302	.0188	.0168	.0206	.0234	.0201
	Taxes	.0006	.0006	.0006	.0005	.0005	.0005	.0005
	Bullion charges	.0170	.0259	.0131	.0121	.0209	.0261	.0247
	General charges	.0201	.0200	.0209	.0205	.0169	.0215	.0175
	Depreciation	.0224	.0238	.0187	.0206	.0154	.0181	.0151
	Insurance	.0012	.0013	.0010	.0017	.0011	.0012	.0010
	Assay office	.0011	.0018	.0009	.0008	.0014	.0017	.0016
	Stables	.0009	.0026	.0021	.0013	.0027	.0021	.0017
	Main ditch	.....	.....	.....	.....	.0003	.....	.....
	Telephone lines	.0003	.0003	.0003	.0002	.0002	.0003	.0002
	Transportation	.0001	.0001	.0001	.....	.....	.....	.0001
	Miscellaneous	.0050	.0050	.0050	.0053	.0050	.0050	.0050
	Total	.1109	.1116	.0815	.0798	.0850	.0999	.0883
	Thawing	.1255	.1228	.1836	.1795	.1234	.1638	.1279
	Total operating costs, dol	.3193	.3255	.3379	.3211	.2701	.3336	.2792

* Total aver costs per cu yd for 8 dredges.

Muck overburden is 2 to 20 ft deep, except where streams of deposits, 20 to 55 ft. Much of the ground is frozen; if not by vertical steam points (Art 127); for descriptions, see Bib (451, 454). Working season, May–Oct. dredge, exclusive of shops and stables: dredge master, oilers, $402.99; laborers, $1 072.85; clean-up men and ; repairs aboard dredges, $176.72; total, $2 740.99.

, Lost Time, Yukon Gold Co

1 dredge		8 dredges, 1912	1 dredge
0.2%	Motors	2.10%	1.2%
5.3	Power	5.50	7.8
11.3	Stepping ahead	23.20	23.6
17.8	Moving over	6.92	5.2
0.2	Clean-ups	8.70	12.4
4.0	Other causes	14.30	5.9
0.7		100.00%	100.0%
0.0	Running time	86.15%	91.1%
1.2	Lost time	13.85%	8.9%
3.2			

ing Costs, Yukon Gold Co (O. B. Perry)

d, sq yd		% naturally thawed	Thawing cost	
lly gravel	Total		Total, $	Per sq yd, $
8	298 061	28.6	275 112	1.557
4	396 837	35.5	500 689	1.773
0	555 393	22.1	696 034	1.617
8	692 522	26.4	804 854	1.455
1	651 505	31.6	685 570	1.546
7	586 737	31.4	602 174	1.428

Thawing Costs, Yukon Gold Co, 1913 (O. B. Perry)

er	Per cent of total	Item	Cents per sq yd	Per cent of total
	0.23	Stables	1.5	0.96
	42.18	Transportation	0.1	0.03
	43.17	Miscellaneous	0.1	0.06
	0.25	Depreciation	6.9	4.48
	5.25	Power	0.0	0.01
	3.38	Total	154.6	100.00

3 004 steam points driven; cost per point, $10.88.

awing Costs, Yukon Gold Co, 1913 (O. B. Perry)

st, cts per cu yd		Dredge No	Location	Naturally thawed ground, %	Cost, cts per cu yd	
al	Thawing				Total	Thawing
.93	12.55	5	*B*	44.8	27.01	12.34
.55	12.28	6	*B*	18.6	33.36	16.38
.79	18.36	8	*BU*	33.7	27.92	12.97
.11	17.95	9	*E*	58.6	21.16	7.49

Hunker Cr. *BU* = Upper Bonanza. *E* = Eldorado.

g tables give dimensions, etc, of several dredges.

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Table 124. 9-cu-ft Dredge (built 1911 by Marysville Dredging Co, C

Wood hull: total length, 115 ft; width, 50 ft; depth, 8½ ft; 200 000 bd ft.
Digging ladder: length, 115 ft; max depth, 8 ft; width, 4 ft 7½ in; wt c
94 400 lb; built up of 5 in × ¾ in L's and ½-in plates.
Stacker ladder; length, 140 ft; depth, 7 ft; width, 3 ft 8 in; wt complet
lb; built up of 4 in × 4 in × 7/16 in L's.
Bucket line: 84 9-cu ft buckets; wt of each complete with pin and b
2 200 lb; pitch, 3 ft; speed, 50 ft per min.
Revolving screen: length, 37 ft; diam, 6 ft; wt complete, 37 360 lb; bu
6 sections, 24 ½-in grizzly plates, ⅜-in holes.
Tables, sluices and save-all: total area, 2 430 sq ft. [L's and 7/16 i
Spud: length, 60 ft 10 in; wt complete, 36 000 lb; built up of 4 in × 4 in
Ladder winch: total wt, 39 000 lb; r p m of drum shaft, 7½; outside dia
Main winch: wt complete, 23 275 lb.
Pumps: high-press, 2 800 gal per min, 50 ft head; 10 in disch; 12 in suctio
low-press, 4 000 gal per min; 25 ft head; 12 in disch; 12 in suction
primer pump, 2-stage, 2 in disch; 2½ in suction.
Motors: digging motor, 200 h p; screen motor, 50 h p.
main winch motor, 25 h p; stacker motor, 35 h p.
high-press motor, 100 h p; low-press motor, 35 h p.
primer pump motor, 10 h p.
Lines: ladder line 1⅛ in-6/19-plow steel hoisting rope, 1 035 ft long.
spud line, ¾ in-6/19-C G S hoisting rope, 553 ft long.
stacker line ¾ in-6/19-C G S hoisting rope, 500 ft long.
swinging line 1 in-6/19-C G S plow-steel hoisting rope, 600 ft long.

Table 125. Principal Dimensions of All-steel California Dred

	Yuba #14	Yuba #15	Marigold #5	Pacific G
Date built	Dec, 1913	July, 1916	May, 1916	Oct–N
Bucket, cu ft	16	17	16	
Digging depth, ft (*a*)	65	82	68	
Total weight, tons	2 090	2 900	.....	1
Hull: length, ft	155.5	165	155.5	
Width, ft	58	68	58	
Depth, ft	11.5	11.5	11.5	
Weight, tons	743	850	.....	.....
Spuds, length, ft	62	70	62	
Weight, tons	40	45	41	.....
Ladder: Length, ft	134	154	132	.....
Weight, tons	125	190	125	.....
Upper tumbler: Weight, tons	.....	20.5	.....	.....
Lower tumbler: Weight, tons	10	11	10	.....
Buckets: Number	87	100	86	.....
Weight, lb	4 250	4 440	4 200	3
Pitch, in	41	.....	42	
Pins: Weight, lb	425(*c*)	430	425	.....
Stacker: Length, ft	136	175	145	
Belt width, in	42	42	42	
Trommel: Length, ft	50.5	50.5(*d*)	50	.....
Diam, ft	9	9	9	.....
Motors: Main drive, hp	400	500	400	
Winch, hp	35	35	50	.....
Trommel, hp	75	75	75	.....
Stacker, hp	60	75	60	.....
Total hp	.....	1 007	.....	

Note. Yuba #14, built in 4 months 4 days; Yuba #15, in about 6 weeks; bot
Construction Co (463). Yuba #16, launched late in 1916, has 18-cu ft buckets,
tons, and is designed to dig at depth of 84 ft; other dimensions resemble #15.
(464), built by Marysville Dredging Co and Union Iron Works. (*a*) Depth below
for which dredge was designed to dig. (*b*) 2 dredges, built by Marion Steam Shov
1 to dig 60, other 70 ft below water level; otherwise same. (*c*) Diam, 7⅜ in. (*d*) W

Gold Dredges, Built by N Y Engineering Co•

Gumaos Placer Co (*a*)	Mambulao Placer Co (*b*)	Chiksan Mining Co (*c*)
Steam	Steam-electric	Electric
1912	1914	1916
6	8	11
83	96	63
1 400	2 000	2 800
30	33	36
Carbon steel	Mang steel	Mang steel
Mang steel	Mang steel	Mang steel
Forged carb steel	Mang steel	Mang steel
4.5	5.25	6
120	175	225
Half rd, ¾"	Half rd, ⅞"	Half rd, 1"
120	135.5	110
40	47	50
8.5	10	10
Wood	Wood	Steel
6 ft 3 in	7 ft	7 ft
Lattice girder	Plate girder	Plate girder
87	114	78
38	55	28
28	32	29
6	6	6
Carb s & mang pl	Carb s & mang pl	Mang s & tip pl
15	18	21
11 000	18 000	24 000
6 to round	Round	Round
Carb s & mang s	Mang s	Mang s
10	12	14
7 500	11 000	14 000
60	78	84
32	42	46
Friction	Friction	Friction
28 000	45 000	56 000
Double bank	Semi-d'ble bank	Double bank
3 200	3 600	4 800
29	29	29
1¼" to 12"	1⅛" to 12"	1⅛" to 12"
8-in, 2 stage	12	12
50	60	60
2 000	4 200	4 400
12	14	14
25	25	25
4 000	4 800	5 200
22.5	22.5	20
6	6	7
18	18	20
9	9	7
24 " × 36-in " × 55 ft	24 " × 36-in " × 70 ft	28 × 42-in × 48 ft
24 000	38 000	Both steel
1¼	1¼	36 000
36	38	1⅜
85	110	42
	150	100
150	40	200
	40	60
110	220	50
20	20	250
10	15	20
290	485	15
90 000	140 000	595
		200 000

Gumaos, Ambos Camarines, Luzon, Philippine Isl. (*b*) Luzon. (*c*) Chiksan, Korea. (*d*) All close-connected, ngarian riffles.

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Table 127. Data on Gold Dredges, Built by Bucyrus Co, Milwaukee, Wis

	1	2	3	4	5	6	7*	8
Date built	Bldg 1917	1912	1911	1909	1911	1916	Bldg 1917	1909
Capac of bucket, cu ft	17	16	15	13.5	9	7.5	5.5	3.5
Digging depth below water line, ft	80	65	55	55	55	25	38	25
" " above " " "	10	10	8	8	8	8	8	8
Nature of digging	Hard	Hard	Hard	Hard	Hard	Medium	Loose	Medium
Hull	Steel	Steel	Steel	Wood	Wood	Steel	Steel	Wood
Depth, with machinery, ft	9	9.5	9.5	9.5	8.16	6.5	5.5	3.66
Length, ft	170	155.5	150	150	135	103.5	110	80
Width of deck, ft	60	58	58	58.5	46.5	45	60	34.5
Depth of hull at center, ft	14	13	12.5	12.5	11.5	10	9	6.66
Length of well	100	85	80	80	75	42.5	59	39.5
Freeboard, aver, ft	5	3.6	3	3	3.3	3.5	3.5	3
Bow gantry, height above deck, ft	59	55	50	50	50	30	36	22.5
Main " " " " "	35	33.3	33.75	33.75	29	23.75	24	21
Stern " " " " "	92.5	80	75	75	75	68	58	40
Total wt, lb, *or* l'mber, M, in hull, gantries, hous'g, etc	1 900 000	1 460 000	1 350 000	650 M	410 M	750 000	850 000	100 M
DIGGING LADDER	Pl girder	Pl girder	Pl girder	Pl girder	Pl girder	Pl girder	Pl girder	Tr I-beams
Length between centers of tumblers, ft	161	134	124	124	123.5	69	86	63.5
Extension beyond bow, ft	56	45	40	40	45	24	26	20
Weight, lb	320 000	234 000	213 000	215 000	150 000	58 000	66 000	14 500
STACKER (*a*). Type of conveyer	Belt	Belt	Belt	Belt	Belt	Belt	Belt	Belt
Length, centers of end pulleys, ft	175	136	136	136	150.5	135	90	75
Width, " " girders, in	58	54	54	54	50	48	44	38
Belt conveyer, width, in	44	42	42	42	38	34	30	28
Speed of conveyer, ft per min	350	325	325	325	350	340	340	300
Max height of discharge, ft	50	41	41	41	44	40	27	22
Motor hp	100	60	60	60	50	35	25	20
Ladder frame, wt, lb	85 000	40 000	42 000	40 300	45 500	35 000	25 000	10 000
BUCKET LINE (*b*). Number of buckets	101	88	85	85	96	66	82	72
Max number buckets per min	18	18	18	18	20.5	23	21.4	25
Bucket pins, pitch, in	42.5	41	39.5	39.5	34.5	29	28.25	24
Two or 3-piece buckets	2	2	3	3	3	2	2	3
Material of bucket (*c*)	Mang st	Ni-chr st	Ni-chr st	Ni-chr st	Ni-chr st	High C st	Mang st	High C st
Weight 1 bucket complete, lb	5 300	4 600	4 150	4 100	3 100	1 970	1 350	525
LADDER. Roller, diam, in	16	16	16	16	14	14	12	9
Weight 1 roller, shaft and bearings, lb	2 500	2 500	2 500	2 500	1 600	1 200	1 000	390
UPPER TUMBLER, diameter shaft, in	25	25	25	25	17.25	16	14	8.5
Span, centers of bearings, ft								

Weight of 1 spud, lb	135 000	81 000	89 100	89 000	61 000	28 000		8 500
Screen, diam, ft	10	9	9	9	7	6	6	5
Length over all, ft	50.5	50.5	50.5	50.5	37	35	30	25.5
Diam of holes, in	3/16, 3/8, 7/16	3/8	3/8	3/8	3/8, 1/2	1/4, 3/4, 7/16	1/4, 3/16, 3/8	3/4, 1/2
Slope per ft of length, in	1 3/4	1 3/8	1 3/8	1 3/4	1 3/4	1 3/4	1 3/4	1 3/4
R p m of screen	6	5.75	5.75	5.75	7.7	9.15	8.55	10
Screen motor, hp	100	75	75	75	50	50	25	20
Total wt of screen, lb	145 000	127 500	115 600	112 000	47 000	38 500	36 000	16 300
Gold-saving tables (g)	2 decks	2 decks	2 decks	2 decks	2 decks	1 deck	1 deck	1 deck
Total area, sq ft	6 066	8 000	7 500	7 000	3 600	1 800	1 560	750
Grade, transverse tables, in per ft	1 3/8	1 1/4	1 1/4	1 1/4	1 1/4	1 1/8	1 1/2	1 7/16
" longitudinal " " " "	1 1/4	1 1/4	1 1/4	1 1/4	1 3/4	1 1/8	1 3/8	1 1/4
Material	Steel	Steel	Steel	Wood	Wood	Steel	Steel	Wood
Width, transverse tables, in	32	31.75	31.75	31.75	28	31.5	29.75	36
" longitudinal " "	32	31.75	31.75	31.75	28	48	29.75	36
Pumps (i) Screen pump, gal per min	6 250	5 500	5 500	5 500	4 200	4 400	1 800	1 000
Motor hp	175	150	150	150	100	75	40	25
Table pump, gal per min	6 250	5 500	5 500	5 500	4 200	3 000	2 500	500
Motor hp	100	75	75	75	50	75	40	20
Hopper pump, gal per min	1 000	1 000	700	700	700	750	(h)	(h)
Motor hp	50	50	30	30	30	40		
Monitor pump, gal per min	3 500	None	3 500	None	2 000	None	None	None
Motor hp	250	.....	250	.....	125	.....	.....	.....
Total weight of dredge, ton	2 500	2 300	2 250	2 300	1 375	830	820	270

(a) Construction, all latticed girders. (b) All close-connected. (c) Lips of all buckets, manganese steel. (d) Including 1 head-line drum. (e) Not including motor. (f) Dredge No 7 has no spuds; on all others, spuds are box girders. (g) Riffles: dredge No 1, 1¼ by 1¼ by 1/16-in angles; No 2, 3 and 4, 1 by 1¼-in wood and 1/8 by 1¼-in bars; No 5, 1¼ by 1¼ by 3/16-in angles; No 6, 1 by 1 by 1/8-in angles; No 7, 1 by 1½-in wood and 1/8 by 1¼-in bars; No 8, 1 by 1¼-in wood and 1/8 by 1¼-in bars. (h) Uses screen pump. (i) For fire service, uses hopper or screen pump. * For swift flow'g river.

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127. Drift Mining

The term is applied to the exploitation of placers by underground

Field of use is in mining rich paystreaks of moderate thickness, cut methods are impossible or would yield a smaller net return. La must be broken by blasting; if numerous, they may increase cost which is prohibitive.

Examples of drift mining are found in the ancient buried channels in C buried beach placers at Nome, Alaska, and the deep leads of Australia (Art creek gravels, where frozen, are often drifted, especially by small operato California and Alaska mining, many placers were drifted for rich streak with small capital; later the same deposits were profitably hydraulicked (385). Drift mining costs more than open-cut work, hence requires rich yield a profit. Ordinary thickness of gravel mined is 5 to 8 ft; up to 10 or cases, and down to 3 or 4 ft, which is about minimum for economic work in

General plan. The methods of OPENING are the same as for o ground mining (Art 15–17), with emphasis on a mode of entry tha the workings and eliminate equipment and operating costs of h pumping. But tunnel or drift entries are limited to bench grave elevated deposits like the old California channels. Details of LATE OPMENT (Fig 618, 620) vary widely with local conditions. In genera haulageway is driven from the point of entry close to the long axis streak, and on or partly in bedrock; in California, this opening is entirely in bedrock and connected by chute-raises with the worki gravel above. Crosscuts are driven from the central drift to the channel, or limits of the paystreak; auxiliary drifts also are driv deposits. These openings have both a mining and exploratory fu latter often predominating in fixing the dimensions of the blocks MINING is done by a form of longwall-retreating (Art 108), called " advancing longwall has been used in a few cases. Ground is b picks where possible; by hand drilling and blasting where necessar gravel is thawed before it is excavated; attempts to use machine failed (455). The roof of working places is temporarily supported posts and head boards serve in firm, cemented, or frozen ground; forepoling may be necessary in loose gr rows are best for transport in small min low roof; otherwise tracks are laid alon ing faces and the gravel shoveled into ca are shifted bodily after the face has ad 8 ft. Equipment for haulage, hoisting, ing is the same as in metal mines of s and output.

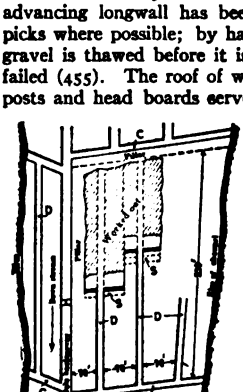


Fig 618. Layout of Drift Mine, Cal

Practice in California (ancient river c 116); data by R. L. Dunn and R. E. Brown, 1890 (456). Practice in the small number o now working does not differ materially from days. Fig 618 shows work in a wide part channel, Sierra Co, and illustrates the us Main haulageways *H* are 6 ft by 6 or 7 ft often requiring heavy drift-sets and close lag of these mines cover enormous areas, making nance of haulage openings a serious item. Cro auxiliary drifts *D* are 5 by 6 ft or 6 by 6 ft, ti lagged in soft ground, but not in as permanent a manner as the main haulage cuts *C* may be driven at an angle to the bedrock slope to secure the desired haula Breasting begins at the ends of drifts *D*, and retreats towards the crosscut from

of the working faces are indicated by dotted lines *S*. At ines, crosscuts *C* were 120 ft apart; in the latter, drifts ntain, development openings cut the gravel into blocks otect gangway *H*. In the Hidden Treasure mine, only sscuts *C* along the rim of the channel. These differ- s follows: At Red Point the gravel was hard and com- the breasts; but the distribution of gold was irregular, to define unprofitable areas. At Hidden Treasure the g ground, and all openings required timber; as the pay openings driven were those essential to mining. The d regular in value; the smaller blocks were doubtless eight breasted was only 4.5 ft. At Hidden Treasure, (120 eated were was with easts roof f ad- rking blast- easts er to

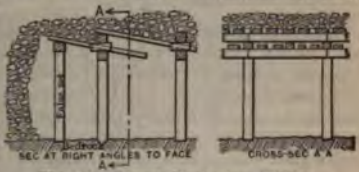
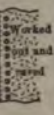


Fig 619. Breast Timbering, Hidden Treasure Mine

Fig 619 shows breast timbering at the Hidden Treas- oft gravel. In narrow paystreaks, California practice

ows a typical layout for paystreaks 75 to 100 ft wide. shaft or an adit; it usually requires timbering even in ts *C* are driven at intervals; in frozen ground they are carried the full width of the pay, retreating towards bered with sets in heavy ground, with posts and caps in mbered in solidly frozen ground. Purington describes bench placer 20 ft deep, on Anvil Cr, Nome: The pay-



streak, about 80 ft wide, consisting of 5 ft of gravel and 2 ft of bedrock, was opened by a central drift (6 by 6 ft) 600 ft long connecting with an adit. The drift was timbered with 3-piece sets 4 ft apart, of 6 by 6-in posts and 8 by 8 caps, lagged top and sides with 2 by 6 plank. Crosscuts were 40 ft apart. Breasts were timbered as in Fig 619; posts were 8 by 8 in by 7 ft, caps 10 by 10 in by 11 ft; posts were at he gravel was shoveled from plates into 21-cu ft cars. er exceeded 20 ft. The ground was so heavy that no ck was all recovered. Timbering cost 67¢ per cu yd ng 21 000 cu yd of gravel.

asts 50 to 100 ft long can be carried safely in winter; ked in small blocks. Timbering consists of 6 by 6 to cks and having short caps of 4 by 12 plank; posts are In winter, a space of 3 sets (9 to 12 ft) is usually kept round is ready to cave, the 2nd row of posts is lagged, Another slice is then taken off the breast. In sum- one time in this ground rarely exceeds 10 sq ft (390). of the Fairbanks district is like that at Nome; H. I. (458, 459). The gravel is nearly all frozen; shafts paystreaks over 50–60 ft wide, shafts are sunk near um aver tramming distance. The length of paystreak exceeds 400 ft (200 ft each way from shaft); greater transport to a point where it is cheaper to sink a new he length that can be worked downstream. Devel-

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opment and mining practice follow the general plan in Fig 620; if the wi
is 150 ft or more, an added longitudinal drift is usually driven on each si
drift. Little timber is used in breasting. Slabby roof is supported nea
row of stulls (Fig 621).

Fig 621. Timbering in Breast (after Ellis)

On Vault Cr, much of the ground is 200 f
streaks are narrow and irregular, but the gr
markably well. As it is important to follow
straight haulage drifts, driven ahead of stopi
ible. Fig 622 shows an advancing method
meet these conditions. A gangway is kept o
worked-out area by timbering with light 3-pie
lagging. Practice in other Alaska districts g
Fig 620.

Washing gravel. At mines opened by a drift or tunnel, th
are dumped at the portal into a bin feeding a sluice. Sluicing
tinuous; the bin is flushed out periodi-
cally with a small giant working under a
low head. A tunnel entry presupposes
rough topography and therefore natural
sluice-grades and dumproom. At mines
opened by shafts, the gravel is usually
hoisted to a sufficient height above the
surface to provide headroom for sluices
and tailings disposal. The gravel may
be dumped directly into a dump-box
(Fig 594) at the head of an elevated sluice, or into a bin arranged fo
sluicing. The latter plan is usual at small mines and where wate
costly. Bins usually hold a day's output; the upper boxes of the sluic
up after each run; frequent clean-ups are usually advisable at drift
of the relative richness of the gravel worked. In the far north, slui
sible in winter; the gravel is stacked on the surface and sluiced duri
summer. In California, tough cemented gravel is disintegrated in sta
with coarse screens, and then sluiced. Gravel containing much stick
passed through hexagonal trommels, or some puddling device, before e

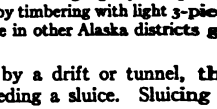

Fig 622. Advancing Syste

Thawing frozen gravel is done with steam or hot water or bo

Early work in Alaska and elsewhere was done with wood fires and ho
methods are still used by prospectors lacking other means of thawing.
enough
by 1 c
a capa
steam
obtaina
pecting
tory w
distri
390, 45

Table 128. Physical Properties of Frozen Material * (393)

* Aver of tests on 46 samples		Black, sandy muck	Gravel and sand	Bed-rock
Sp gravity, frozen		1.401	2.189	2.590
Sp gravity, thawed and dry		2.411	2.691	2.655
Sp heat, frozen		0.196	0.172	0.183
% ice, frozen ground	by vol	68.2	29.1	9.6
	by wt	44.7	16.0	4.26
% solids, frozen ground	by vol	31.8	70.9	90.4
	by wt	55.3	84.0	95.74
% voids, frozen		0.0	1.28	0.0
% voids, thawed		6.1	3.97	1.65
Lb ice per cu ft frozen ground		39.11	22.00	6.96
Lb solids " " " "		48.39	115.50	154.94

Phy
erties
gravel
tests
by H.
perma
gravel
dike,

lowing results; temperatures (Fah) of bedrock, 2° to 14°; of gravel,
black muck, 17° to 24°; of sandy muck, 19° to 24°. Mean temperat

ely on the kind of material and not on the depth, or on the presence or absence of muck overburden.

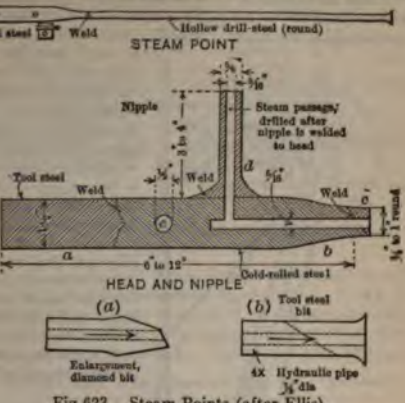

Fig 623. Steam Points (after Ellis)

with a hammer, and allowed to stand. Points he gravel thaws ahead of them; heavy blows are avoided, except where required by tight ground, as they may injure the point.

H. I. Ellis gives the following data at Fairbanks in 1915 (455). There are 3 types of bit. The diamond or square bit (*a*), Fig 623, is used in easy-driving ground, where stones can be pushed aside. The chisel bit (*b*) is for hard driving, requiring actual drilling; a cross-bit (like he chisel bit is hard to f 8, 10, 12, 16 and 20 ft ed). The 8-ft length is with short points, fol- breasts the points are d spaced There e driven ; steam e points allowed picked a face; wn and -in pipe the pipe the hole rocedure . Plan uch less hovelers thawing ers. In tely can together; but pointmen usually work in pairs, one point. Depending on the character of the gravel,

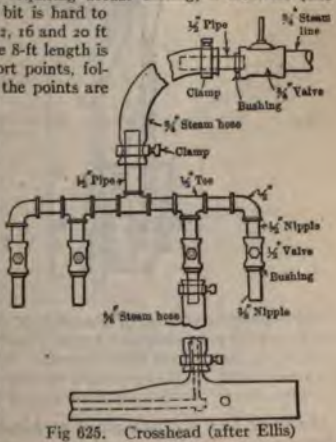

Fig 625. Crosshead (after Ellis)

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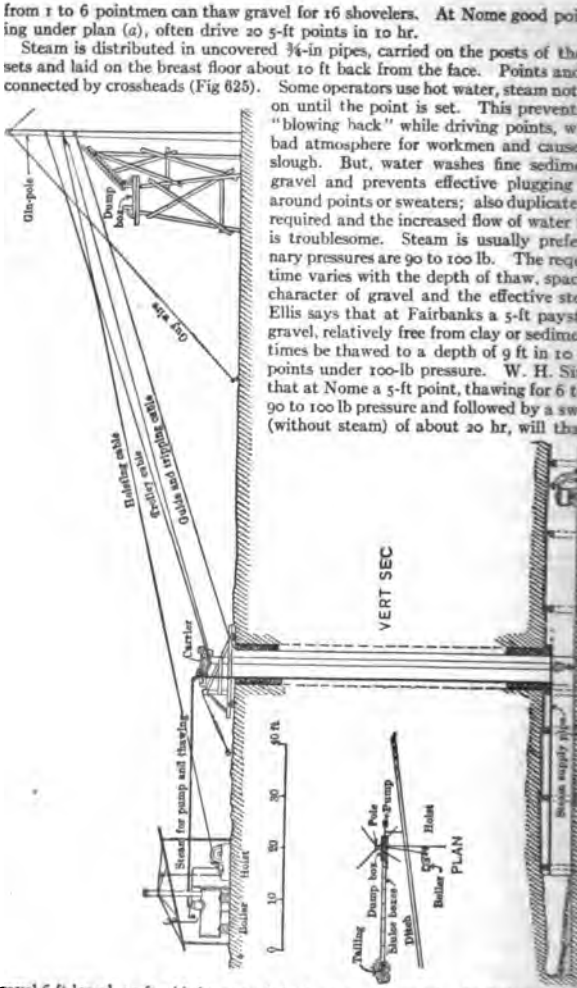
from 1 to 6 pointmen can thaw gravel for 16 shovelers. At Nome good poi
ing under plan (*a*), often drive 20 5-ft points in 10 hr.

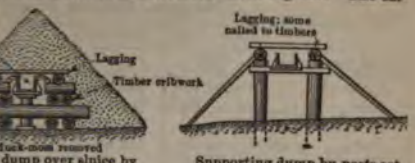
Steam is distributed in uncovered ¾-in pipes, carried on the posts of th
sets and laid on the breast floor about 10 ft back from the face. Points an
connected by crossheads (Fig 625). Some operators use hot water, steam not
on until the point is set. This prevent
"blowing back" while driving points, w
bad atmosphere for workmen and cause
slough. But, water washes fine sedim
gravel and prevents effective plugging
around points or sweaters; also duplicate
required and the increased flow of water
is troublesome. Steam is usually prefe
nary pressures are 90 to 100 lb. The req
time varies with the depth of thaw, spac
character of gravel and the effective st
Ellis says that at Fairbanks a 5-ft payst
gravel, relatively free from clay or sedim
times be thawed to a depth of 9 ft in 10
points under 100-lb pressure. W. H. Si
that at Nome a 5-ft point, thawing for 6 t
90 to 100 lb pressure and followed by a sw
(without steam) of about 20 hr, will tha



gravel 6 ft long by 3 ft wide by 3 to 4 ft high, or 2 to 2.5 cu yd. In drifting in t
the duty of 6-ft points averages 3.75 cu yd per 10 hr (408). BOILER CAPACITY
varies locally, but is about 1 b hp per point. When drift mining on the Third Be
was at its height, the b hp per point was from 1 to 2, aver 1.5; attention to d
duced this figure. For further detail, see cost data below and Bib (390, 408, 45

Hoisting and surface plant at Alaskan drift mines differs fro
mining practice in the extensive use of inclined cableways, the bucke
descends into the shaft (Fig 626).

levices, etc, see (390, 459); also Sec 26, Art 23. For for supporting the dump-box is replaced by a sluice, and supported and protected as in Fig 627. The car-


Lagging; some nailed to timbers
Lagging
Timber cribwork
Track soon removed
dump over sluice by that settling will not
Supporting dump by posts set in holes thawed in muck

Methods of Protecting Sluice under Winter Dumps (after Ellis)

390). Brown and Dunn give following data (456): s in hard gravel, requiring blasting, $4 to $7 per ft; 6 ft or 6 by 6 ft, $3 to $5 per ft. In ground less diffi- and in soft picking ground requiring timbering, main ary openings, $1.75 to $3. Table 129 shows effect on

Mines (Work done about 1888 to 1890 (Brown))

Mine No (*a*)	1	2	3	4
	Lo	*Hc*	*Hc*	*Mc*
	250	75	50	120
	4–7	2–14	2–7	2–12
)	25	35	25	30
	7 700	3 900	5 400	2 300
	100	66	66	66
le	70	60	.....	75
	Pc	*Db*	*Db*	*Db*
	Sl	*Mi*	*Mi*	*Sl*
	275	130	30	100
	12	.....	.....	.....
ays	12	12	4	7
	86	44	16	28
	.....	6	2	.....
rground (*b*)	120	130	27	50
	$2.15	$2.75	$2.70	$2.40
man-day (*c*)	3.20	2.95	1.87	3.57
	.....	20	10	.....
24 hr	.....	6.5	6.0	.....
	.....	325	325	175
er (*d*)	.....	$0.25	.....	.....
er (*d*)	.....	$0.35	$0.50	.....
milling (*e*)	$1.10 (*f*)	$3.25	$3.25	$2.00
(*g*)	$1.75	$7.00	$10.00	$2.50

. *Mc* = medium cemented. *Pc* = picking and caving. cing. *Mi* = milling. (*a*) Mine No 1, Hidden Treasure; No 4, Red Point. (*b*) Includes all surface and under nent. (*c*) Tons of pay-gravel per man-day, computed s following data on tons of pay-gravel loaded into cars Paragon, 2.7; Red Point, 2.7; Dardanelles, 1.95 (hard nita, 6.82 (soft ground, little waste). (*d*) Per ton. (*e*) ining and milling (or sluicing) per ton gravel delivered vel breast, not including management, improvements, ring periods of non-production. See (*f*). (*f*) Includes ed at surface. (*h*) Large stones thrown back from face.

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costs of local conditions at 4 typical mines (456, 460). NUGGET MINE, B
data from J. D. Hubbard (461). Table 130 gives costs in a buried chan

Table 130. Costs at Nugget Mine, Cal

	Aver mo	Max mo		Aver mo
Tons gravel mined	604	573	Tons gravel mined	604
Dynamite	$33.60	$80.96	Oil	$1.20
Fuse	5.64	11.30	Air pipe	4.60
Detonators	3.06	4.92	Water pipe	5.40
Rails	14.00	16.00	Wiring, lamps	1.30
Ties	2.16	0.88	Labor	479.25
Lagging	1.92	0.96	Superintend'ce	150.00
Timber	2.67	1.80	Deprec, 5%	100.00
Candles	3.45	5.27	Total	$830.89
Power, lights	22.64	22.00	Per ton	$ 1.374

an incline, hence involving hoisting and pumping expense. The gravel wa
ing," no cement being found. EXCELSIOR MINE, Placerville, Cal; data from

Table 131. Costs at Excelsior Mine, Cal (10 340 tons cement gravel mined and milled, 1907)

		Per ton
Breasting	Labor	$0.989
	Powder	0.031
	Fuse, caps	0.010
	Candles	0.020
	Timbers (a)	0.039
	Lumber	0.045
	Lagging	0.010
	Wedges	0.002
	Hardware	0.002
Total breasting		$1.148
Milling		$0.158
Total		$1.306

Note. Development work in 1907: 131 ft bedrock drift @ $3.33 per ft; 472 ft drifts @ $3.85; 666 ft incline @ $4.33; 78.5 ft raises (in gravel, 4 by 5 ft, about 6 ft high) @ $3.33.

for 1907. Cement gravel occurs i
channel; bedrock is slate and soft
which from 1 to 1.5 ft is taken up.
pay, 0 to 16 ft; aver, 6 ft. Breastin
points about 5 500 ft from the mou
tunnel. The pay-gravel was milled
mill. Wages (9-hr shift): miners, $2.
$3.50; night boss, $3; millmen, $3
(day); foreman, $125 per mo.

Nome, Alaska; data by A. Gibso
462). Tables 132–134 show data fo
drift mines in frozen gravel. All
done during night shift; mining, in cl
Water for sluicing was pumped by
distillate engines; mines No 2 and
the same management and had a con
ing plant; head on pumps about t
mines No 4 and 5. Other conditio
costs were as follows. Mine No 1:
done in winter; 1.5 ft of bedrock was
much as possible of the winter dump
was hydraulicked into sluices, the re
eled in; sluice tailing was removed
scrapers. Water was pumped day a
18 days in larger amounts than necessa

No 2 and 3: All summer work; 1.25 ft of bedrock was mined. Gravel w
into bins and sluiced intermittently (about once a day). Mine No 4:
work was done in early spring; all mining and sluicing in the summer; 1.5 ft
was mined. Gravel was handled on the surface as at No 3. Mine No 5:
summer work as in No 4. Pay-gravel averaged 2 in above, and 2 ft in, bedroc
was dumped into a mud-box; sluicing was continuous, hence pumping was
on day shift. The bedrock in No 1, 2, 3, and 4 is mica schist; at No 5, black
waste or overburden at No 1, 2, and 3 is coarse gray sand; at No 4, light
sand and clay; at No 5, a fine light red gravel or gray sand. Gibson concludes
data that thawing and mining cost more in winter than in summer.

Fairbanks, Alaska; data by F. J. Katz, 1913 (400). 80% of the prod
posits lie at depths of 40 to 260 ft; all lie in valley bottoms which have flat grad
of them are solidly frozen. Transport is difficult and water supply meager.
mining is done by leasers on blocks extending 500 to 1 000 ft along the pay
containing 50 000 to 200 000 sq ft of workable ground. Individual 20-ac
1 000 to 1 320 ft long, are worked by 2 or 3 independent plants. On group

n Drift Mining, Nome, Alaska (Gibson)

Mine No.....	1	2	3	4	5
.....	53	81	81	45	50
.....	2.5	2.5	2.5	3.5	2.166
.....	1.5	2	2	1.5	2.333
.....	4	4.5	4.5	5	4.5
.....	45	70	70	35	50
.....	128	205	257	128.6	327.1
cu yd.....	80	114	143	90	157.5
).....	18	16	24	13.5	13.5
isted per day.....	120	192.4	161	180	315
.....	7	7	7	6	7
.....	40	90	90	25	46
.....	9	12	12	8	11
.....	1.12	0.78	0.78	1.4	1.08
.....	8	7	9	7.5	7.5
.....	1	2	2	....	2.5
lay.....	3.2	2.28	2.85	5.14	7.11
.....	168	168	168	84	210
.....	52.5	42	42	31.5	52.5
.....	1.3125	0.8195	0.6537	0.6533	0.6419
.....	0.656	0.368	0.294	0.35	0.333
ravel.....	2.756	1.842	1.469	1.283	1.667
sluicing water, gal	132.8	30	30	14	18
, gal.....	2.73	3.80	4.77	6.43	8.75
.....	$2.97	$3.30	$3.30	$2.90	$2.71
ts.....	24.25	25.17	25.17	25.5	25.83
.....	3	3	4	2	2
.....	1	1	1	1	1
.....	1	½	½	1	1
.....	1	1	1	1	1
.....	1	1	1	1	1
.....	16	17	17	9	20
.....	(*f*)	½	½	....	1
.....	(*f*)	1	1	1	....
.....	....	25	26	16	27
an-day)					
ravel (*b*).....	32.0	51.25	51.40	42.85	109.04
avel (*c*).....	7.11	10.79	13.53	11.69	14.87
).....	4.44	6.0	7.53	8.18	7.16
ing labor).....	18.11	76.0	95.33	90.0	137.5
icing (*e*).....	....	4.65	5.61	6.0	6.06

u yd are of loose gravel, not place measure. (*a*) Bucket 626). (*b*) Pointmen and fireman. (*c*) Foreman, en- ers only, 8, 12.06, 15.12, 14.28 and 16.36, respectively, engineer and laborers. Duty, laborers only, 5, 6.7, ctively. (*e*) All labor but manager included. (*f*) 10 er part of winter dump was being sluiced; 21 men per to shovel-in.

the paystreak, 8 leases are in operation. Individual 000 sq ft of bedrock per year, hence the life of a lease years.

hp boiler, costs for machinery and supplies about $7 000. ing installation for sluicing, each of which costs about plant add about $3 000. Sinking and timbering shafts s 6 by 6 ft to 7 by 7 ft in the clear; opening and tim- add $2000 to $5 000. Total preliminary expense for east $15 000. These figures are based on costs in 1908.

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Table 133. Costs of Drift Mining, Nome, Alaska (Gibso

		Cost, cu yd of pay-gravel
	Mine No.....	1	2	3	4
Thawing	Crude or fuel oil.....	$ 14.85	$ 11.58	9.23	$ 6
	Labor and board.....	25.62	24.40	24.41	22
	Repairs and renewals.....	9.38	6.58	5.24	2
	Total.....	$ 49.85	$ 42.56	$ 38.88	$31
Mining	Crude or fuel oil.....	$ 4.64	$ 2.89	$ 2.31	$ 2
	Management and board.....	14.06	4.86	3.87	12
	Labor and board.....	99.37	103.97	82.88	77
	Repairs and renewals.....	9.37	6.58	5.25	8
	Total.....	$127.44	$118.30	$ 94.31	$101
Sluicing	Distillate.....	8.89	6.62	5.28	3
	Management and board.....	3.10	.....	.....	....
	Labor and board.....	42.64	8.44	6.72	6
	Repairs and renewals.....	9.27	2.19	1.75	1
	Total.....	$ 63.90	$ 17.25	$ 13.75	$ 12
Total operating cost.....		$241.19	$178.11	$146.94	$145
Value of mining plant.....		$5 000	$ 7 500		$2 0
Value of pumping plant.....		2 500	2 500		1 2
Total value of plant.....		$7 500	$10 000		$3 2

Note. Costs do not include interest and amortization charges, nor costs ing and preparatory work.

Table 134. Wages * and Board †, Nome, Alaska

	Winter	Summer		Winter
Pointmen.....	$4	$6	Common labor.....	3
Asst pointmen.....	.....	5.50	Cook.....	4†
Firemen.....	3.50	5.50	Cook's helper.....	.....
Foremen.....	5	6.50 to 7	Manager.....	10†
Engineer.....	4	6	Cost of board.....	(a)

* Per 10-hr shift. † Per day. (a) At the 5 mines cited in Table 133, resp for board per man-day were $1.25; $1.08; $1.08; $1.15 and $1.35. This i and cook's wages; company pays wages plus board.

Wages in 1908 were board and $5 per 10-hr day for common mine labor, pointmen, $7 for hoistmen; at a few mines, $5 for 8 hr, or $6 for 10 hr. Boa $2.50 per man per day, always included. Wood fuel costs $4 to $12 (gene $10) per cord, cut and delivered. Gin poles cost $50. Water for sluicin costs $20 per "sluice-head" per day. Cost of pumping water for sluicing a larger plants is stated as $30 per day.

Total cost of drift mining (25 estimates by operators) ranges from 50¢ to of bedrock; aver about $1. Work covered by Table 135 was done prior to

Table 135. Drift Mining, Fairbanks, Alaska (J. F. Newsom

(a) For 17 claims (b) For 14 claims	Fairbanks Creek			Cleary C	
	Max	Min	Aver (a)	Max	Min
Total depth gravel, ft.....	55	14	26.7	90	12
Width of pay, ft.....	450	70	193	350	80
Height gravel mined, ft.....	16	2.5	4.9	9	2.5
Depth bedrock mined, ft.....	3	1	1.3	2.5	0.5
Total boiler hp.....	50	15	.....	50	9
Hp of hoists.....	48	6	.....	38	6
Operating cost per sq ft.....	$1.30	$0.75	$1.00	$1.30	$0.65
Total cost per sq ft.....	$1.40	$0.87	$1.13	$1.51	$0.8
Total cost per cu yd.....	$7.85	$2.44	$5.40	$9.70	$2.5

cost at 50¢ to $1.25 per sq ft of bedrock; aver, 75¢. In
lers is taken at 30 to 35 sq ft of bedrock per 10 hr, equiv-
lace. Output per shoveler is 75 to 150 barrow loads per
inary gravel, working conditions and wheeling distance.
The 1916 Canadian Gov't report on Yukon Terr con-
Cost of sinking an 8 by 8-ft shaft, 30 ft deep, in frozen

of Plant * for Drift Mining, Klondike

$1 300	4 buckets	60
375	10 wheelbarrows	100
300	100 ft ⅜-in steam hose	65
450	12 shovels	18
100	200 ft ¾-in pipe	24
38	Miscellaneous tools, etc	125
50	Total	$3 205
200		

r 10 shovelers if pumping is required, or 16 shovelers if
sed on smaller plants, employing only 5 or 6 shovelers;
d with consequent reduction in first cost. (*a*) Scotch
turn flue.

n one case $9.16 per ft, including timbering. Wages,
lay. Advance per 12 hr, 7 ft. Another shaft of same
windlass for $9.96 per ft. A 6 by 6-ft drift in frozen
el was hoisted by a self-dumper. Advance per 12 hr,
the Indian River watershed showed an operating cost
, of which shovelers wages were $60; other wages, $49;
ed 590 sq ft of bedrock per 10 hr, the pay being about
er cu yd, $2.70. At another mine 16 shovelers cleaned
ift, the pay being 3 to 4 ft thick. Operating cost for
a self-dumper and sluicing, was 31¢ per sq ft.

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ECTION 11

OUND TRANSPORT

BY

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eneral Considerations

derground transport of mineral, waste and sup-
ine operation. Though it usually appears as an
full importance is seldom indicated, and, in some
intenance of workings used primarily for transport
total cost of transport would be shown to exceed
d operation.

put of a mine can be brought to one level or gang-
etimes be so concentrated that hand tramming can
nal haulage, or the latter by mechanical haulage.
ue to shape of the deposit or distribution of work-
evels is lowered to one main haulageway, the in-
be more than offset by the saving due to a cen-

led by comparing the total capital and operating
d. If these involve changes in mine development,
hat saving in development and maintenance costs
centralized system, which reduces the num-

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length or size of sub-levels, crosscuts, laterals, shaft stations and trackage wise required. Such saving must be balanced against cost of installin maintaining the proposed haulage system.

General formulas. Let D = useful duty per shift, mineral-ton-miles; G = gro per shift, ton-miles; d = distance mineral is to be moved; ft; s = aver speed moving, ft per min; t = time loading and discharging per trip, min; T = mineral per trip, tons; T' = tare per trip, tons; C = time of 1 complete cycle, min; m = working time per shift, min.

$$D = \frac{m}{\frac{2d}{s} + t} \times \frac{dT}{5\,280} = \frac{mdT}{5\,280\,C}, \quad \text{and} \quad G = D + \frac{m\,2\,dT'}{5\,280\,C} = \frac{md\,(T + 2\,T')}{5\,280\,C}$$

Useful duty D varies directly with distance, speed and wt of mineral per trip, versely as the time required at terminals. If the gross duty limit has been reac useful duty will vary inversely as $(T' \times 2)$. These principles apply to all kinds o port, from shoveling to locomotive haulage. Despite the rule that cost per t other things being equal, varies inversely as the unit load, underground conditions require small-scale equipment. But many large-tonnage installations exist, e in coal, iron and copper mines, and methods now vary from the most primitive to of standard R R equipment.

2. Primitive Transport Methods

Packing. Mineral is carried in sacks, skins or baskets on backs or heads of m women, in primitive countries, or occasionally in narrow, tortuous workings, or i gency work. Unit load is usually 60 to 150 lb, but, in Latin America and the peones and coolies can carry more. Duty varies widely, but approximates 12 to hr per ton-mile. Boxes or hand-barrows, holding 200 to 700 lb, and carried by 2 men, are slightly more economic, but if there is room for such practice better met usually feasible.

Shoveling. Transport by carrying or casting with shovels is prop as a last resort. One man can move 1 ton 100 ft in about 2.5 hr, equiv 125 to 160 man-hr per ton-mile. (MUCKING is a common term for under shoveling in stopes or development work.)

In stopes, with chutes 30 ft apart, the standard duty in many mines is 20 cars (tons) per shift, including sledging or blockholing lumps, and stowing filling. chutes are advantageously placed with reference to the mineral faces, shovelers h to assist miners, but if they have also much sorting to do, their output may b reduced. (For mucking in connection with tramming, see Art 9.)

Wheelbarrow transport, exclusive of loading, costs at least 1 man- ton-mile, and is justified only in temporary work, or where first cost of tr equipment is not warranted. Short exploratory workings and small, l stopes, where the only alternative is shoveling, are the only proper field barrow in mining work.

Best mining barrow has 3 cu ft capacity, No 14 to 16 B & S gage pressed s handles, frame, and wheel-guard of steel pipe; max height 21 in, wt 70 to 80 $35 to $45 per doz, fob factory. For surface work in very hot or cold climat handles are preferable. For thin seams a flat barrow without legs is used.

3. Mine Cars (1, 2, 3)

Standard design. The importance of the underground car justifi care in design or selection. If possible, a single design suited to local co should be adopted. This cannot be done where hand-trammed, s or sub-level cars and larger mechanically hauled main level or adit all used, and in large mines several designs and sizes are employed. compromise design is feasible, standardization is advantageous in sim repairs and the stock of repair parts.

el or composite construction. Wooden cars are bulky, are less durable, but are more easily repaired than steel. ing, and in collieries the number of composite and steel osite cars usually have wood stringers for the truck, and r is hinged on its truck for dumping in one direction, or ection to truck to dump in any direction. When con- e centered over the wheel-base, to equalize wheel loads ping.

rs are simpler and usually lower than others of ease of loading because of the low sides, simplicity, pacity to wt. Disadvantage: they can be dumped

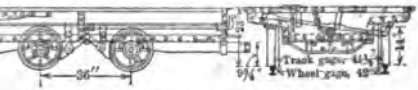
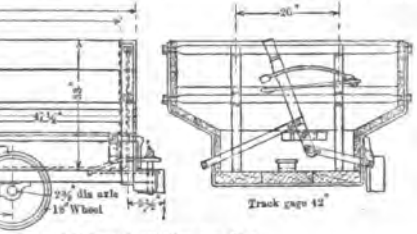


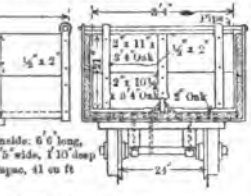
Fig 1. Watt All-steel, "Low-vein" Car

capac, level full, 43 cu ft; price, $65 to $75 fob factory, arings. Metal miners could often advantageously adopt



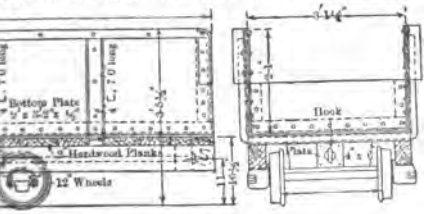
ockensmith Standard Wooden Car

this design. Fig 2 is a standard car of Connellsville, Pa, district; wt, 2 200 lb, capacity, 40 cu ft. Fig 3 is a wooden car used in sub-levels in the Mesabi iron range; simple, cheap and easily repaired. Fig 4 is a car 9 ft long, open at both



Mesabi Iron Range

ngers, 2-in wood bottom with 0.5 in plate liner, outside 100 lb. At stations of inclined shafts it is dumped by a



Trimountain Copper-mine Car

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cradle directly into skip. Fig 5 is an all-steel, 16-cu ft, rigid-body, sub-level car largely supplanting Fig 3. It resembles revolving car in outline, but is about

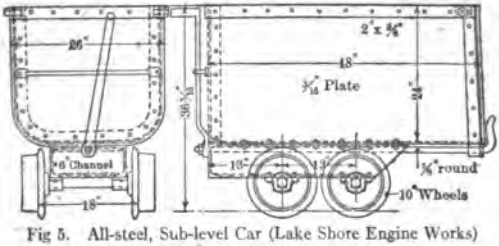


Fig 5. All-steel, Sub-level Car (Lake Shore Engine Works)

Hinge car (Fig 6) one direct For easy by 1 man should be over 2.5 i of load ce

Fig 7 is 12-cu ft ca Copper Qu Fig 7a is

type from a Michigan copper mine; body is open at both ends; capacity, 40 cu Fig 8 (Truax car) has a longitudinal hinge permitting side dump only. To pre

turning a patented rail clutch is provided, as shown.

V-body, supported on trunnions, may be single or double side-dumping (Fig 9). It is little used underground, but for soft, fine ore, discharged at the side, it may be better than the revolving type. The *V* can be made high and narrow, or low and wide as required by gangway and chute clea has no doors and can readily be made water tight.

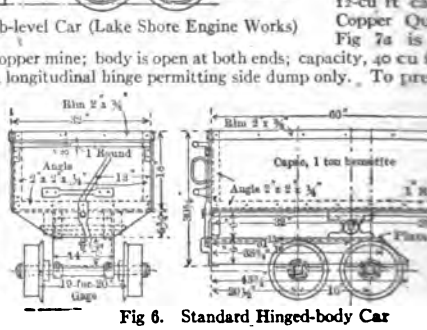


Fig 6. Standard Hinged-body Car

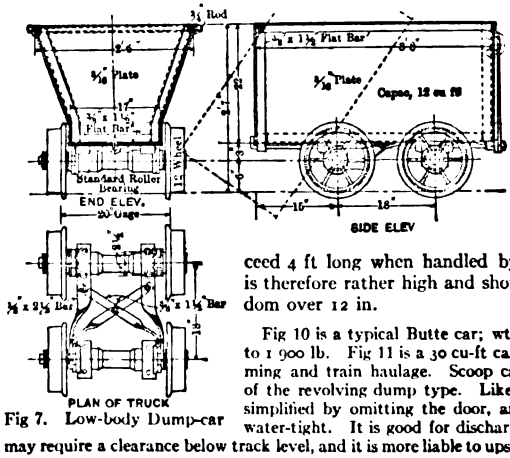
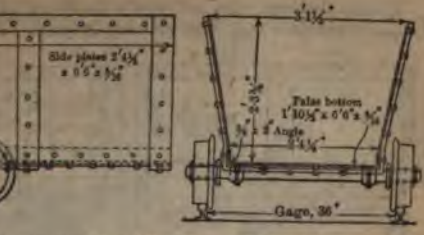


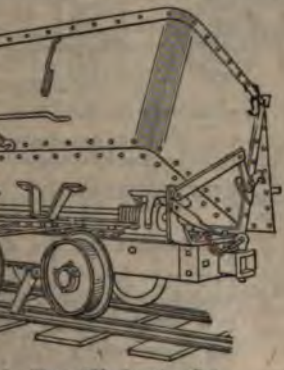
Fig 7. Low-body Dump-car

Revolvi car can any direc hence is metal mi must be hi for the b clear the when side To avoid overturni body shou ceed 4 ft long when handled by 1 man. is therefore rather high and short. Whee dom over 12 in.

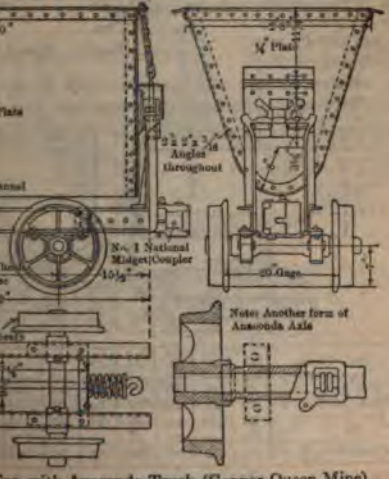
Fig 10 is a typical Butte car; wt, 965 lb; c to 1 900 lb. Fig 11 is a 30 cu-ft car for both ming and train haulage. Scoop car (Fig 12) of the revolving dump type. Like the V-bod simplified by omitting the door, and can the water-tight. It is good for discharging into a may require a clearance below track level, and it is more liable to upset when sid



d Dumping Car (Mich Copper District)



8. Truax Side-dumping Car



Car, with Anaconda Truck (Copper Queen Mine)

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Hopper-bottom cars are rarely employed except for mechanica and where loads are too large to dump by hand. They are unusual un but are transfer s charging the rails.

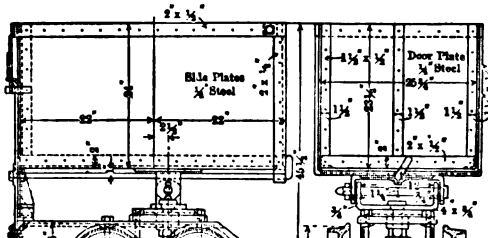


Fig 10. Revolving Dump-car, Butte, Mont

Gable "saddle- holding 1 are now main leve haulage n charging sides they clean and pockets e 13 is a 7- car; wt, 3 load soft i lb. Fig 1 view of a similar 45 cu-ft car, at an Ariz copper mine; it has an improved Capacity of these cars may be increased by adding a strip 6 to 9 in high arou

Special bodies are required for drinking water, toilet and man cars. Standard trucks with removable frames are best for drill steel, wedges and powder, and with side stakes for timber, drills and large supplies.

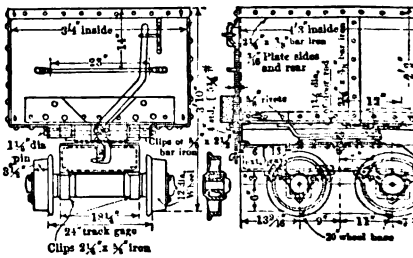
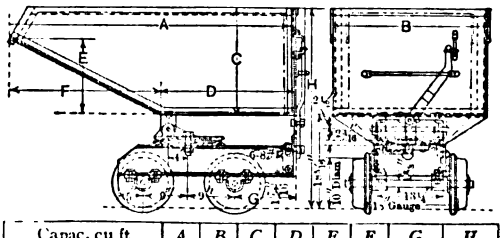


Fig 11. Miami (Ariz) Mine Car

Steel colliery cars have recently been introduced by many of the Pa anthracite companies; some all-steel, others with wooden truck frame and sheet steel bo others have composite bodies, of wood and steel (Fig 15, 16). Wei from 3 75 lb; capac cu ft; co $133.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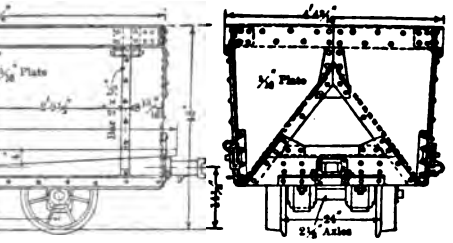


dle-back Car (Oliver Iron Mining Co)

of axles, when a wooden car bottom carries journal boxes. ich iron mine have the axle boxes carried flexibly by light ed that one stacle on the UBLE-TRUCK, rge loads; it ght rails, and a car of same ig 17 is a 12- conomic, and will increase l 96, p 1170). urves of mine short; rarely eptions: loco- are of large ope haulage, revent derail- ase to radius k gage, Art 5.

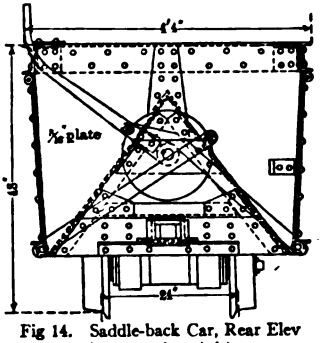
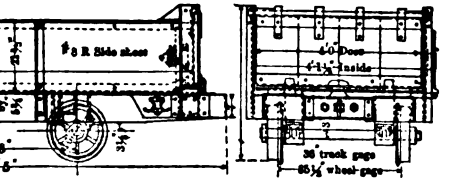
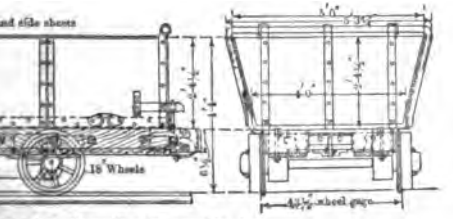


Fig 14. Saddle-back Car, Rear Elev (copper mine, Ariz)



Steel Colliery Car (L V Coal Co)



Steel Colliery Car (Shamokin, Pa)

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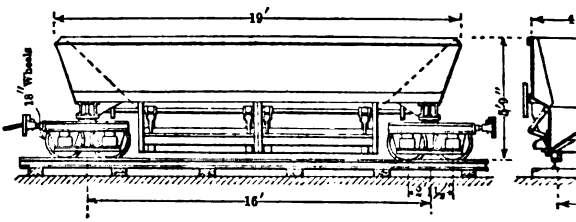


Fig 17. Double-truck Car (Crown Mines, Transvaal)

4. Mine Car Details (2)

Draw-bar, coupling, and bumper. When cars are to run in t frame or body must transmit and withstand both tractive and impac For light cars, separate draw-bars should be protected by bumpers, a signed for 10 times the nominal locomotive draw-bar pull, to allow braking and abuse.

Small metal-mine cars are sometimes coupled by chains and rings at the ver of body (Fig 11). This requires 2 connections, but on curves the outer chains tension, curve friction is supposedly reduced, and couplings are more accessi of 2-ton capacity or over should have combined spring couplings and heavy C- (Fig 9, 13). 1 spring and 1 rigid coupling make a good combination for light cars. should be rounded to avoid interlocking on curves.

Car wheels. MATERIALS: C I, cast, pressed or manganese steel. Sta wheels are of definite composition, annealed to give toughness, and with dee flange and tread. The treads are too hard to be trued on a lathe. Hubs are worn, but readily machined and bushed. When scrapped they are salable to l ries. Cast-steel wheels are at least 30% lighter than C I; are tougher, not breal impact (as iron wheels may), wear more rapidly on the tread, but are easily In remote regions they are worth less per lb as scrap. Wheels of pressed-steel tougher and lighter than cast steel, easily machined, but cannot be made in one self-oiling recesses. Some mines give satisfactory reports of them. Manga wheels are hard and tough, and outwear any others of the same design. They cannot be machined with tool steel, and in many mining districts make unsala DESIGN. Diam, 8 to 20 in; tread, 2.5 to 3.75 in, coned $\frac{1}{16}$ to $\frac{3}{16}$ in (M C B coning is 1 in 20). When treads become grooved, wheels should be trued on discarded; grooving causes derailment on frogs and increases tractive resistance, on curves. Flanges are from $\frac{7}{8}$ to $1\frac{5}{8}$ in deep. M C B standard gage is m $\frac{5}{8}$ in above base of flanges, and is $1\frac{3}{16}$ in less than the straight track gage.

Wheel bearings. LOOSE WHEELS are held on the axle by cottars or linch-pins by bolts or key blocks engaging annular grooves in axle (Fig 18), or have outside boxes and inner axle collars (Fig 4). With finished collars, hubs should be co to fit. Hub bore may be machined, or left rough for babbitting, or reamed out steel, brass or bronze bushing (Fig 19). ROLLER-BEARING WHEELS (3) are he reduce tractive resistance, and are now widely used on colliery cars. They re inspection and lubrication. SELF-OILING WHEELS have hub caps, thus maki reservoir (Fig 11), or are cast with reservoirs in the hub, spokes (or wheel disks), feed holes to the journal, and a screw or spring-plugged supply hole (Fig 10, 18). many designs on the market. TIGHT WHEELS may be keyed, or bolted through cro in the axle; but a close fit is better, the wheel being forced on the axle by a hydra

Axles may be fixed or rolling. Simplest and cheapest is a square axle bolte frame or car bottom, with outside loose wheels (Fig 10). As there is no protec dust or oil leakage, the hub bore wears rapidly, increasing friction and liability ment. A hub cap keeps out most dust and saves oil; bushing takes up wear an friction. Roller-bearing wheels, with long bearings, are excellent on fixed axles, are often used in pairs on rolling axles, which is not a logical design; rolling a

tation positive. Axle boxes often have no caps, and even . Open pedestal boxes with axle held by a U-bolt are heap, but the axle wears rapidly. e better. Many dust-proof bear- from simple shrouds, multiple felt e continuous sleeve enclosing the es, as the Anaconda (Fig 9) or

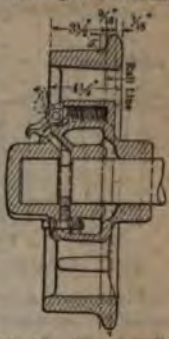
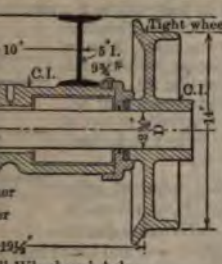


Fig 18. Faught Self-oiling Wheel

standard R R practice, the tread the increased length of the outer flanges crowding against the rail, smaller diam. With the short- eads cannot be sufficiently coned hus, on 24-in gage, a 20-ft radius wheel to travel 3 ft or 10.5% far- ed wheels must therefore either is avoided by loose wheels. Or,), both wheels are tight, but are inuous sleeve journal. Each axle or saddle key, riding in a slot cut the design is the 2-piece axle and ts. McCaskell truck (Fig 19) has r shroud between bearings, and a solid axle, with 1 tight heel gives differential travel on curves, so that hub wear is negligible, or is provided for by bushing. Cased roller bearings for the axle (3), and a self-oiling loose wheel, would give an ideal combination to minimize tractive resistance.



ll Wheel and Axle

Lubrication of wheels (1). Thorough lubrication is essential for low tractive resistance. An oil bath on a standard R R axle reduces friction to 10 to 15% of one side of the journal. Open, unprotected pedestal every trip, and, when used, there should be oil reser- with revolving brushes or cams to oil or grease axles ver.

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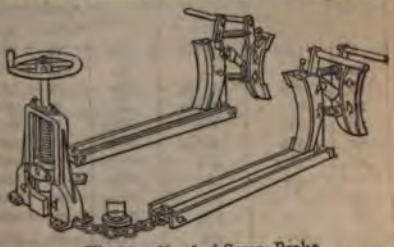
Fig 20. Vertical Screw Brake

is a hardwood sprag, thrust between spokes and forcing at wheels and is usually applied to but 1 wheel at a time. n wheels brakes 2 wheels on one side. A simple steel-strap

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brake is shown in Fig 3. Brake shoes of wood or steel, in chairs, with toggl connected under body to engage all 4 wheels, are best (Fig 1, 2). Brakes sh ratchet, latch or counterweight, to set or hold them. Fig 20 shows brake

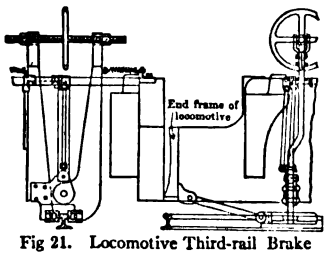

Fig 21. Locomotive Third-rail Brake

Gen Electric Co's trolley locomot comotives on steep tracks a speci is sometimes provided to grip bo third brake-rail to control train is said this brake will stop wi a 100-ton train and locomotive miles per hr down 8% grade.

Making *vs* buying cars. S as for drill-steel, timber, wate men, can advantageously be m mine shops during dull period nal labor cost; but, for star even the best appointed mine rarely compete with mine car makers, either in cost or quality of product rates on manufactured cars are usually higher than on lumber or down car parts.

5. Mine Track (4, 6)

Wooden rails, 2 by 3 in or larger, are often used in metal mines, for small-s porary work, and in collieries for branch track in rooms or breasts. Friction to sliding of braked wheels on heavy grades is greater on wooden rails and le to wheels. In its simplest form, the rail is spiked to the ties by 4-in or larger wire nails. A better form, for convenience in relaying, holds the rail by wedges in daps in the ties (Fig 22). Facing wooden rails with strap iron increases their life, but reduces sliding friction practically to that of the T-rail. Straps are ⅜ or ½ by 2 in, fastened by countersunk spikes or screws.

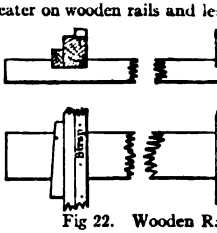

Fig 22. Wooden R

Steel rails are made in 3 standard series of sections, the Amer Soc C E, and A and B series of Amer Ry Assoc; also in many special shapes. For mine track thick web and broad base are best, because of t ties, and frequency of corrosive water. Amer Soc C E sections most n

Table 1. Rail Sections (Amer Soc Civ Eng Standard)

Wt, lb per yd	$b=d$	c	t	Long tons per mile single track		Lb per 100 ft	Size of hole, in
				ton	lb		
8*	1 9/16	1 3/16	5/32	12	1 280	267	5/8
12	2	1	3/16	18	1 920	400	5/8
16	2 3/8	1 11/64	7/32	25	320	533	3/4
20	2 5/8	1 13/32	1/4	31	960	667	3/4
25	2 3/4	1 1/2	19/64	39	640	850	13/16
30	3 1/8	1 11/16	21/64	47	320	1 000	13/16
35	3 5/16	1 3/4	23/64	55	000	1 167	13/16
40	3 1/2	1 7/8	23/64	62	1 920	1 333	7/8
45	3 11/16	2	27/64	70	1 600	1 500	7/8
50	3 7/8	2 1/8	7/16	78	1 280	1 667	1
55	4 1/16	2 1/4	15/32	86	960	1 833	1
60	4 1/4	2 3/8	31/64	94	640	2 000	1
65	4 7/16	2 13/32	1/2	102	320	2 167	1
70†	4 3/8	2 5/16	33/64	110	000	2 333	1

Letters refer to Fig 23. * Few mills roll this size. † Heavier rails, up to 150 lb,

are sold by the long ton, specifications usually call-
h not over 10% of shorts down to 24 ft. Shorter
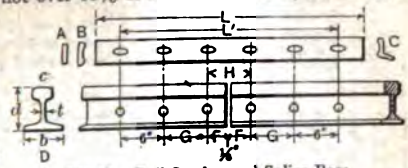
Fig 23. Rail Section and Splice Bars

G, *H* and *l*, refer to Fig 23. $G = 2F$; $H = 2F + 1/8$ in;
ail; *L* for 6-bolt joints, for 70 lb and heavier.

ENGTH: Fig 24 shows minimum wt of rail and max
ed for 1½ and 3-ft tie spacings, to give reasonably
an indicated for given loads will reduce track resist-
ance and maintenance costs, while lighter rails should be used only for temporary work. For locomotives, rails lighter than 25 lb will not keep alinement, and should not be used on main lines. FOR CONDUCTIVITY: rails for heavy traffic are usually amply large for return circuit for electric haulage, but with light equipment the size of rail for economic conductivity, rather than the wheel loads, may be the limiting factor in determining wt of rail (Sec 42, Table 3).

½ and 3-ft Tie Spacing

Bonding. The contact resist-
o great as to be equivalent to an open circuit. The best of
lice bars, and rails should be cross-bonded at about every
eing connected to the negative pole to minimize corrosion.
rt 8.)

nearly as possible the same strength and stiffness as the
ed for by leaving spaces between rail ends, using splice-bars
illing the rail holes ⅛ to ¼ in larger than the bolts. Flat
3, *A* and *B*) are made for rails up to 40-lb, but for rails over
; 23, *C*) is better.

he greatest holding power, should be straight, smooth,
with a sharp cutting edge beveled back 2 diams.

ary over 10% in wt; number per keg should be verified when
e for extras. Spikes are staggered on opposite sides of the
for the other rail, the inside spikes being near the same side of
ith holes ⅛ in smaller than the spike, the holding power is
le, spikes must be pointed instead of beveled.

e of ties, are more effective, but cost more. Though much
spensable for tropical hardwood, they are not yet common in
a spikes on some R R lines. Holes must be bored for them.

d-gage track are 8 to 9 ft long, not less than 6 in thick,
ace, or, if sawed, minimum cross-sec is 6 by 8 in. For

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Table 2. Rail Joints (Cambria Steel Co)

Wt of rail per yd, lb	No of joints per long ton: 24-ft rails	No of joints per long ton: 33-ft rails, 10% shorts	Track bolts*: Size, in	Track bolts*: No per 100-lb keg	Splice bars, wt per pair, lb†	Wt of rail per yd, lb	No of joints per long ton: 24-ft rails	No of joints per long ton: 33-ft rails, 10% shorts	Track bo: Size, in	lt
8	35.00	25.86	½×1¾	952	....	40	7.00	5.17	⅝×3	
12	23.33	17.24	½×1¾	952	2.58	45	6.23	4.60	¾×3	
16	17.50	12.93	½×2	909	4.53	50	5.60	4.14	¾×3½	
20	14.00	10.34	½×2	909	3.71	55	5.09	3.76	¾×3½	
25	11.20	8.27	⅝×2¼	488	5.60	60	4.67	3.45	¾×3½	
30	9.34	6.89	⅝×2½	465	9.10	65	4.31	3.18	¾×3¾	
35	8.00	5.91	⅝×2¾	435	11.1	70	4.00	2.95	¾×3¾	

* Track bolts of different makers may vary over 10% in wt. Those in th square nuts; for hexagon nuts, add 5% to No per keg. Length does not allow

† For 65-lb rail and lighter, 4-bolt joints; others, 6-bolt. Fish-plates f lighter (see *A* and *B*, F splice bars (*C*, Fig 23) and heavier. To join ent wts, as may be req ing from main to b special offset fish-plat chairs are used (Fig 2

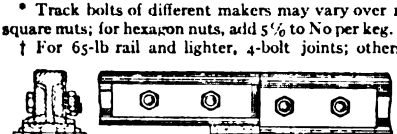

Fig 25. Offset Fish-plate

Table 3. Track Spikes

Size under head, in	No per 200-lb keg	Ties 2 ft centers, 4 spikes each: Spikes, lb per 1000 ft, single track	Ties 2 ft centers, 4 spikes each: Kegs per mile, single track	Suitable rail, lb per yd	Size under head, in	No per 200-lb keg	Ties 2 ft centers, 4 spikes each: Spikes, lb per 1000 ft, single track	Ties 2 ft centers, 4 spikes each: Kegs per mile, singl track
2½×5/16	2 230	179	4 74	8 to 12	4½×7/16	690	580	15.4
2½×⅜	1 650	243	6 40	12 to 16	4 ×½	605	652	17.5
3 ×⅜	1 380	290	7 96	12 to 20	4½×½	518	772	20 4
3½×⅜	1 250	320	8 70	12 to 20	5 ×½	475	841	22 3
4 ×⅜	1 025	390	10 22	16 to 25	5 ×9/16	405	988	26.1
3½×7/16	890	450	11 88	16 to 25	5½×9/16	360	1 120	29.3
4 ×7/16	780	515	13.65	20 to 30	6 ×9/16	320	1 250	33.0

narrow-gage track, length of ties should be twice the gage, at least than spike length, and 1⅜ times spike length in width. But for lig porary work, or prospecting, ties are often no thicker than the spike but 4 in longer than distance between outside spikes.

Steel ties of pressed and rolled steel, have the advantages of lightness, str bility (where mine water is not acid), and of requiring less depth or headr should have flanges along all edges, extending into the roadbed to hold th Rails are fastened to them by bolts, or clips and wedges, or combinations of th shows some of the numerous designs.

Spacing of ties. Average for standard gage is 24-in centers. U ties are spaced from 16 in. on some main haulage ways with soft be even 6 ft centers in room or stope tracks. An excessive span, with li heavy wheel loads, greatly increases resistance, and the bending spikes and cut ties. For relation of tie spacing to wt of rail to be used,

edbeds, broad, close-spaced ties are used, but in some by using heavy stringers, 4 by 8 or 5 by 10 in, laid on the ties.

ies under rails and at spike holes. They are seldom used eavy traffic, or with treated ties. RAIL BRACES, to rein-

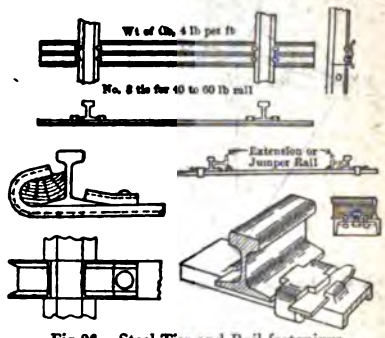

Fig 26. Steel Ties and Rail-fastenings

afford new places for spiking. For frequent relaying, as e-killed in from 1 to 5 relayings; steel ties, barring cor-y.

roken stone, gravel, tailing, waste, cinders or slag.

s & Maintenance of Way Assoc, for Class C R Rs, is a er 8-ft ties, with 3 to 1 slopes beyond the ends and filled s at 24-in centers, 1 180 cu yd per mile. Special ballast The floor is generally hard, and care of the drainage spots. Ballast will not help a swelling or heaving floor. d ties are worn by the hoofs and require more frequent ystems.

less than half the extreme width of car or loco- by the roadway clearance and sharpest curve.

e wheel-base does not greatly exceed gage, is: Gage st radius. Gages range from standard, 56.5-in, down to -in are commonest; in collieries, 36 to 42-in. ADVAN- large-capacity cars, lower total cost of rolling stock and DVANTAGES OF NARROW GAGE: lower first cost of entries curves which are made possible. GAGE ON CURVES must of wheel flanges. In R R practice, for each 2° over 8° er, to a max increase, including wear, of 1 in. For sharp eels may require all the extra play that their width of gage is obtained by starting the curve of inner rail before (Sec 17, Art 30).

rves, Switches, and Crossings (5, 7)

rcular curves fill all speed requirements of under- or transition curves are unnecessary.

e angle at the center subtended by a chord of 100 ft; dius are best designated by length of radius R. Trigo- elements of circular curves, and modes of laying them

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Layout of curves is done by transit methods, or in various ways no instrument work. They should not be left, as they often are, guided judgment of trackmen.

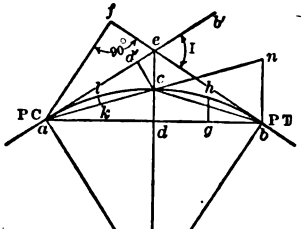


Fig 27. Laying Out Curve by Offsets

A reasonably accurate curve layout c without a transit by means of offsets, rod *ab* (Fig 27) 10 ft long, and with a offset *dc* at center *d*, of a length = in Table 4 for curve of required ra curve is started with the rod laid in p of the tangent from the P C (Sec 17, the position *ab'*. Offset *d'c* will then i first point *c* on the curve. Subsequent be indicated by point *b* when rod *ab* is the curve. When the P C and P T sight of each other, the middle and q nate method is most convenient. W ings are driven by directions from th office, blueprints should be furnished timbering and track as indicated in with a minimum of instrument work. Art 21, Shape and cross-sec of drifts.)

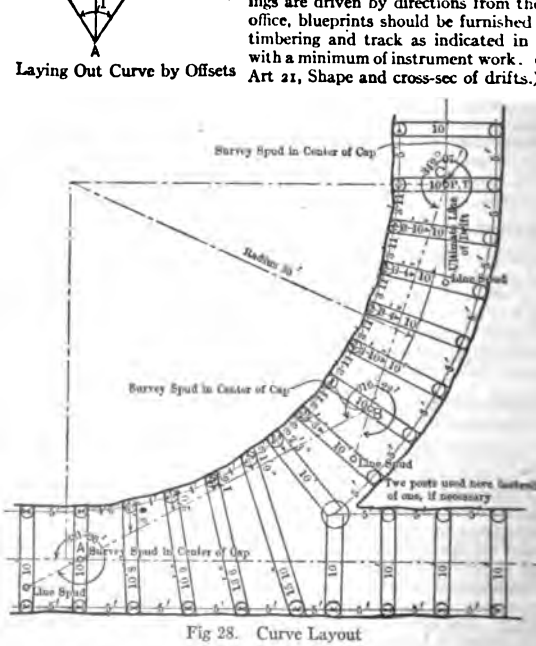


Fig 28. Curve Layout

Minimum-radius curve is limited by length of wheel base, diam of wheels clearance. Fig 30, from an empirical formula (Baldwin Locomotive Wks safe values for curves of same gage as on straight track. With liberal spread curves, 10 to 25% may be added to wheel-base dimensions.

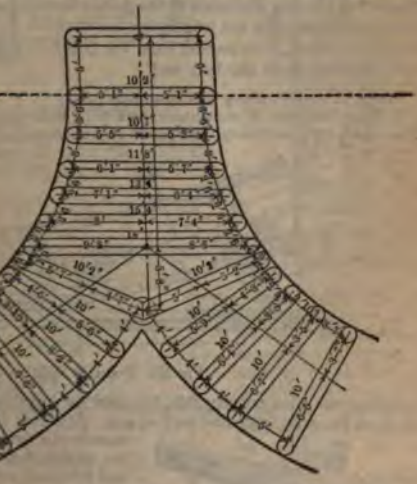
Bending rails. The common rail bender, the "Jim Crow" (Fig made in 4 sizes for different wt of rail, the span between claws being

Roller bender is a convenient modification, and hydraulic or pneumatic be 31, *b*) are made for heavy rails. Rails are bent to fit curves of given radius by

Ordinates *M*, in Inches, of 10-ft Chords

orrection, + for inner, − for outer rail, in*				
18	24	30	42	56½
+4.8 −3.3	.....	.....	.....	....
+2.9 −2.2	.....	.....	.....	....
+2.0 −1.6	.....	.....	.....	....
+1.5 −1.3	.....	.....	.....	....
+1.0 −0.9	+1.2 −1.0	.....	.....	....
+0.6 −0.5	+0.8 −0.7	.....	.....	....
±0.3	±0.4	.....	.....	....
0.2	0.3	+0.4 −3	.....	....
0.1	0.2	±0.2	.....	....
0.1	0.1	0.2	±0.2	....
0.1	0.1	0.2	0.2	....
.....	0.1	0.1	0.1	....
.....	.....	0.1	0.1	....
.....	.....	.....	0.1	....
.....	.....	.....	.....	±0.2
.....	.....	.....	.....	0.1

of 10-ft rail chords, not for chords taken radially from



Layout for Double Turnout

e above); or, for short radii, by laying out the curves of oor, the value of *M* for same length of rail being greater ent in the shop, the max length of a curved rail is limited Difference in length of inner and outer rails, D = gage × s,

rail, to balance centrifugal force, is $e = dV^2 \div gR$, rail, in; d = distance between centers of rails, in;

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V = velocity, ft per sec; g = acceleration of gravity, ft per sec per sec = radius of curve, ft.

On grades, these elevations should be reduced. With rope haulage, the cr rope on curves modifies conditions, and may even require that the inner rai (Art 17). On self-acting planes, tr can not satisfy requirements for t directions, and level ties, guard r speed are the compromise.

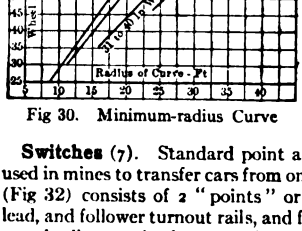
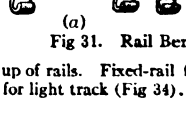


Fig 30. Minimum-radius Curve

Fitting track for temporary cutting rails. For headings or track rail is laid on its side along inner side rail, with loose rail head against web The wheel flanges then run on web which need not be fastened or match To put in a curve without cuttin spread end rails and fasten right an switch points at required P C insic

Switches (7). Standard point and stub switches and many other used in mines to transfer cars from one track to another. The point or (Fig 32) consists of 2 "points" or latches, lead, and follower turnout rails, and frog with guard rails opposite frog. On blunt switches the toes of the frog and heels of the point rails meet, otherwise filler rail lengths are inserted. Facing a switch, it is passed from the point end; trailing, from the frog end.


(a)
Fig 31. Rail Ben

Frogs. The rail crossing may be a single casting (Fig 33), preferably of manganese steel, or be built up of rails. Fixed-rail filled and bolted, for heavy track, or riveted on plate for light track (Fig 34).

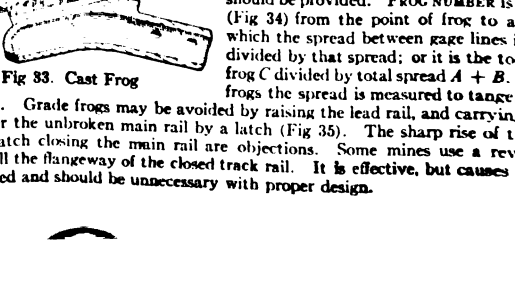


Fig 32. Point or Split Switch

frogs have one wing rail movable, but held by a spring against the frog p wheels on the main track pass over no gap. Standard frogs have straight r narrow-gage blunt turnouts, curved f ter, though they are not interchangea and left-hand turnouts, and a turnou should be provided. FROG NUMBER is (Fig 34) from the point of frog to a which the spread between gage lines divided by that spread; or it is the to frog C divided by total spread $A + B$.

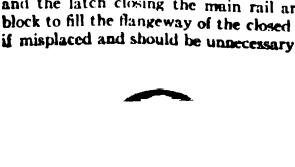
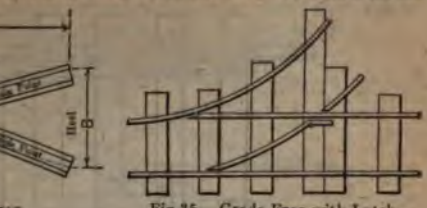


Fig 33. Cast Frog

frogs the spread is measured to tange frog point. Grade frogs may be avoided by raising the lead rail, and carrying flange over the unbroken main rail by a latch (Fig 35). The sharp rise of t and the latch closing the main rail are objections. Some mines use a rev block to fill the flangeway of the closed track rail. It is effective, but causes if misplaced and should be unnecessary with proper design.

2, let F be frog angle, N frog number, G track gage in ft, curve in ft, and L switch lead in ft. The lead is the dis-



rog Fig 35. Grade Frog with Latch

switch point to frog point, and must be known to locate switch with curved point and frog rails, or a circular int of frog:

$$= R \div L = \sqrt{R \div 2G}$$

$$\frac{G}{\sqrt{R^2 \div L^2}} = \frac{1}{2N} \text{ (approx)}$$

$$+ 0.5\,G)\ sin\ F = \sqrt{2GR}$$

$$N^2 = (L \div sin\ F) - 0.5\,G$$

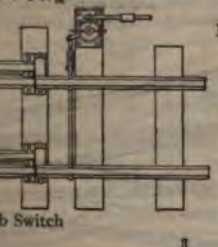
point of switch to point + filler $= \pi R\ (F \div 180)$. ls, values of L and R are

$$\tfrac{1}{2}G$$

$$- sin\ \alpha) + w\ cos\ F + p$$

w = length of wing rail, th of point rail. For a (Fig 36), let F_m be the

$$\frac{G}{(R + \frac{1}{2}G)} = \tfrac{1}{2}\ vers\ F_1 = \frac{}{\div 2N_m}$$



b Switch

Fig 36. Double Turnout

Stub switch (Fig 37) is simpler than the point switch, and cheaper if both are made at the mine; if supplied by manufacturers, the stub saves little. If the switch has no locking lever, derailments may be caused by the rear

Fig 39. Stub Switch without Frog

s. This danger is reduced by making the turnout stub a distance equal to the car wheel base. Special tie plates

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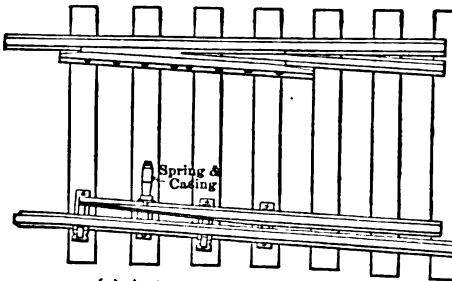
or chairs (Fig 38) should be used for stubs. A modified form of stub (Fi
the frog, and is good even for motor haulage, if there is enough room for



(a) Automatic Switch-point with Mate

(b) Spring Split-switch and Ground Throw with Slotted Connecting Rod

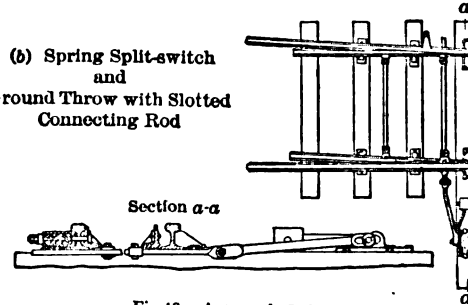


Fig 40. Automatic Switches

Automatic switches have their latches or points normally held by a spri
track; facing trains must follow that track, but trailing trains may pass fro

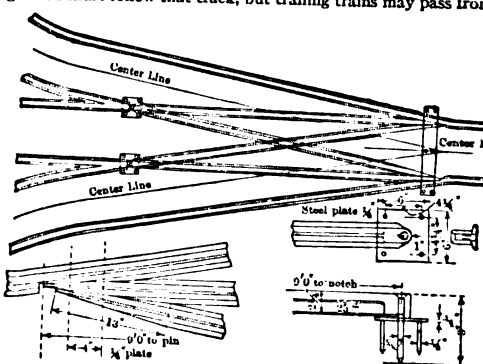
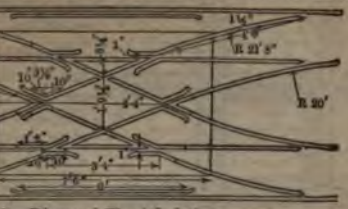


Fig 41. Three-way Finger Switch

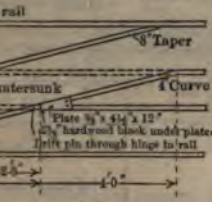
the wheel flanges forcing the point over against the spring when coming from
These switches may have a lever to engage a projection on motor or car for

n have a spring connection to the throwing lever to reduce especially serviceable for by-passing at tipples, shaft sta- a simple automatic switch; *b*, combined with lever throw.



2. Diamond Fixed Switches

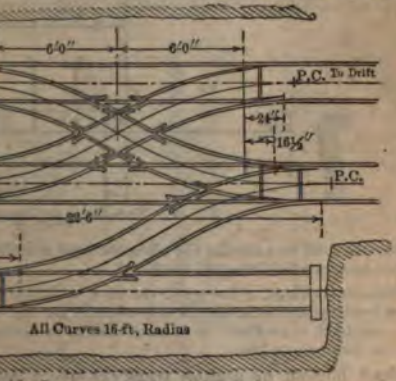
a modified point switch, omits one point rail and the frog. of a curve, but it is a cheap and good switch for light, s its application to a 3-way switch; Fig 43 to a crossover.



er Finger Switch

Fig 44. Diamond Switch

le only for hand tramming or animal haulage. All rails learances at the points, and cars are crowded toward the



uble Crossover or Diamond Switch

main rail to take or pass the switch. Lowering the outer s to protect main-line traffic.

mond switch (5) is the standard for tipples and how cases where distance between tracks is greater ere the distance is less.

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Track crossing may be an assemblage of frogs, or a built-up riveted plate
crude crossing which avoids cutting for flange-ways is made by running 1 track
the other by the depth of the rai
ing it across by one long pair or t w
of latches. One track is thus alw

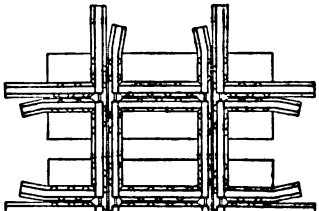


Fig 46. Riveted Plate Crossing

Turnsheets (tarantulas),
heavier plate, fastened to sills
sunk wood screws or drift bo
simplest switching device;
shaft stations, turnouts and
when the gross wt per car do
ally exceed 3 000 lb.

Sheets should be finished on ed
ported by planking or concrete ur
are closely spaced. Flared trea
rounded guards are riveted to pla
wheels onto tracks. Fig 47 shows
shaft station; Fig 45, a track-laic

Turntables are for cars or motors too heavy to handle on a turnsheet, and
is not room enough for a track switch. Construction: a pair of rails on a swiv

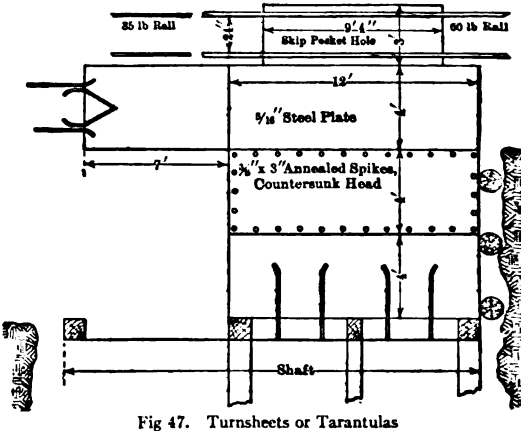


Fig 47. Turnsheets or Tarantulas

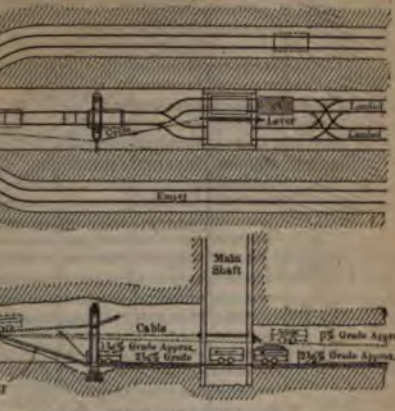
of plank or steel, or a circular C-I plate, with ball or roller bearings, on a base
nished by makers of industrial track.

Transfer carriage, a low-truck car on a sub-grade cross-track, carrying
corresponding gaps in the main or side tracks by moving the carriage, thus trans
from one track to another. It is much used in industrial plants, and sometim
tops for loading into bins.

Shaft bottom or station track (5) layout must provide for rapid
and minimum interference of empty with loaded cars. Standard coll
bottoms have a wide through entry across wall plates, with double
on both sides and grades in favor of traffic, so that empties will
switch when pushed off the cage by loaded cars.

A by-pass track returns empties, when production from both sides is not be
through station with a blind end should have kick-back and spring switch in

(Fig 48), to raise empties to top of return grade, makes
Fig 49 is an end-on shaft station with 8 spring switches



Shaft Bottom Layout, Pa and Ill

l (c), making switching of empties automatic to track *A*

ide deposits are planned with parallel loading cross-
rved into a main return drift. Hence, the locomo-
ive usually travels in but one direction, hauling a
rain of empties from the shaft, dropping the rear
ars singly at loading chutes while picking up loaded

Fig 50. Locomotive Haulage Level, Ely, Minn

m around the loop and back through double-track
50, 51).

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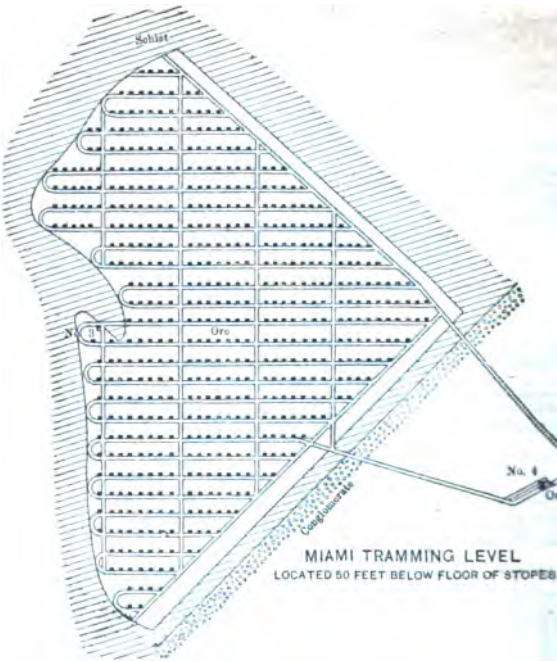

Fig 51. Locomotive Haulage Level, Miami Copper Mine, Ariz

7. Mine Track Costs

Arizona. 30-lb rail, 24-in gage, ties 18 in centers; length 7 000 ft, in medium h

Labor digging bed and laying track (hard ground)	$
Ties, 4 by 8 in by 3.5 ft, $27.50 per M bd ft, fob mine, including 10% waste.	
Rails, 30 lb, @ $37 per ton, plus haulage to mine	
Spikes, fish-plates and bolts, $0.075; shop labor, etc, 0.015	
Total cost per ft	$

Turnouts and points, fob factory, $20 per set.

Idaho. 35-lb rail, 18-in gage, ties 2.5 ft centers:

Labor bringing material from surface and laying 1 mile of track	$ 591.00
2 112 ties, 4 by 8 in by 3 ft, 11.2¢ ea	236.54
55 tons rails, @ $41 08 (July, 1913)	2 259.40
360 splice bars (4 356 lb), @ $2.70 per 100 lb	117.61
1 440 bolts (625 lb), @ $2.60 per 100 lb	16.25
10 560 spikes, 0.5 by 5 in, 4 140 lb @ $2.75 per 100 lb	113.85
Total cost 1 mile single track	$3 334.65 = $0

Mesabi iron range. Main haulage track, 40-lb rail, angle-bar joints, timber ties faced to 6 in thick, 2 to 3 ft centers depending on the ground, cost per 100 ft. Sub-level drift, 12-lb track, fish-plate joints, 3-ft ties of half-round l 3 in thick, 2 to 3 ft centers, $18 to $22 per 100 ft.

and steel (F. E. Brackett):

per yd	Steel rails	per yd
$0.084	20 lb rails @ 1.5¢, 33% deprec...	$0.200
0.112	2 ties @ 15¢.....	0.300
0.120	Spikes, bolts, and ½ plate.....	0.101
0.013	Labor, 0.135; misc, 0.016	0.151
$0.329		$0.752
0.11	Total per ft.....	0.251

udes variable quantities of brushing and maintenance, nd misleading. ANTHRACITE MINE TRACKS (4). M. S. mile with 16-lb rails, $1 906; with 24-lb rails, $2 605;

ck and Car Resistances

prise rolling resistance between rails and wheels, ertia or resistances of starting and acceleration, urves, and atmospheric friction. COEFFICIENT of y measured in terms of lb pull per ton gross load.

not well determined. Resistance does not increase depends greatly on condition of track and roadbed; per- ate impact and oscillating resistances. JOURNAL FRIC- with heavy wheel loads; (*b*) with minimum diam axles r speeds from 10 to 20 miles per hr for standard R R, mine equipment; (*d*) with perfect lubrication; (*e*) with -shaft tests by Thomas and Maurer (Amer Soc Mech 2 to 3 times the power of ball bearings, and babbitt bear- ISTANCE is usually 1.5 to 3 times the running resistance; , or in cold weather, it may be 5 times as much. This in train work by first backing and then picking up the spring couplers. INERTIA (resistance to acceleration) $(V_2^2 - V_1^2) \div S$, or $P = 95.6\ (V_2 - V_1) \div t$; where increase veloc from V_1 to V_2 miles per hr, in S ft or t = 0. A good empirical formula (T. Robson) allowing = 107 V; where V = increase per sec of speed in miles to 0.5.

ine of grade. For grades under 10%, there is no grade resistance at 20 lb per ton of moving wt for % the error is + 1 lb.

the grade at which the resistance of an ascending empty g loaded car. Drifts and adits should be driven at this nsiderations prevent. Exceptions: In hand tramming to keep the grade 0.1 or 0.2% above that of equal trac- nove easily, and the trammer can give due attention to otor haulage, this grade may also prevent overheating ection. Formula for grade of equal traction is, $\sin g =$ here g = angle of grade in favor of load; M = net wt of s; f = coeff of tractive resistance for the loaded car on car. Table 5 shows that the difference between f and f' formula for grade of equal traction, assuming $f = f'$, is 2 *c*). Practical determinations of f and f' are scanty.

= angle at which car will continue at rest or in uniform ng friction of car = % grade of repose, and varies from loaded cars on straight heavy rails to over 4% for plain and higher on curves. F. E. Brackett reports tests of a , wheel base 24 in, gage 42 in, wt loaded 6 400 lb. Grade by 4-in wood rails, A_f = 4.23 to 4.35%; on wood rails, .85 to 2.03%; on steel T-rails, A_f = 1.62%.

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Angle of inertia, starting car or train from rest, $= A_f + \sin^{-1}$ (107 V = accel in miles per hr per sec and W = gross moving wt, tons. AN FRICTION = angle at which cars with brakes set or wheels spragged will co On grease covered rails it may be as low as 6 to 8% for loaded cars, while steel-tired wheels may hold above 14°. It is usually unsafe to trust brakes of over 6° on steel rails; on clean, dry, wooden rails, the coeff of friction is

Curve resistance for standard-gage track varies from 0.5 to 1.72 lb; ge 0.8 lb per ton per deg of curve. On short-radius curves used in narrow-g factor is variable and with worn wheel treads is considerably higher. (*Trans* A I M E, Vol 18, p 514) showed 21 lb per ton friction due to curve on 85-ft radius curve, and 7 lb per ton for a 20-car train moving at 1 oo curve of 350-ft radius.

Air resistance varies approx with end area and length of train, and t veloc, and is independent of train tonnage. It is usually unimportant in but because of the piston effect of cars in small-section gangways, it shou equivalent to a wind veloc double that of the train. With a 20-car train hr, the total resistance may be 50 lb.

Coefficient of traction (ratio of draw-bar pull to wt of moving uniform motion on a straight level track $= \tan A_f$), may be obtained

Table 5. Tractive Resistance of Mine Cars on Straight Level

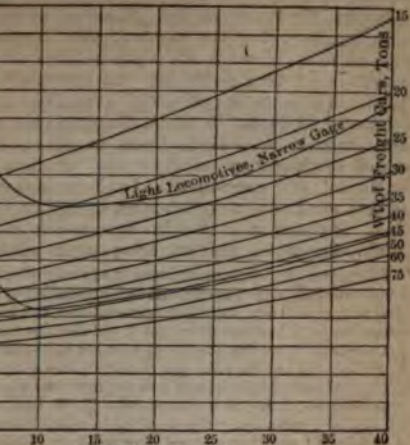
Type of car	Wt, ton	Bearings	Speed, ft per min	Resista lb per
Coal mine, empty*	1.12	plain open	starting	89.20
" " " *	1.07	" "	1 000	57.80 } si
" " loaded*	3.44	" "	"	66.40 } ca
" " empty*	1.07	" "	"	44 00 }
" " loaded*	3.44	" "	"	39.00 } tr
" " empty*	1.21	self-oiled	50	33.20
" " " *	"	" "	starting	62.00
" " loaded*	4.08	" "	50	32.60
" " " *	4.56	" "	starting	49.00
" " empty*	1.21	" "	1 000	30.40
" " loaded*	4.08	" "	1 000	38.60
" " " †	3.57	" "	492	24 30 tr
" " " †	"	flexible roller	526	12 79
" " " ‡	1.91	self-oiled	.....	27.20
" " " ‡	1.94	flexible roller	.....	25.34
Coal mine, loaded**	7.00	babbitted	{ starting 380	129 0 trai 16 75
" " " **	"	brass	{ starting 450	81.10
" " " **	"	flexible roller	{ starting 480	55.0
" " " ††	3.57	plain	490	24.3
" " " ††	"	flexible roller	530	12.8

* Norris, *Trans* A I M E, Vol 18, p 514. † Liebermann, March, *Mines & Min*, Vol 25, p 49. ** Edwin Ludlow. Tests by Hyatt Roll at Coaldale Colliery. †† Edwin Ludlow. Tests at Greensburg Colliery.

Fig 52 gives resistance per ton of R R freight cars of different at various speeds; also starting resistance of light, narrow-gage loc passenger cars. Starting resistance curves for freight cars should

Roller bearings. Few verified tests are as yet obtainable. Rust's test 9-car, self-oiling, and 5-car, new roller-bearing trains, gave 7.3% saving. tests were on Hyatt-Hockensmith bearings and indicated 48% saving. Co tests were with new equipment, 9-car trains, on perfect surface track.

rack." Greensburg tests were for 20-car trains, under Iron, Coal & Coke Co's tests, for 3 trips of 20 cars with



Tractive Resistance of R R Equipment (Baldwin Locomotive Works)

and Tramming (8, 9)

man pushing a car can readily exert about 20 lb intervals of rest, about 50 lb; when starting, or on seful output will depend on weight and capacity of character of car bearings, track, and gradients, to stances will not seriously exceed the above figures.

1 ft cars, on 18-in gage aver mine track, varies r 100 ft; or approx one useful ton-mile per hr, me. The duty varies directly as the unit load ely as the time per cycle (Art 1).

earings, capacity 1 800 lb, having 1.5% loaded and 2% factor 3, grade in favor of load 0.625%; then the start- unning resistance 23.6 lb for the loaded car, as against ning empty. When traveling at 2.25 miles per hr, the is at the rate of 8.1 useful ton-miles per 8 hr. Using a traight, clean, perfect track with 0.3125% grade, the t 0.75% loaded and 1% empty. Hence a gross wt of ould be handled with the same exertion, and the tram- 6.3 useful ton-miles per 8 hr. Such figures are prac- Certain southwestern copper mines are now (1915) hand tramming.

ts are frequently included in tramming charges, thods, they should be kept separate. For LOAD- ng to a shaft station, under best conditions, the gh the shift will be not less than 4 min per round 800-lb load 1 man can tram 32 cars 1 000 ft, or 90

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cars 100 ft in 7.5 hr actual work, and his output will be 5.5 and miles respectively. With chutes stopped by bridging or jammed blocked by spill, derailment, ore shortage, or repair work, the o reduced to the vanishing point. LOADING BY SHOVELING. When mittently, good shovelers working under contract will handle up hr from steel plates into low, open-end cars, or 2 tons from a rc buckets or revolving dump cars. Aver performance when trammir ft is from 10 to 20 tons per shift. AVER DAILY DUTY of contract s Missouri lead districts is (H. A. Guess) 18 tons, while Higgins esti Mucking costs for all kinds of work usually run much higher on d or with irregular supply of ore or cars. On basis of loading at rat hr, the duty of a man with the 1 800-lb car load would be: loading 19 cars 100 ft = 0.32 ore-ton-miles, or 14 cars 1 000 ft = 2.39 ore shift. If trammers or shovelers have to help miners place timber or shoveling plates, their tramming output will be proportionatel

Examples. In MICHIGAN COPPER MINES tramming is usually done on t mum-wage and bonus systems, and includes loading by shoveling and hanc floor or footwall into large, low, open-end cars, and tramming by 2 men According to J. MacNaughton (1913), the distance trammed at 11 proper 171 to 900 ft, averaging 609 ft. Output per man-shift was 12.12 to 19.48 14.76, and the ore-ton-miles per man-shift, including loading, was 0.47 t aver. In loading from chutes into the cars shown in Fig 4 (Art 3), the tramming 600 ft is 6 to 9 ore-ton-miles (F. W. Denton); when shovel-lc 10 cars or 3 ton-miles. The standard in many BUTTE MINES is 15 cars shovel-loaded and trammed not over 100 ft in drifting work, and appr loading from chutes. Cars aver 1 600 to 1 700 lb rock, or 1 700 to 2 000 lb CONSOL MINE, Ariz, on tramming drifts, the aver duty for a distance of 25 to 1-ton cars per man-shift, with bonus of 3 to 4¢ per car over 50 cars. On same mine, a gang of men loaded from chutes 150 tons per man into 5-ton ca A I M E, Bull 102). In PORTLAND MINE, Cripple Creek, Col, trammin and waste from Lee stope 330 to 530 ft to shaft or waste chute cost 15.4¢ p per ton-mile; duty was about 2.2 ton-miles per trammer shift. In the 35 365 tons ore and waste were trammed an aver of 975 ft, at cost of 1: 98¢ per ton-mile, or 3.2 ton-miles per shift. In MT ROYAL TUNNEL, Ont, back and 4 loading cars ordinarily loaded 7 cu yd per man-shift; in a rec loading muckers averaged over 12 cu yd (max 15 cu yd), nearly 2 hr bein blasting (*Trans* Can Min Inst, Vol 17, p 182) (8).

10. Animal Haulage (10, 11, 12, 13)

Duty. Mules and horses at underground work travel 3 to 15 With enough cars, and grades not over 2% against the empties, t age 7 to 12 miles on main haulage lines, and usually below 9 mile work. Many records show duties from 50 to 100 gross ton-mile and empty haul) per animal, or 25 to 66 mineral-ton-miles per sh

To justify animal haulage as against tramming, a minimum of about miles should be done on a given level or entry per shift, unless there are The economy of animal haulage increases with tonnage. Using tandem, and longer trains, increases effic of both animals and drivers, though more sidings are required. For duties below 200 mineral-ton-miles per day on 1.25%, it has been claimed that no mechanical system can compete i animal haulage using a string team of four. The exact dividing line dep conditions, and costs of labor, power, and feed.

Cost of animals. Good mules cost $150 to $300, depending c general condition. Their underground working life in the U S is

Horses and ponies used in English collieries average 7 to 10 yr. It is keep an animal in the mine after it has begun to deteriorate. It should s to $75 for surface use. An annual deprec of 20% of first cost should pro

nules varies from 18¢ in farming regions to 50¢ in
ore in isolated mining districts.

e $1 to $2 per month, interest and deprec on mule and
e, when stable attendance and full time maintenance are
le service per working day costs 60¢ to $1.25, exclusive
working 6 days per week; except for reduced use of grain
onately higher when time is lost by illness or shut-down.

dangerously overfeed if permitted, especially when
more than they will finish at a meal, as damp grain
using illness. For same reason, and because of fire
eed supply should be taken underground at one time.

condition, weight, and work of the animal. Table 6
food fuels for the important U S feed stuffs. Table 7
estibles required, per 100 lb live wt of animal when idle,
With these tables and local prices of feed, the most
calculated, just as metallurgical furnace charges are de-
bulk, about 10 to 18 lb of the daily feed should be con-
e, the proportion and quantity of roughage decreasing
th the severity of work. Most of the roughage should
he concentrates in the morning. The mineral require-
animals not in foal, except that about 0.25 lb salt should
y nights, or when the animal is to be laid off, replace
gularity in DRINKING is more important than the time.
l, and it is better to allow it to drink moderately before

e Nutrients in 100 lb of Typical Feeding Stuffs *

	Total dry matter, lb	Digestible crude protein, lb	Total digestible nutrients, lb	Nutritive ratio, 1 : —	Wt per bushel, lb
...	89.5	7.5	85.7	10.4	56
...	90.8	9.7	70.4	6.3	32
...	89.8	9.2	80.1	7.7	60
...	90.7	9.0	79.4	7.8	48
...	88.2	9.0	80.0	7.9	56
...	90.9	30.2	77.9	1.6	35
...	92.5	37.0	78.2	1.1	48
...	89.9	12.5	60.9	3.9	16
...	92.5	21.5	65.7	2.1	19
...	90.8	19.0	76.2	3.0	67
...	81.7	3.0	53.7	16.9	.....
...	81.0	2.1	46.1	21.0	.....
...	88.4	3.0	48.5	15.2	.....
...	93.5	4.0	47.9	11.0	.....
...	91.4	10.6	51.6	3.9	.....
...	87.1	7.6	50.9	5.7	.....
...	88.5	1.0	45.6	44.6	.....
ed.	26.2	1.1	17.8	15.2	.....
...	25.3	3.3	14.6	3.4	.....
...	26.3	1.1	17.7	15.1	.....

ıd Morrison, "Feeds and Feeding," 15th ed, 1915.

will respond to good treatment, and will make the
ed. A "bad" mule is often made so by abuse.

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Some animals are not intelligent or quick-footed enough, or are too nerv strung for mine work, and purchases should be made with a trial option cla boss, not the drivers, should do the feeding; he should inspect animals comi report cases of abuse, and see that harness fits well and is kept clean. Especial be taken of the feet, and cuts and sores should be given antiseptic dressing (1 ground stables are always drafty, and sweaty animals should be blanketed unt bedding, 100 lb stra or 350 lb sawdust per month per animal are reasonable

Underground stables (12) must be well ventilated; a separate direct return to the upcast is required by law in most coal mining sta should be dry, not less than 5 by 10 ft, with not more than 2 in grade for drainage, and little or no cross grade.

There should be a passageway both behind and in front of stalls for handl and refuse, and a raised platform for feed storage. Roof over stalls and should be watertight. There should be a pressure water supply; tracks sho with floor, and partitions raised above it, so that the whole floor may be hos

Table 7. Modified Wolff-Lehmann Standards for Work Horses or (Henry and Morrison)

	Daily per 100 lb live weight			N
	Dry matter, lb	Digestible crude protein, lb	Total digestible nutrients, lb	
Idle	13–18	0.8–1.0	7.0– 9.0	1 : 8.
Light work	15–22	1.1–1.4	10.0–13.1	1 : 8.
Medium work	16–24	1.4–1.7	12.8–15.6	1 : 7.
Heavy work	18–26	2.0–2.2	15.9–19.5	1 : 7.

Horses *vs* mules. Mules average smaller, require less headro heat and neglect better, and are less liable to foot lameness than hor will eat roughages which horses refuse, but are less apt to overeat, and subject to colic or founder. Horses are heavier, better built draft an mules, average more reliable, haul larger loads, and require little or no per 100 lb live wt than mules on similar work. They are more sp mules, and a nervous horse is apt to rear and injure his head ag Mules are more generally used underground than horses, but, whe ways are large enough and the duty sufficient for heavy horses, the of mules are debatable, and where there is headroom many coal ope use large horses in preference to mules.

Examples. According to the Illinois State Co-operative Reports for 19 mules costing $175 to $275, used for gathering colliery cars for locomotive worked at high speed, and the underground working life has been reduced to mules then selling at about $40. Cost of feed, shoeing and harness repair is 7 day. In the best managed mines, cost of feeding mules with corn @ 60¢ pe hay @ $15 per ton averages $10 per month, some exceeding $14 per month. 1 300-lb mules haul 75 loaded trips 700 ft on 2% grade in favor of the load. consists of 4 1 000-lb cars, with 3 500 lb coal each, or 18 000 lb gross. Gros mileage per mule is 54.67, or 34.80 coal-ton-miles, equivalent to about 10¢ p mile. At another mine, 1 mule on gathering work on 0.5% favorable grades, of 3 cars 1 000 ft, with a duty of 24.43 coal-ton-miles, total cost being 17.6¢ p mile. At another mine, with 2% favorable grade, a 3-mule spike-team hand 17 1 800-lb cars, each holding 2 100 lb coal, or 22 100 lb gross per mule, co to a tractive effort on the return trip of at least 300 lb per mule.

In Alabama-Tennessee mines (1904, C. E. Bowen) aver for 500 mules in 6 and care, 34.9¢ per mule; int on cost, 3.3¢; deprec, 7.0¢; total, 45.2¢ per d 276 days = 59.8¢ per working day. Shoeing and harness 3 to 6¢ (aver 4.3¢) $1.762 per working day. Aver duty 3.9 to 33.2 net ton-miles per mule, as fo

es per mule		Cost per net ton-mile, cents	Conditions
	Net		
	6.9	35.7	Unfavorable
	13.8	17.9	Aver
	23.0	10.7	Best
	21.4	11.5	Aver

llowing haulage costs were reported: feed, harness, and deprec, 20% per annum, 24 days per month, on $200, on investment at 6%, $0.41; total, $25.79. The haul 2.5% and against empties 5%; aver duty, 400 tons per n-mile. Total daily cost per mule on gathering work, ton.

dled in 1 year, on 0.75 to 2% grades, aver haul 1 300 ft, rs, $8 169; keep of 16 mules, $2 302; deprec, $392.50; lue of 16 mules, $181.20; aver cost per ton, 5.496¢, or

aulage is stated at 10.6¢ per ore-ton-mile.

hauled five 800-lb cars, each with 1 700 to 2 000 lb of ore, nen 860 ft up a 1% grade, making 14 trips, or 15.8 to on favorable grades 8-car trains were used, making about rs per train in each case. Costs were probably between

ule haulage with 1 800-lb cars; 2 500 lb ore, grade 5 in ver distance 1 400 ft, cost 16.5¢ per ton for labor, and or 66¢ per ore-ton-mile, exclusive of int and deprec.

omotive Haulage (14 to 21)

internal-combustion locomotives are independent; ery types require intermittent, and trolley, constant of power. For special service, trolley and storage-rail drives, are combined in a single locomotive.

ge is justified at mines with easy track grades and large amming or animal haulage, the high speed, large ton-d, may balance the larger interest and deprec charge on e haulage, especially if there are branches or many er to extend and maintain. It requires less headroom, (Art 6), than animal haulage and cannot compete with of less than about 175 net ton-miles on grades below grades averaging over 4% against or 5% with loads. y momentum, may be 1 to 3% steeper. Geared loco-athering cars.

motive is determined by its drawbar pull, speed, loaded cars. If it has enough power to make its

f Adhesion (Baldwin Locomotive Works, 14)

	Chilled C-I wheels		Steel-tired wheels	
	Without sand	With sand	Without sand	With sand
	0.10	0.15	0.10	0.15
	0.15	0.20	0.15	0.20
	0.15	0.25	0.15	0.25
	0.18	0.22	0.18	0.22
	0.20	0.25	0.25	0.31
	0.20	0.25 to 0.30	0.25	0.31 to 0.37
	.....	0.38 to 0.42	.....	0.47 to 0.52

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driving wheels slip, its max drawbar pull is limited by the adhesion wheels to the rails. COEFF OF ADHESION is the tractive force requi drivers ÷ wt on drivers. Table 8 gives approx values for mine lo Makers' tables are usually based on 20 and 25% adhesion, and 20 to 4 resistance for train on the level.

General formulas applicable to all types of adhesion locomotives: Let adhesion; C = force required to move complete train, including locomotiv drawbar pull, lb; E = fric resistance of locomotive, lb; F = fric resistance per ton; G = grade resistance, lb per ton; H = dynamometer or drawbar motive; L = wt of locomotive on drivers, tons; S = speed, miles per hr; T of train, tons. Then:

$D = T(F \pm G)$ ..... (1) $C = D + E \pm (L \times G)$ .....

$L = D \div (2\,000\,A - G)$ ..... (2) $H = D \times S \div 375$ .....

Total dynamometer hp-hr for M miles, @ aver drawbar pull D, $= DM \div 3$

From (1) and (2), $T = L(2\,000A - G) \div (F + G)$ .....

Effect of grade on locomotive capacity. Assuming a max tracti the level of 25% of wt on drivers in Eq (6), then locomotives can move on drivers the gross train tons given in Table 9. Multiply these weights on drivers for the locomotive under consideration, and for satisfacto deduct approx 25% from the total to allow for acceleration and cor

Table 9. Tons Hauled per Ton on Drivers, Besides Wt of Locomoti Drawbar Pull on Level = 25% of Wt on Drivers

Resistance of trip, lb per ton	10	15	20	25	30	40
On level track.....	50.0	33.0	25.0	20.0	16.6	12.5
Up 0.5% grade.....	24.5	19.6	16.3	14.0	12.2	9.8
" 1% "	16.0	13.7	12.0	10.7	9.6	8.0
" 2% "	9.2	8.4	7.6	7.1	6.6	5 7
" 3% "	6.3	5.9	5.5	5.2	4.9	4.4
" 4% "	4 6	4.4	4.2	4.0	3 8	3.5
" 5% "	3 6	3.4	3.3	3.2	3.1	2.8
" 6% "	2.9	2.8	2.7	2.6	2.5	2 3

For greatest effic in power haulage, the grade should be that of equal tract The rapid loss in effic with increasing grades makes the economic grade lir motives usually below 3%. There is also liability of runaway accidents on ste

Rack-rail locomotives have much better effic on grades than others, becau bar pull with ordinary drivers can not exceed the adhesion, and while the pow a locomotive, $D \times S$, is not constant for varying speeds, the rack-rail type, l to the track, can, by decreasing S, increase D above the adhesion limit, or for a of D can decrease locomotive wt below the adhesion requirement. Rack rail geous on long moderate grades, or in rolling seams; it is the only practical ty over about 6%, and can be safely but not economically used up to 15 o Schneider, *Trans* A S Mech Eng, Vol 14, 1893.)

12. Steam Locomotives (14)

Steam locomotives are usually impossible underground because of their e and vapor, but may be used in adit mines having large entries and abundant Oil burners are preferred to coal burners, because of better control of combust retical MAX TRACTIVE FORCE of a steam locomotive is $C = (d^2 \times l \times p \times B) \div$ where, d = diam cyls, in; l = stroke, in; B = boiler press, lb; W = diam dri in; p = 0.85 for simple, 0.6 for cross-compound, or 1.2 for Mallet compou d being for high-press cyls. Substitute for C its value from Eq (3) in E termine sizes of cyls and drivers. MACHINE FRICTION E = loss between cyl bar; it increases with the ihp, but its ratio decreases with increasing load. consolidation locomotive show E = 10% of ihp with heavy load and 22% wit 82, Ill Exp Sta).

umption of simple R R locomotives aver approx 4.5 uel oil and 27 to 32 lb water per hp-hr. Superheaters very 4 10° superheat. Compounding saves 10 to 20% ood condition, but advantage may be more than bal- he overall consumption of locomotives, including stops approx 50 to 100% more than above figures.

eports of Pa (1912) showed 45 steam locomotives under- bituminous mines. Many were oil burning and some were oil.

6-wheel switch locomotives, with 62 tons on drivers, were wing are aver results for 3 yr: round trip, 4.5 miles; grade 4 16-cu yd 49 000-lb cars = 90 tons material per trip; 18 12 563 gross or 3 645 net ton-miles. Daily cost of operation er, 48 tons; oil, 5.32 pints; grease, 4 oz; waste, 1 lb; wages man), $9.95 per shift; locomotive maintenance, $50 to $70 for tires, boiler, machinery, and painting.

Gasolene Locomotives (15, 16)

have the flexibility of steam locomotives in radius of k are geared for speeds of 3 to 12 miles per hr. They no fuel when idle, and are less dangerous than steam The exhaust gases and cost of gasolene are the limiting other types.

od (16), the max volume of CO produced when feeding gaso- s 5.75% of the piston displacement, and, under the worst % of the exhaust gases are CO as against 4 to 6% normally. 000 volumes air for continued breathing and should never re concludes that gasolene locomotives require from 7 840 ; by 5-in motors, to 35 140 cu ft per min for 6-cyl, 8 by 7-in % dilution. Stopping in dead air, or traveling with the air y be dangerous. The exhaust should be washed and cooled deodorizer before discharging, which does not, however, se of motor should be limited by the available air current. gainst leakage at the carburetor and when charging tanks.

icensed Automobile Manfrs for gasolene motors is: e d = diam cyl, in; n = number of cyls, with piston per min. This formula is conservative for automobile ouretor unfortunately used in many mine locomotives than good automobile motors. The uncertainty of

ble 10. Gasolene Locomotives *

s, in base	Speed, miles per hr	Overall dimen, in Length	Width	Height	Price, factory
....	.....	.....	.. ..	.....	$1 250 (1914)
36	4 and 8	113	44	62	.....
36	4 " 8	120	48	40	.....
42	3.5 " 7	144	50	46	3 500 (1916)
....	4 " 8	132	48	46	.....
48	4 " 8	146	54	72	3 500 (1915)
50	4.3 " 8.5	180	60	48	5 000 (1916)
48	5 " 10	150	56	50	.....
51	5 " 10	156	56	52	4 500 (1915)
54	6 " 12	171	60	54	.....

ee, Vulcan and Whitcomb, 4-cyl, 4-cycle engines.

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the products of combustion of lower distillates or fuel oils prohibits tl underground locomotives. Makers claim fuel consumption to be o. lene per hp-hr. Hood's tests (Bureau of Mines, *Bull* 74) show 0.7 gasolene per brake hp-hr for full load and full speed, 1.2 to 2.2 at hal half speed, and 3.5 to over 6.0 lb at one-eighth load and half speed.

Examples. Righter Coal & Coke Co, Lost Creek, W Va (*Coal Age*, Vol 5, ton Vulcan locomotive; 25-lb rails; undulating grades of 1 to 2.5%; sharp cu 12 cars, each 0.5 ton empty, 3 tons loaded; haul, 1 800 ft one way; output, 23 t tons per 9-hr day. Operating cost: 1 motorman, $2.50; coupler, $2.00; 10 g $1.60; oil, $0.30; total, $6.40 per day = 0.93¢ per ton or 2.72¢ per coal-ton m add repairs, deprec, and int.

Herbert mine, Pa (*Coal Age*, Vol 8, p 415). 5-ton Milwaukee locomotive, 400 cars, wt 2 000 lb, coal 4 000 lb; daily output = 2 × car capacity. Opera motor per day: 2 men, $5.35; 11 gal gasolene, $1.32; oil and waste, $0.11; tot 500 tons coal hauled 2 700 to 3 000 ft, showing saving of $8.64 per day over ho

Shinnston, W Va. 6-ton Whitcomb locomotives; 25-lb rails; 36-in gage; 17 cars each, gross 52.6 tons; haul 3 100 ft. Cost per month of 10-hr days: g gal; oil and waste, $10.99. Aver cost per ton for year 1914: labor, 1.44¢; gas oil, 0.08¢; repairs, 0.40¢; total, 2.58¢ per ton = 7.92¢ per coal-ton-mile.

Shade Coal Co, Pa (*Coal Age*, Vol 5, p 647). 7-ton, 4 4-in cyls, 4-cycle locomotive, 36-in gage; 30-lb rails; haul, 3 400 ft one way; max grade, 6% loads, 5% against empties, aver 2% and 2.2% respectively; daily output, 375 t motorman and trip tender @ $2.75, $5.50; 16 gal gasolene @ 15.5¢, $2.48; and waste, $0.41; repairs @ $35.81 per month, $1.50; int and deprec @ 16% $1.94; total cost per day, $11.83; cost per ton, 3.15¢, or per ton-mile, 3.79¢. U conditions, but hauling only 145 tons per day, so that equipment was idle at total working time, cost was 11.73¢ per ton-mile.

Eagle Mining Co, Canton, Ill. 5-ton Milwaukee locomotive, 36-in gage, ha cars; loaded wt, 3 500 lb each; train is split at a 3% grade against loads near 500 tons per 8-hr day; round trip, 6 000 ft or 284 coal-ton-miles. Operati cluding runner, trip rider, gasolene, oil, and grease = $7.25 per day = 2.6¢ p

Shiloh, Ill. 6-ton Whitcomb locomotives; 42-in gage; 20-lb rails; 0.5% gr of load. Duty per motor per 8-hr day, 36 trips of 16 cars = 42.4 tons gross, 8 haul, 2 436 ft. Cost per motor per mo of 110 working hr: gasolene, 194 gal, waste, and repairs, $11.46; 2 men, $85.52; total, $122.03 = 1.5¢ per coal-ton-m

Cleveland Co, Wis, zinc mine. 4-ton Whitcomb locomotive; tonnage (191 total distance traveled, 6 970 miles; cost, labor, supplies, and maintenance,

Trojan Co, S D. 7-ton Milwaukee locomotive; 18-in gage, wet track; 4% gr of loads, bad curves limiting return to 8-car trips. Cost for 25 days: engine gasolene, 382 gal; engine oil, 16 gal; lubricating oil, 10 gal; total, $76.97. 612; ton-miles, 3 516; cost per ton-mile, 2.2¢ not including int or repairs.

Huerfano Co, Colo. 6-ton locomotive; 15-car trips, of 41.25 tons gross; 3 max grade, 5.5%, using engine plane to pull locomotive and train. Cost for 1 per coal-ton-mile.

14. Compressed-air Locomotives (17, 18)

Characteristics. They carry their own power supply, are flexible in within the radius of travel from charging stations for which they are require no fire, do not spark nor emit noxious exhaust, but aid venti are the safest type for gaseous, dry timbered, or poorly ventilated inflammable buildings, or magazines. (For data on air pressure, tank reheating of air and tractive force, see Sec 15, Art 29.) Compound motive requires approx 30% less power than the simple cyl, and is dis H. K. Porter Co report for their compound locomotive that brake 2 000 ft-lb work per cu ft of free air compressed, when running at 15 capacity, 2 500 at 30%, and 2 800 at 37.5% or over.

Operating requirements: (*a*) a multi-stage compressor to furnish air at t of consumption and at approx 200 lb greater press than the locomotive r

ransmission line large enough to act as storage, near charg- harging stations so located that the locomotive can always charge is exhausted; (*d*) a locomotive heavy and powerful el drawbar pull and with sufficient tank capacity and press ad to the point most distant from the station. Because of line compressor can not be used, except that it would be compressor to take its supply from a cool receiver at the he atmos.

ge (see Sec 15, Art 29).

5, Art 29, Fig 54).

is determined as in Art 1, and size of cyl by Eq (7), Art 12, press for B and 0.95 to 0.98 for p, because the air at full diation as steam does. REQUIRED TANK CAPACITY of loco- al work done per max load and longest round trip. RADIUS y adding an air-tank tender, together with corresponding orage.

. **Compressed-air Haulage Plants**

Locomotives			Total cost	Round trip, ft	Train, tons		Cost, ¢ per net	
No cyl	Size cyl, in	Wt, tons			gross	net	ton	ton-mile
2	7×14	8	$13 673	5 100	62.4	42	...	1.875
2	...	...	13 701	{3 200 / 5 200}	...	...	3.6	...
1	9×14	...	.....	9 000	87.7	...	...	1.25*
2	7×14	8	15 156	{8 000 / 4 200}	147.0	...	{... / ...}	{1.39 / 3.05}
2	...	...	15 800	.....	...	...	...	2.17

Vol 18, p 538. (*b*) Baldwin Locomotive Wks. (*c*) Peele, orris, *Trans* A I M E, Vol 30, p 566. (*e*) Two-stage loco- d deprec, 2.17¢ per ton-mile, as against 2.56¢ for single- (estimates by H. K. Porter Co).

ery Locomotives (19, 20) (See also Sec 16, Art 11)

are flexible prime movers, cheaper to install than imple to operate and safer than any type except com- hort hauls in temporary workings, as in gathering, or here trolley lines are not warranted or low-roofed wet erous. Grades must be easy; max speed per hr, 5 n on gathering work, while cable-reel locomotives is greater than $D + A$, because of the necessary bat- useful load capacity on grades than other types, and battery output rate.

on : (*a*) max rate of current discharge required; (*b*) ate on 1 charge.

be expressed: $H_e = (D \times S \times 746) \div (375 \times p)$, where H_e locomotive in watts, and p = locomotive effic = 0.6 (approx. ormal voltage and current discharge capacity = $H_e \div K$, e works continuously at max capacity, 2 for approx 25% , and 4 or 5 when max is only momentary. For (*b*) the trips be analyzed, the ft-lb for every move calculated and the total -hr required; to this add 30 to 40% to allow for excessive tly set, contingencies and reduced effic of battery when dis-

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charging above its normal rating. Batteries must be out of service at least
for recharging and maintenance, but with duplicate batteries locomotives ca
shifts and charging can be done slowly. DIRECT CURRENT at 250 volts or l
charging, and if not available must be supplied by a motor-generator set,
is determined by required rate of charging. (For batteries and motors, se

Examples. In COPPER QUEEN tests (*Trans* A I M E, Vol 48, p 295)
locomotive required 1.6 kw-hr at power station per useful ton-mile, or 1.
as against 1.1 and 0.875 for trolley locomotive. BIG FIVE TUNNEL, Co
locomotives and tenders; full-load speed, 4.5 miles per hr; train, 20 to
tons gross; 20 and 12-lb rails; grade, 0.25%. Storage battery was used
trolley on surface. Operating costs for 6 months: labor, $1 098; materi
$33; power at 1.2 to 5.5¢ per kw-hr, $306; total, $1 551; duty, 18 914
to 12.2¢; aver, 8.2¢ per ton-mile. STERLING COAL CO, O. 11 2.5-ton
locomotives are used for gathering cars from 12 to 15 rooms, each handling
4 trolley locomotives are detailed to serve 11 entries, requiring 19 men.
tives (Sec 16, Art 11) could do the work, saving 4 locomotives (approx
requiring 22 men to operate. GRANT MINE, Ind. A 5-ton locomotive,
cells, averages 75 volts @ 60 amp normal discharge; train of 16 cars
carrying 3 600 lb coal, used 90 amp on level, 160 up 1% grade and 220 on
Some CŒUR-D'ALENE mines, and others with conditions favorable to
have adopted storage-battery haulage, because of casualities from trolley

16. Electric Trolley Locomotives (21) (See also Sec 16

Characteristics. These are the prevalent type in mines.
pact, simple to operate, have high effic and usually the lowest po
types. Disadvantages: cost and danger of the trolley circuit; dang
ing at commutator or trolley wheel; necessity for wired and bonc

About 80% of all mining trolleys use 250-volt d c, which is slightly m
design of motors and control and less dangerous than same nominal a-c
500-volt circuits are used, and there is at least 1 3-phase 2-wire and gro
in U S. MOTORS are of approx 10 hp per ton wt of locomotive. MAIN
MOTIVES weigh 3 to 30 tons. GATHERING LOCOMOTIVES are 2 to 8 ton (a

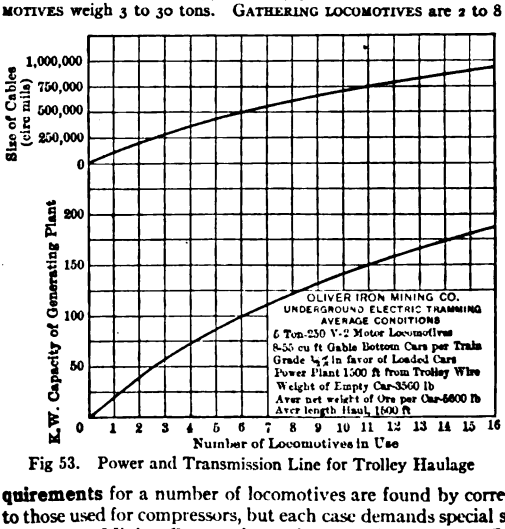
Size of Cables (circ mils)
1,000,000
750,000
500,000
250,000
0
K.W. Capacity of Generating Plant
200
150
100
50
0
1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16
Number of Locomotives in Use
OLIVER IRON MINING CO.
UNDERGROUND ELECTRIC TRAMMING
AVERAGE CONDITIONS
5 Ton-250 V-2 Motor Locomotives
8-55 cu ft Gable Bottom Cars per Train
Grade 1½% in favor of Loaded Cars
Power Plant 1500 ft from Trolley Wire
Weight of Empty Car-3560 lb
Aver net weight of Ore per Car-5600 lb
Aver length Haul, 1500 ft

Fig 53. Power and Transmission Line for Trolley Haulage

quirements for a number of locomotives are found by correction
to those used for compressors, but each case demands special study.
Oliver Iron Mining Co practice under conditions stated.

4) f o b factory for 2-motor, single end control: 2.5-5-ton, 16.6¢; 15-ton, 9.4¢; 25-ton, 8.3¢; for separ- $400 to $500.

locomotives on main haulage in 18 ILLINOIS COAL MINES iven in Table 12. Most cars have wood body, with plain of haul, grades, curves, coal or car supply, and in ratio to 46%, account for varying results. BUNKER HILL & haulage (1911). Distance, 9 135 to 13 555 ft one way, ths hauled 227 070 tons of ore; operating cost, $15 997, and supplies = 7.04¢ per ton or 3.03¢ per ore-ton-mile. abor, and supplies, $1 070; of 102 cars, $1 735. Main- ering, $720. Later costs, length of haul not given, but able 13).

y-motor Main Haulage (Illinois Mines)

n-mileage		Locomotive wt, tons	Miles traveled per shift	Ton-mileage	
	Gross (exclud- ing loco)			Coal	Gross (exclud- ing loco)
	1 556	10	22.7	568	977
	1 593	10	32.5	683	1 366
	575	12	36.0	2 203	4 095
	1 337	12	15.9	444	730
	1 598	12	23.0	667	1 127
	888	12.5	26.5	716	1 432
	875	13	20.3	908	1 560
	1 167	15	29.8	848	1 509
	675	15	34.1	1 107	1 823
	920	Aver 10	28.3	743	1 358

Account, Bunker Hill & Sullivan Mine

1913 436 060 tons		1914 440 819 tons		1915 (5 mo) 186 860 tons	
Total	Per ton	Total	Per ton	Total	Per ton
$9 093	$0.021	$9 233	$0.021	$3 121	$0.017
2 427	0.006	2 800	0.006	1 129	0.006
1 721	0.004	1 747	0.004	645	0.003
8 307	0.019	6 907	0.016	2 343	0.013
1 212	0.003	1 201	0.003	501	0.003
1 305	0.003	736	0.002	425	0.002
545	0.001	417	0.001	250	0.001
4 468	0.010	1 535	0.003	1 299	0.007
9 277	0.067	24 575	0.056	9 713	0.052
5 778	0.059	21 619	0.049	8 544	0.046
1 373	0.003	1 159	0.003	459	0.002
2 126	0.005	1 797	0.004	710	0.004

ER IRON MINING Co for 1 week of 17 8-hr shifts, with s. Cars, 55 cu ft, gable-bottom, weigh 3 560 lb (Fig lb; prevailing grade, 1% in favor of load; total, 890 ; aver haul, 1 488 ft = 4 375 ore-ton-miles = 10 943 ive. Total tramming energy, 2 423 kw-hr, aver = uding empty return haulage. At a SOUTHWESTERN ul ton-mile was 0.875 kw-hr at the motor. COPPER notives, making 10 trips per 8-hr day, hauled 18 cars, 08 tons (empty, 30.6 tons including motor); aver haul

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one way, 2 900 ft; power, 3 066 kw-hr per locomotive per month of 240 wo
0.97¢ per kw-hr; total operating cost per motor per month, $212; aver repa
per motor per month; aver total operating and repair cost, 4.5¢ per ore-ton-
cluding trolley line maintenance and repairs. COPPER MINE IN ARIZ: 3.5, 6,
electric locomotives hauled aver distance of 3 000 ft (max, 8 475 ft) at cost
ore-ton-miles, of 13.9¢ per ton of ore, or 21.78¢ per ore-ton-mile, including rep
motive track and ditch. At BISBEE, Ariz (1912), cost for 408 000 useful t
cents per ton-mile, was: for maintenance of locomotives, 2.95¢; cars, 1.64¢; t
trolley, 3.80¢; power, @ 1.1 kw-hr, 1.6¢; total, 20.03¢. COPPER MINE IN M
locomotives hauled 15.4 tons gross, 2 000 ft one way, 7.8 ton return round t
30 min; 420 working hr per month, 1 040 kw-hr per locomotive, repair cos
IRON MINE ON GOGEBIC RANGE; track gage, 24-in; 30-lb rail; 1% grade;
motive; gross trip, 42 tons; empty, 20 tons; aver haul one way, 1 450 ft; 12
monthly cost per locomotive, 200 working hr, 1 854 kw-hr at 1.5¢; oil and w
motorman and trip tender, $122.50; aver repairs (approx) per locomotive,
car, $0.91. Aver total cost, 81.8¢ per ore-ton-mile. Mine operating on half tir
reported. IRON MINE IN MICH. 4 and 6-ton locomotives; track gage, 24-in
grade, 1% aver in favor; reverse, 4%. 8 trips of 8 cars, weighing gross 37.3,
tons; haul, 900 ft one way; 3 750 kw-hr per locomotive per month of 200 h
and repair cost, $124.75 = 14.06¢ per ton-mile. Motor haulage costs in
COLLIERIES (Gen Electric Co) are given as follows, deprec @ 5 % for 200 w

Company	Cost per day		Duty, tons	Cost ton
	Operating	Deprec		
Glen Ridge, Scranton, Pa.....	$7.96	$1.90	288	2.7
N Y & Scranton (Sturges).....	5.05	1.52	250	2.6
" " (Tunnel).....	4.67	1.75	600	1.0
Hillside, No 2 shaft.....	16.47	5.20	989	2.1
" F C slope.....	9.37	2.60	541	2.2

17. Rope Haulage (22, 23, 24)

Systems: (*a*) self-acting or gravity plane; (*b*) engine plane; (*c*) ta
endless-rope. Of these, (*b*) and (*c*) are commonest. CONDITIONS for
age: large tonnage, and grades which make locomotive haulage ineffici
Table 9). Straight haulways and uniform grades are desirable but no

Ropes, sheaves, and rollers. ROPES on flat grades, or wherever especia
to abrasion, as in dragging on the ground or over track ties, should have
strands. The small-gage wires of standard 19-wire hoisting rope wear too
Art 14). A short length of chain at the coupling protects rope from dam
bending, if the cars overrun. Horiz SHEAVES or vert ROLLERS on curves shoul
alined, and of large diam, to reduce bending stresses; 1 large sheave is better
small ones. Where rope crosses the rail on curves, guide strips should be
minimize friction (Fig 56, *b*). Track rollers aver 5 in diam by 12 to 18 in lo
to 25 ft on straight track; closer on curves. At points of special wear, C-I ro
(often made of a pair of car wheels, bolted together with flanges outside).

Self-acting or gravity plane. Descending loaded cars raise t
It is often used on the surface, is effective in slopes or long rooms on
flat for mineral to run in chutes by gravity, and in general serves to t
to a lower level, on main haulage lines, or from a blind level to main

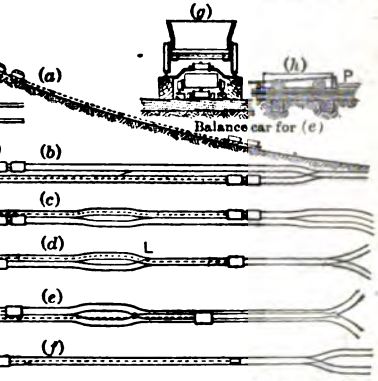
Angle of flattest self-acting slope. $Sin\,\alpha =$ (total friction F, lb) ÷ (wt of
of unbalanced portion of rope R, lb), in which $F = [$(wt loaded and empty c
on both tracks, lb) $\times \cos\alpha + f'$ (wt rollers + drums, lb)$] \times f$; where $f' = 0.1$
coeff rolling friction = 0.012 to 0.025. Friction is often figured on basis of
only, in which case f should be larger. This equation shows that profile of pl
concave, and that heaviest trip, highest ratio of useful load, shortest distanc

rkable slope. Using a tail-rope cancels unbalanced R, teepened at top, or "knuckle," so that the grade exceeds rting trip promptly. Acceleration $a = [(P_d - P_a) +$ = accel, ft per sec per sec; P_d = descending pull, lb = ing pull, lb = $(T_t + R)$ $(\sin\alpha - f\cos\alpha)$; T_l and T_t = = slope angle. If slope changes, take value for α at top for R. Length of slope l (ft) required to attain veloc v nsion R_t in rope will occur when starting or stopping with $(\sin\alpha - f\cos\alpha)$. Number of cars required per trip to

$$\left| + \left[\left(\frac{\sin\alpha}{f} - \cos\alpha \right) W_1 - \left(\frac{\sin\alpha}{f} + \cos\alpha \right) W_2 \right].$$

ope, lb; W_1 and W_2 = wt of 1 loaded and 1 empty car friction.

54, a to i). It is best to avoid switches on planes by n (d) a simple automatic switch is used at L, as shown in



for Self-acting Planes (Hildenbrand, except e)

pivoted at KK are thrown by the descending car for ack plan for narrow slopes, avoiding switches and saving h) illustrate a single main track, with inner counter-wt enough to haul up the empty car and unbalanced rope. nk "growler" or safety pin P, to stop car if rope breaks.

raised or lowered on a slope by a stationary engine d shaft. Simple engine planes require single track, , and use power only for raising load. The plane grade (if all are against the load), provided mini- ne angle of rolling friction; it may serve different from main slope, with rollers or sheaves to guide

oo ft long, under aver conditions, trains of 25 to 30 loaded 1.75% grade, while empty cars required 2.25% (22). require 3.5%. A double engine plane has double track, ending trip assists engine to raise ascending trip, thus e. With engine at head of plane, the max rope pull at $= (T_l + R)$ $(\sin\alpha + f\cos\alpha + P)$, where P = coeff of ble-track plane, $D = (T_l - T_t + R)\sin\alpha + (T_l + T_t$ at foot of plane, D is the same as in (a) but rope is bal-

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and engine pull is $D = T_1 \sin\alpha + (T_1 + 2R)(f\cos\alpha + P)$. For signification see under Gravity Plane, above.

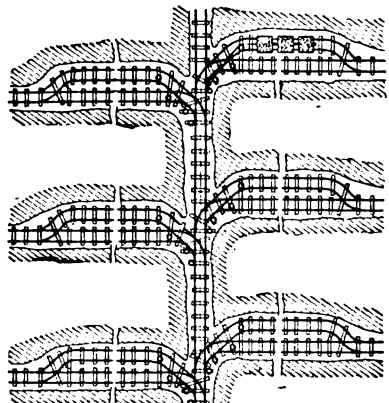


Fig 55. Engine Plane (Leschen)

Tail rope, the comm haulage system, is app straight, curved, incli or undulating tracks, t main line, or including (24). MAIN ROPE hau trip of 1 to 75 cars, wit TAIL ROPE, to shaft bott face, and tail rope, pass a large RETURN SHEAV empties. The engine reversible, with separa clutches and brakes, a placed anywhere on t on the surface with through a shaft, slop bore-hole. If main an drums and engines are ends of run, each rop of run; or, with a do engine at one end, the 2 × length, and both r length of run. Trips a 16 miles per hr.

When main rope is haul trip, the tail-rope drum cl and with light braking t be kept in tension, to p from bumping or over grades. The return run from tail sheave to drum,

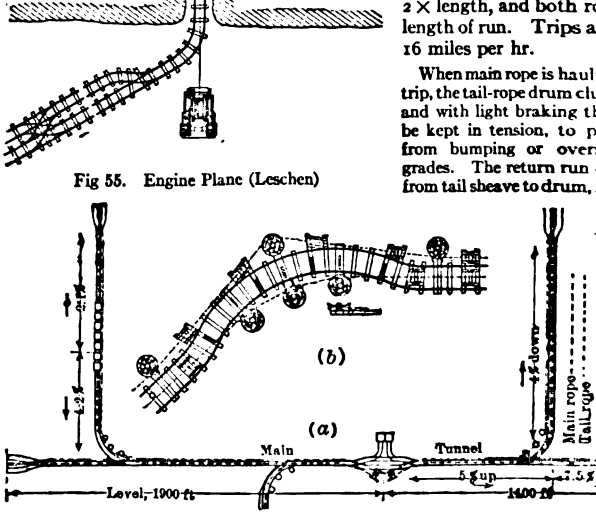


Fig 56. Tail-rope System, Sherrard Mine, Ill (part)

small sheaves 10 to 16-in diam, spaced 25 to 60 ft apart, and attached to roof o as is most convenient. MAX TENSION on haulage rope is: $D = T_1 (\sin\alpha$

where α = max slope angle; h = total drop of line, ft; ope per ft, lb; T_1 = wt of loaded trip, lb; P = coeff of

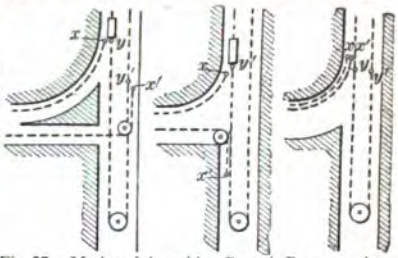
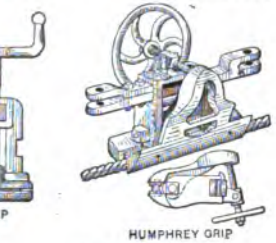
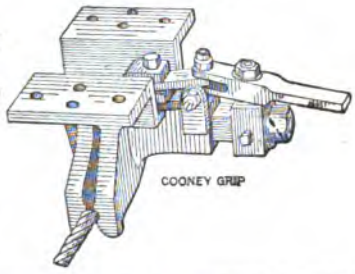
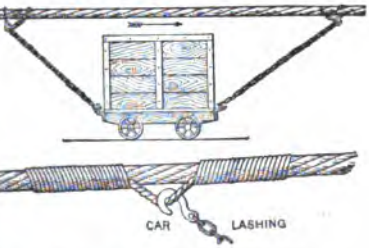


Fig 57. Modes of Attaching Branch Ropes x, x' to Main rope y, y'







ps and Lashing for Endless-rope System

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Cars are attached to rope singly, or in trains, by grips. A single-track syste its travel for loaded and empty cars; double-track system is continuou attached without stopping rope. Aver speed, 4 to 5 miles per hr; max, rope may be over or under the cars. Some foreign collieries support overh cars, requiring steady supply and regular spacing of cars and slow speed (per hr). With under-rope, at switches the rope must pass below ties, by cast rollers. Cars are attached to rope by lashings, or more commonly by (*a*) lever which grips rope either by the eccentric pull of car, on continuou clamping for undulating grades; (*b*) wedge grip (sometimes combined wit dulating grades; (*c*) screw grip, the better designs being good for variab Fig 58 shows various grips and lashing. Double lashing is required on unc Branch lines can be run by a bevel-gear drive, from a fleet sheave on mair rate endless rope on branch, but branches are more difficult to install and tail-rope system. Endless-rope system requires ⅓ less rope than tail-r cheaper, and can take or leave cars at several sidings more easily, but ten increases maintenance, especially on curves. With double track, both ru be between rails, on uncovered rollers so that grips may pass.

Max tension in rope, in lb: $D = f \times \left[\frac{Ml}{v}\left(1 + \frac{2W_2}{W_1 - W_2}\right) \times 2rl\right]$, wh fric; M = mineral output per min, lb; l = length of haul, ft; W_1 and W_2 and empty car, lb; r = wt of rope per ft, lb.

Endless chain is used in U S for short car hauls, and occasionally ab ground systems. Chains are 5 times heavier and cost more than rope o and break without warning; but they have 2 to 4 times the life of rope, are because of wt are good for over-chain haulage on synclines, and cars are by forks. Under-chain is used on short inclines with clips or grip carriage (See catalogues of Jeffrey Mfg Co, Robins Conveying Belt Co, and Link-

Rope-haulage costs are usually combined with gathering cost power charge is estimated, data are only approx. At Sherrard collier when hauling 700 to 800 tons per day, aver haul 4 000 ft, max 2.25 including maintenance of track, wear of ropes and sheaves for rope mule haulage, was 14¢ per ton; rope speed, 10 to 16 miles per hr.

Rope Haulage in Silesia (*Trans* Inst Min Eng, Vol 19, p 5

Method	Distance, ft	Annual ton-miles	
Chain haulage	1 312 to 6 069	1 357 224	
Endless rope, with clips	1 837 " 7 218	3 176 253	
" " with lashings	1 804 " 6 561	1 930 432	
Locomotive haulage	.....	.....	

18. Miscellaneous Haulage Methods

Gravity roads must have sufficient grade in favor of loaded car in motion, the empties being returned by hand, animal haulage,

Grade at the top or "knuckle" must be greater than angle of inertia (must have a starting push; aver slope must exceed the angle of rolling fr required vary with quality and condition of rolling stock and track; for s slope is short, max grade should be less than the probable minimum angle o In rooms or slopes on flat rising pitch, cars can be delivered to loading po animal, or locomotive, and returned to the drift or entry by gravity. On ropes should be used.

Telphers or monorails with buckets are used occasionally where tr impracticable. SHAKING or BUMPING CHUTES are used in flat stopes on U S coal seams, for short local transport (Sec 10, Art 92). Skips o sometimes replaced by conveyers for slopes under 25° (25, 26, 27).

is seldom practicable entirely to eliminate hand development and exploitation should be planned or. In METAL MINES, tramming is usually the only ore, waste, and timber in flat-back or slice stopes, ary workings (Sec 10).

ould be so placed that all material will reach its destina- ween working places and main-haulage system. Thus, at g averages about 300 ft for ore and 500 ft for rock from en mine, less than 200 ft; at Miami, Ray, and other new 25 to 100 ft for normal stopes; on the Mesabi iron range work and seldom over 150 ft in irregular bodies (see s, the miners' contract commonly includes tramming or breast road. Mules or gathering locomotives then p siding on the main haulage and return the empties. ssed air locomotive often hauls to the working face; but, nals, the necessary delays and small tonnages handled in nalysis of cost of gathering and returning cars, to prove tive service is profitable (E. Brackett, *E & M J*, Vol 74, pe haulage systems are more definitely determined by the workings, than when the more flexible prime movers are frequent examples are found of alternating animal and sual cases where locomotives with their trains are taken

g Costs (from Company Reports)

	Year	Tonnage	Cents per ton
.....	1914	910 000	11.6
.....	1914	225 000	12.7
.....	1912	.....	12.0
.....	1912	74 900	18.7
.....	.....	.....	53.0(*a*)
.....	.....	stoping	40.0
.....	.....	.....	5.7
.....	1912	195 200	6.2
.....	.....	.....	15.0
.....	1912	76 500	24.3
.....	1912	415 800	8.0
.....	1914	283 000	13.4
.....	1914	.....	21.6(*b*)
.....	1914	.....	20.5(*c*)
.....	1912	53 874	22.4(*d*)
.....	1912	.....	37.2
.....	.....	.....	35 to 93
.....	1912	35 300	11.7
.....	1912	4 200	21.6
.....	1912	.....	15.8
.....	.....	.....	7.3
.....	.....	.....	39.9
.....	1912	556 000	17.0
.....	1912 1914	209 000	55.1 32.3
.....	1912	.....	21.0
.....	1912	.....	17.4

n. (*c*) From drifting. (*d*) Labor only.

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20. Haulage Accidents (See Sec 23, Art 12, 13)

Approximately 17% of underground fatalities in U S coal mines (6% in metal mines (1911–3) occurred around mine cars and locon approx 3.7% were electrocutions, chiefly by trolley circuits.

To reduce accidents: (*a*) illuminate haulage ways, so that men will not n lights on motor tracks; (*b*) have head light on motor and on rear car of tra ardize equipment, so that bumpers, couplings, etc, are alike on all cars, where the hand can not be caught; (*d*) have safe clearance between cars on ably both ribs of entry (a close rib is better than one with unsafe clearance); Bureau of Mines, recommends refuge bays with 2.5 ft clearance at not less intervals; (*e*) use block fillers to top of rail-web in flangeways and wedge s and switches; (*f*) use low-voltage trolley current; support wire at short that sag will not exceed 3 in, and, when less than 8.5 ft above rails, guard wi lower than wire; do not run wire over footway side of track; avoid trolley sy roofed, wet workings; (*g*) start locomotive only on signal from trip rider an warning bell, and sound bell before all junctions; (*h*) keep car doors and lat and inspect reclosing; an open side door is especially dangerous; (*i*) keep re age delays, and offer bonus to driver having minimum derailments and locom

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ECTION 12

ANT, SHAFT POCKETS D ORE BINS

BY

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WINDLASS AND WHIM

1. Hand Windlass

Windlass has low efficiency and small capacity; hence, limited to sinking winzes, beginning shafts, and small-scale development prospecting, economical limit of depth is, say, 75 to 100 ft.

Principal dimensions for ordinary windlass (Fig 1): Diam of barrel, 6 t to suit size of shaft; crank arm, 14 to 16 in; length of handles, 15 to 18 in; of barrel above platform

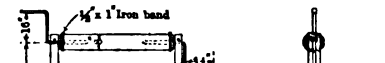


Fig 1. Hand Windlass

$12 to $18 complete, with cost of labor than

Capacity and cost An average man can of work per min, app 20 lb at crank. Hence ity is 125 ft per min 15-in crank, gives 16 r ing speed of 33.5 ft per barrel. At 75 lb gross load per man (125 × 20 ÷ 33.5 = 75, nearly), allo time for filling, dumping, and lowering bucket, 2 men can hoist 7 200 lb from Fig 2 shows cost and capacity, for different depths, with 2 men and 1 buck per hr, 3 min being allowed for filling, dumping, and lowering.

Manila rope, 0.75 to 1 in diam is best for windlass; wire rope tends to kink and is not suited to small diam of barrel.

Buckets may be of heavy sheet iron, or made from half of an oil barrel, with iron bands top and bottom. Bail, 0.75-in round iron, with eye at center; ears for bail are the ends of a 0.25 by 1.25-in strap, passing down each side and under bottom of bucket, with 3 or 4 rivets on a side. Wt of bucket, 60 to 75 lb (Art 21).

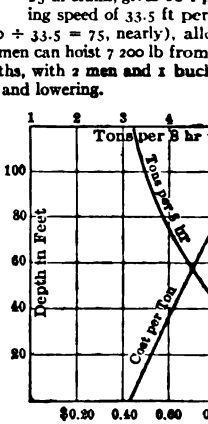


Fig 2. Hoisting with

2. Horse Whim

Uses. For sinking when depth and capacity required are too great for a windlass; for preliminary or small-scale development work, not warranting a power hoist; or for beginning a shaft while installing steam or electric hoist.

Construction. When made at the mine, whim is entirely of wood exc small quantity of bar iron (Fig 3) (1). This design is called "malacate American countries. Axis of drum is vertical, and arm or sweep is attac

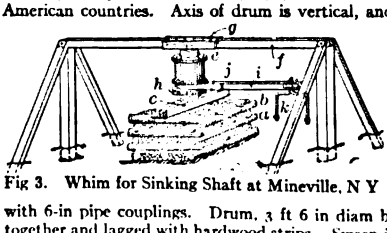


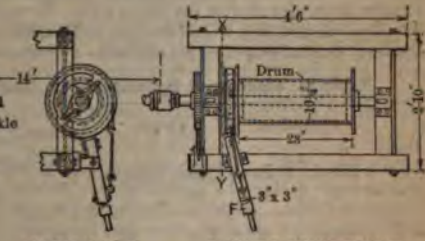
Fig 3. Whim for Sinking Shaft at Mineville, N Y

bottom of drum. If at horse passes under ro head sheave; if at top rope, which is then led o trench. DETAILS OF Fi tion *a* of rough logs, dri filled with stone. Footst drum of two 10 by 12-i bers *b*. Spindle *e*, 12 down at ends *c* to 6 in dia with 6-in pipe couplings. Drum, 3 ft 6 in diam by 2 ft 6 in high, of 3-in together and lagged with hardwood strips. Sweep *i*, 6 by 12 in, is mortised t reinforced with 3 by 6 in by 5-ft pieces *j*. Crosspiece *f*, 10 by 10 in by 30 ft at center by 5-ft pieces, and supported by 12 by 12-in posts,

Yoke *k* is of 3 by 0.75 in by 6-ft steel, swedged to 2 in
of sweep, so that horse can turn sharply and travel in
f 3-in planks, may be added for safety. With this whim
oad of 2.5 tons at 27 ft per min. Cost of whim, $70.
e bottom bearing may be made by a socket in a large
the ground.

ot be less than 12 to 14 ft, as a horse does not work
of less than 24 to 28 ft diam. Decrease in hoisting
counteracted by increasing drum diam; and this
h of sweep should be 5 to 7 times radius of drum.

ainable in a number of forms from makers of mining
s a horizontal cast-iron drum, on a wooden frame, and
ars from a vertical shaft, to which sweep is attached.
rated capacity for one horse on low gear is 2 400 lb, on
2 000 lb; cost, f o b factory, $125 to $150.
g, has a large steel-plate or cast-iron drum, on a vertical
ring may be done by brake. Hoisting speed is greater
t power loss due to gearing is avoided and wear on rope
; to 6 ft diam; for 2 horses, 6 to 8 ft. Shipping wt, 1 200



SECTION X-Y PLAN OF DRUM FRAME

m and Drum used in Joplin District

o to $150. Fig 4 shows Joplin, Mo, type of whim, in
of gear and brake applied with one lever *F*. Drum is
l connected to the horse-power gear by a shaft with a

Capacity,
nin; cost,

ting. For
power of
min), time
d lowering
g 5 shows
m hoisting
r following
o ft-lb per
nin; gross
lb; wt of
2 men and
time for

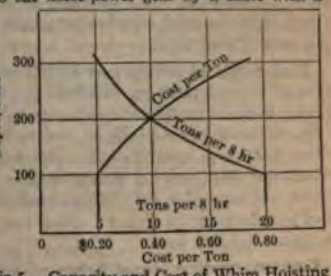


Fig 5. Capacity and Cost of Whim Hoisting

t min (requiring good arrangement at shaft mouth); low-
For depths of 50 to 100 ft the duty and cost are constant,
at 4 shovelers (maximum number that would be employed

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under the conditions) could load in time required for each hoist. (**Note.** — Ma rating of whims is sometimes equivalent to 2 or 3 times the average work of a

Rope for whims is sometimes hemp, but is better cast steel, of ordinar ⅝ in diam. (For wire rope data, see Art 14.)

Buckets should be light; ⅛-in plate is thick enough, if reinforced at shallow bucket is better than a deep one, as it is easier to fill and dump. Dia at top, to prevent large stones from wedging fast when dumping; but for in where bucket slides on skids, a barrel-shaped bucket is necessary. Bucke attached at top are safest, but those swinging on trunnions on the sides, bel gravity, are easier to dump. Proper capacity for 1 horse, 8 to 9 cu ft. An 8- of ⅛-in plate, weighs 130 to 150 lb and costs 8 to 10¢ per lb.

HOISTING ENGINES

3. Geared Steam-driven Hoists

Small geared hoists, for prospecting, development, or mines of put and depth, are self-contained, with inexpensive foundation, an design. For smooth running, gears are cast from machine-cut patt cut from the solid are better, but cost more. Ratio of gearing, from 1 at least 2 teeth of pinion are always in contact with those of spur g cone and band friction gear, see Art 5.) Drums should be at least 20 in less the diam, the shorter the life of rope. Lowering is usually don but reversing gear may be had at about 10% added cost, and lowering u is safer. Cost of first-class geared hoists averages about 10¢ per lb (

Table 1. Lidgerwood Portable Hoists, Single Cone-friction D

Rated h p	Cylinders		Drum		Average hoisting load, lb	Average hoisting speed, ft per min	Bed-plate	
	Diam, in	Stroke, in	Diam, in	Length, in			Width, in	Leng in
12	6.25	8	24	26	1 500	250	49	55
20	7	10	29	26.5	2 000	275	54	72
30	8.25	10	34	32	3 000	350	55	74
35	9	10	41	40	3 500	350	73	74

Medium-size geared hoists are suitable for more extensive d work, or mines of small tonnage, to say 500 ft in depth. They ar adapted for hoisting with buckets, light cages or skips, at speeds of 4 per min (Table 2).

Large geared hoists may be self-contained, or with engines and ported on independent bed-frames. They are especially suited heavy loads at moderate speeds, up to say 1 500 ft per min; hence, tageous for slopes or inclines, where high speeds are not permissib these conditions, geared engines give better steam economy, due to hi speed and ability to use drums of large enough diam to prevent und stresses in rope (Art 14). Geared hoists are useful where economy of and foundation is an object. Their cost is 35 to 50% that of direct-a for same hoisting load.

Constant-running geared hoists are sometimes used, as in the Michig copper districts. Two to 4 drums, serving as many compartments, are clutches, lowering being done by brake. By using a flywheel and Corliss economical working is secured, provided the drums can be kept in fairly cont ation. If made reversing, these engines evidently lose their advantage.

ver-Morgan Single- and Double-drum Geared sts, Band Friction Clutches

in 1 layer	Gear ratio	Unbalanced load, lb, with 100 lb steam	Rope speed, ft per min	Weight		Over-all dimensions		
				Single drum	Double drum	Width, single drum	Width, double drum	Length
						ft in	ft in	ft in
410	1 : 5.5	4 700	480	8 700	15 000	7- 8	12-0	7- 6
450	1 : 5.5	5 000	520	10 200	17 500	7- 8	12-0	7- 6
540	1 : 6	7 300	490	17 300	28 600	9- 0	13-6	9- 4
540	1 : 6	7 800	470	19 000	30 000	9- 0	13-6	9- 4
540	1 : 5	7 800	550	26 400	40 000	10- 8	15-6	10- 7
480	1 : 4.5	8 700	610	37 000	51 100	10-10	17-3	11-10
570	1 : 4.5	7 200	730	39 000	56 600	11- 0	17-3	11-10
570	1 : 4.5	8 300	730	44 500	60 800	11- 0	17-3	11-10

mates, cost may be taken at 10¢ per lb at factory.

irect-acting Steam Hoists

ıcting hoists are useful for large output, or for ng speeds. Besides the advantage of eliminating w speed of stroke permits use of Corliss or other n. Though usually designed for greater economy first cost is higher, and they should not be used for o ft, unless output is large.

ines have a limited application. Their high cost is expensive, and large loads are hoisted in rapid suc- When not operated condensing (condensers often ting service), their economy is but little better than g into account the intermittent running. They are vin tandem-compound. Though more costly, the because in the cross-compound the high-pressure gh to start the load when low-pressure cylinder is on ın economical ratio of cylinder volumes. Throttle signed so that, in starting and until full speed is steam at such pressure as will give it a starting effort is is done in 3 ways: (*a*) By exhausting h-p cyl into ing l-p cyl with steam at proper reduced pressure exhausting h-p cyl into the receiver and cutting off ı then receives steam at required pressure through a g receiver with steam at the reduced pressure for it also the h-p cyl. First method gives the best ım economy. Third method is simplest and requires in tandem-compound, cylinder ratios can be pro-

lor best operation is a mooted question. Though relation inder areas should be such that half the work is performed ys practicable with hoisting engines, on account of adverse ame of the h-p and V that of the l-p cyl, then the ratio of $V \div v = 0.85\sqrt{R}$ to $V \div v = \sqrt{R}$, R being ratio of expan- R^2).

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Area of l-p cyl is first calculated as if all the work were to be done in Approx diam of l-p cyl is found by:

$$D^2 = \frac{8\,500 \times \text{horse-power}}{\sqrt{p} \times N \times S}$$

in which D = diam of l-p cyl, p = gage pressure, N = rev per min, S = For cross-compound engines the horsepower would be that required for period, because during acceleration the h-p cyl is operated at late cutoff mitted direct to l-p cyl. For twin tandem-compound engines the horse that required during full-speed period.

Cylinder diameter for each cyl is found by:

$$\text{Area of cyl} = \frac{\text{horse-power} \times 33\,000}{\text{m e p} \times \text{piston speed}}$$

$$\text{Mean total pressure} = \text{terminal pressure} \times (1 + \text{hyp log } R)$$

R = nominal ratio of expansion r in h-p cyl = length of stroke in ft ÷ cu

$$\text{Mean effective press (m e p)} = \text{mean total press} - \text{back pres}$$

$$\text{Absolute initial press} = \text{abs terminal press} \times r$$

$$\text{Total expansion in both cyls} = \frac{\text{abs initial press in h-p cyl}}{\text{terminal press in l-p cyl}} \div$$

First calculate area of l-p cyl for doing total work. Terminal press in l-p c from which R is computed for l-p cyl, and mean total press is found from m e p from Eq 4, the abs back press being assumed. Area of l-p cyl is fou If half the work is done in l-p cyl, the m e p would be half that found from E total press is then recalculated. From Eq 3 the value of R for l-p cyl is obt recalculated value of mean total press and the terminal press for l-p cyl. for l-p cyl (Eq 5) is equal to terminal press in h-p cyl, assuming no drop betw R for h-p cyl is then equal to abs initial press at throttle divided by termin cyl. Compute mean total press in h-p cyl from Eq 3, and m e p from Eq 4, in this case being equivalent to terminal press in h-p cyl or initial abs pr Area of h-p cyl is then calculated, or area of h-p cyl may be quickly foun area of l-p cyl by R for that cyl. In above calculation no account is take compression, drop between cylinders, nor area of piston rod. To cover th m e p of each cyl is multiplied by a DIAGRAM FACTOR, which may be tak compound hoisting engines.

Examples of compound cylinder ratios. Old Dominion Copper C tandem condensing, 17 and 31 by 48 in, ratio of areas 1 to 3.4; 20 and 37 by to 3.45. Copper Queen Consol Mining Co, Ariz, twin tandem condensing, 48 in, ratio 1 to 3.08. Homestake Mining Co, So Dak, duplex cross-compoun 28 and 52 by 42 in, ratio 1 to 3.47. Randfontein, Transvaal, cross-compoun 22 and 40 by 48 in, ratio 1 to 3.3. Village Deep shaft, Transvaal (Table 4) condensing, 17 and 28 by 60 in, ratio 1 to 2.73; duplex cross-compound ge and 21 by 20 in, ratio 1 to 2.25. Cambria Steel Co, Johnstown, Pa, cross-c densing, 28 and 50 by 48 in, ratio 1 to 3.2. Grand Central Mining Co, tandem, 16 and 24 by 42 in, ratio 1 to 2.25.

Table 3 shows test of a Nordberg twin tandem-compound condensing and 28 by 48 in, Corliss valve gear; 2 clutched drums, each 7 ft diam, hold 1.25-in rope; total rope pull, 19 000 lb.

Valve gear of hoists should be as simple as is compatible wit struction, and economy in steam consumption; with minimum num and all motions positively controlled when possible.

Three types used in American engines: slide, piston, and Corliss valve. are always used for small and generally for large geared hoists, and for many hoists. In small engines, valves are unbalanced; in large, balanced valv used. PISTON VALVES are used in the simpler forms of heavy-duty direct they are balanced as to steam pressure, and are well adapted to high press and also to long-stroke engines, because each end of cylinder has its own v ducing length of ports. But since the piston valve is not held on seat by s

ento Hoist, Copper Queen Consol Mining Co
11. (Charles LeGrand, Cons Eng)

	A M 7–11:30	P M 12–3:30	Total shift 8 hr
ted)	144	144.5	144.25
ed)	137	137	137
	430	430	430
	19.7	19.9	19.8
	337 700	199 000	536 700
	79 100	.....	79 100
	248 900	312 900	561 800
	177 700	280 900	485 600
	104 500	28 200	132 700
	947 900	821 000	1 795 900
t hp-hr	435.88	418.20	854.08
skip, shaft hp-hr	23.33	20.05	43.38
	459.21	438.25	897.46
	102.05	125.21	112.18
	910	1 010	968
ding condensation			
	13 942	12 158	26 100
	1 215	945	2 160
	15 157	13 103	28 260
condenser, lb	30.36	27.74	29.08
luding condenser, team for oil pump, lb	33.00	29.90	31.49

kip for 232 skips hoisted, 3.81 tons. Wt of empty skip, lb per ft. Condensation in pipe line from boilers, 625 hoist not running. Steam for condensing plant charged w being used.

High cost of CORLISS GEAR is justifiable when fuel is d in decreased maintenance cost exceeds the added first pecially suited to large output from depths of 1 000 ft or from the automatically controlled variable cutoff, which han to other valves. **Variable** cutoff is operated by ball ssive speed), or by an **auxiliary** lever to throttle, by which ened. With slide and piston valves, cutoff may be short- erferes with exhaust. Corliss valves with non-detaching ntage of separate steam and exhaust valves with quick expansion gear must automatically return to position of r slowing down of hoist. As Corliss gear is limited to is rarely applied to geared hoists.

is in balance, or if lowering is done by steam when not tion type, or a modification of it. Both open and crossed rods are best as they increase lead of the valve as cutoff ssion and earlier admission of steam for cushioning when Corliss hoists, valve rods are sometimes driven from a ed by a hollow shaft geared from main shaft. Inner but may be rotated independently by the reverse lever, erse the engine. Medium size direct-acting and geared e hoists by an auxiliary engine (Art 5).

g engines is necessarily poor, due chiefly to inter- **ust start under full load** and have rapid acceler-

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ation, thus requiring uneconomical admission of steam during ne
Also, intermittent work involves loss of stored energy near end
braking is done, and the large variation of load within the short
prevents economical operation. The frequent stops and peri
allow cylinders to cool and increase condensation loss. Superh
condensation, and is justifiable for large hoists. Hoisting eng
about 0.85 stroke in starting, and with slide or piston valve this is
engine comes up to speed, when hoisting from shallow shafts. T
cards are useful for determining condition of valves and pistons,
comparative, rather than absolute, because with varying speed a
engine strokes give the same card. Hence, steam consumptio
from hoist indicator cards is an approximation at best. Steam
unit of work may be obtained very closely by measuring water su
if plant is so arranged that one or more boilers supply the hoist
auxiliaries. Such tests should last several hours, useful work b
from tonnage hoisted.

Steam consumption varies between wide limits. Small hois
from boiler require 150 to 175 lb per shaft hp-hr; for large compo
hoists, winding from deep shafts, 25 lb per shaft hp-hr is obtaina
ate size, simple, non-condensing hoists, with slide or piston valve

Table 4. Test of Allis-Chalmers Duplex Tandem-compound Hoist, Whiting Type, Village Deep Shaft, Transva

Cylinders	17 and
Whiting hoisting sheaves, 5 grooves each	12 ft
Duration of test	7 hr 5
Total ore hoisted from vertical depth of 2 290 ft	1 542
Total indicated work in cyls	4 145
Total steam consumption	45 579
Aver initial steam press (abs)	132.3
" final steam press (abs)	2.0
" moisture in steam	0.5
" shaft horse-power	251.8
" indicated horse-power	295.6
" steam consumption	6 434.6
" steam consumption per shaft hp-hr	25.5
" steam consumption per i hp-hr	21.7
Mechanical efficiency	85.1

Main engine, jackets and condenser pumps, consumed 23.707 lb steam p
92.775%. Reversing and brake engine consumed 1.846 lb steam per shaft

Steam consumption per i h p of 2 direct-acting hoists (3):

No 1. Simple, slide-valve, condensing; cylinders, 24 by 42 in; fixe
steam press, 60 to 70 lb; depth shafts, 2 000 ft; average hoisting speed,
i h p, 316; steam per i h p per hr, 30 lb.

No 2. Simple, piston valve, condensing; cylinders, 24 by 48 in; cutof
then shortened as speed is attained; steam press, 110 to 120 lb; vacuu
10 in at middle of trip; depth, 2 300 ft; average speed, 1 760 ft per
steam per i h p per hr, 21 lb. In both cases the i h p and steam consu
culated from average of indicator cards taken every fifth revolution be
second; hence results are comparative, rather than a positive measure o

Steam consumption of a twin tandem-compound condensing hoist,
Copper Queen Mining Co, Ariz, is given as 32 lb per shaft hp-hr. This is
operating conditions and includes power for circulating condenser water

5. Drums, Brakes, and Clutches

Drums are cylindrical, conical or cylindro-conical. Small dr
one piece, with brake ring and clutch surfaces. Large drums ar

ortation and to prevent shrinkage stresses. Drum ast in one piece and each spider separately, or shell o reduce weight the shell may be of steel plate (Fig 6), y when conical) may be of cast steel instead of iron.

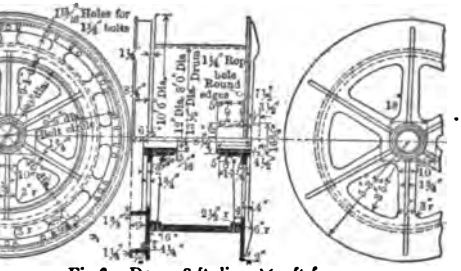

Fig 6. Drum, 8 ft diam × 4 ft face

antages compensate decreased economy of operation. a small internal drum, on which is wound extra rope pened.

ke conical drums, compensate the weight of rope. occupied, absence of fleet angle, permitting hoist to ghter weight, permitting quicker acceleration. Dis- t more, have shorter lives, and are heavier than round rt 14).

ost type (Fig 7, 8 and 9), applied through systems of ot, or auxiliary power cylinder, according to size of used for both large and small hoists, post brakes only evers are sometimes designed to increase the pull on plication of braking force; more desirable for hand nd emergency brakes, on hoists equipped with power tial motion tending to tighten the brake automatically instantaneous action is a disadvantage.

g diameter of breaking surface is same as that of drum, ust stand a greater tension than that of the rope

tween band and brake surface = pull on rope,
pectively at anchorage and at end of brake band,
perian system of logs = 2.71828,
of friction, wood on iron = 0.50,
gth of arc of contact to radius of brake surface,
c of contact, deg.

$$W = T_1 - T_2 \tag{6}$$

$$+ e^{f\alpha}, \quad \text{or} \quad \frac{T_1}{T_2} = e^{f\alpha} = e^{\left(f\frac{\pi}{180}\alpha\right)}$$

$$= f\frac{\pi}{180}\alpha \log_{10}(2.71828) = \frac{\pi}{180}(0.4343) f\alpha$$

$$\frac{T_1}{T_2} = 10^{\frac{\pi}{180}(0.4343) f\alpha} = 10^{0.0076 f\alpha}$$

$$\text{and} \quad f = 0.50, \quad \frac{T_1}{T_2} = 10^{1.455} = 10.55 \tag{7}$$

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Substituting in Eq 6, $W = 0.905\ T_1$; whence, T_1 is about 1.1 times the rop
best conditions and with drums nearly at rest. If braking surfaces are in
tion, f is redu
increased. If br
plied suddenly,
motion, T_1 may
or tripled, if tens
ficient. Hence,
loads, the limit o
ing with practi
ratios is soon
hoisting in ba
over and under
drum, brake ba
in halves (Fig 7)
anchorage will
the load due to
otherwise one r
against the bra

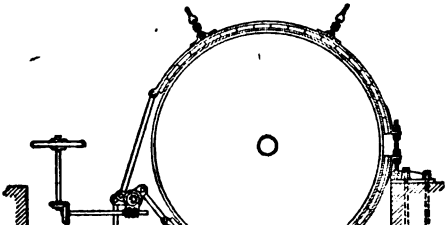


Fig 7. Band Brake

Post brake requires a smaller movement for necessary clearance
ring. Also, the load is always carried by the anchorage, regardless
in which the rope winds upon drum. Brake frames 2.5 ft radius or less may be pivoted at lower ends, and operated by bell crank and rod (Fig 8). Those over 2.5 ft radius should have a PARALLEL MOTION, and are then supported by sliding shoes moving on the sole plate, or by swinging links (Fig 9). Post brakes increase first cost of hoist 4 to 8%.

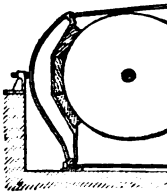


Fig 8. Post

Clutches are of jaw or friction type. JAW CLUTCH is simple and safe, but, because of its positive action, both drum and engine must be at rest, or nearly so, when clutch is thrown in gear, and the two parts of clutch must be in certain
tions before they will mesh. It is suitable where hoisting is generally in bala

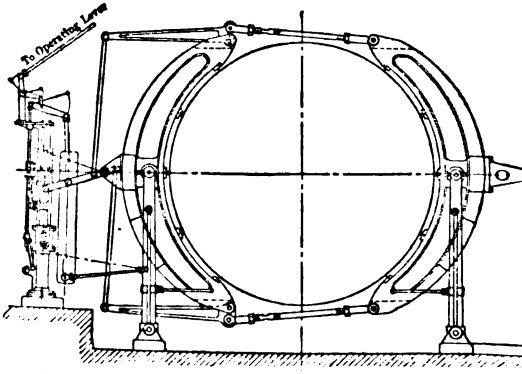


Fig 9. Parallel-motion Post Brake, Operated by Compressed-air Cylinde

one level. FRICTION CLUTCH is preferable where relative positions of cages
frequently, or a nice adjustment of distance between them is necessary. It
to general mining purposes. CONE FRICTION CLUTCH is satisfactory for drun

rger drums. But, with large drums, heavy loads, or high m-shaft bearings and friction on thrust pin cause trouble. Ex- ating rust haft. rated rrion rwise ected

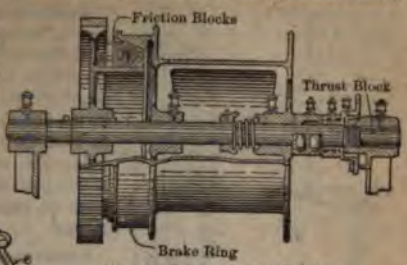
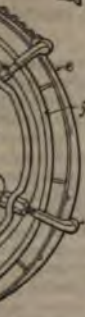
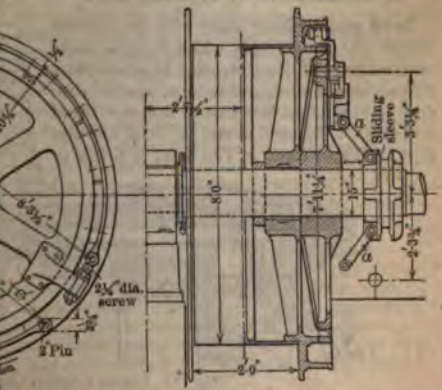


Fig 10. Exeter Patent Friction



on Clutch

and adjusted for wear. Fig 11 shows the Lane type. It comprises a cast-iron spider *a*, keyed to drum shaft, and fitted with fixed arm *b*, and movable arm *c*, pivoted at *d*. The arms carry steel band *e*, lined with wooden blocks. Inner end of *c* is connected to sliding sleeve *f* by toggle *g*. When the sleeve is moved toward drum the band tightens on the friction ring *j*. When released the band is prevented from sagging by the lifters *k*. On drums 8 ft diam and over, the band is usually made in halves (Fig 12). As toggle levers *a* approach the vertical the tensile strain in the band becomes very great, f clutch is set too early. Since clutch path may be greasy, .2. With the rope pull reduced to its equivalent at periph- n at ends of band may be calculated as for band brakes.



ion Clutch for 24.5 × 72 Corliss Hoist (4)

on drum that the greatest pull is on fixed end of band; less clutch. It is recommended (4) that the leverage between eve be 20 to 1. Assuming efficiency of 50% for the clutch

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gear, when the band tension and force required on the sliding sleeve are
of clutch-engine cylinder is determined by working back through the leve

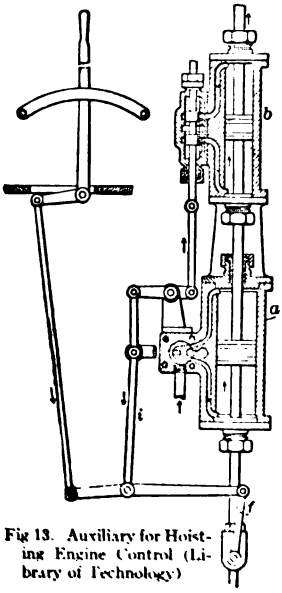

Fig 13. Auxiliary for Hoisting Engine Control (Library of Technology)

Auxiliary engines for brakes reverse are operated by electric cally driven hoists, by air for ele pressed-air hoists, and by steam power for steam hoists. Brakes be applied by a weight and relea so that they are set automati fails. For reverse and clutch acts in both directions. In stea form of engine operates brak reverse.

Steam operates expansively, and c condensation, operator can not d motion of piston of auxiliary engine ract cylinder is placed tandem wit (Sec 40). Valves of both cylinder by one hand lever with a "floating Opened by hand, the valves are auto by motion of the pistons. Fig 13 tion of ordinary auxiliary engine; *a* and *b*, oil cataract cylinder. Har steam and cataract valves, and mot *d*, acting through *f* and *i*, closes the ment proportional to that of hand place. Leyner steam- or air-oper 14) can be applied to geared hoists

6. Equalization of Hoisting Load

Load equalization deals with the varying weight of hoisting rope, which in deep shafts may exceed net load of ore. It economizes steam consumption, and reduces size of cylinders and therefore first cost of engine. Usual methods: 1, Conical drum; 2, Reed and flat rope; 3, Taper rope; 4, Tail rope; 5, Koepe or

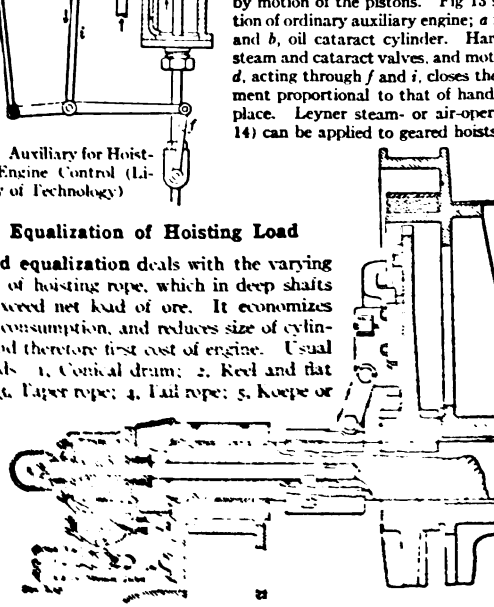

Fig 14. Leyner Friction Clutch

When [illegible] "Hoisting in bal
[illegible] and cars counterbalar
[illegible] sometimes unwieldy
[illegible] equalize rop

y, the varying radius of a conical drum should be moment is constant. But this is true only when the ace is a curved line. Curve-faced drums are costly e not used. A single, straight-faced cone gives per- ginning and end of hoist (Fig 15); moments at all the one ent ing all nly mi- ght and wn, ble aft. ical vice ums. n of not (Art

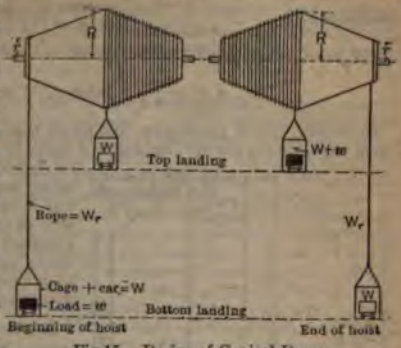

Fig 15. Design of Conical Drum

rge end of drum, ft; r = radius small end of drum, ft; op and bottom landings, lb; W = wt of cage and car, or

litions, the algebraic sum of the static moments in the two ages at top and bottom of shaft must be equal:

$$) r - WR = (W + w) R - (W + W_r) r \qquad (7a)$$

whence, $$R = \frac{r\,(2\,W_r + 2\,W + w)}{2\,W + w} \qquad (8)$$

in which R is the only unknown quantity. Since rope weight appears in only two terms of Eq 7a, the drum as designed equalizes only for the assumed depth of shaft. Width of drum depends upon depth of shaft, size of rope, and angle of drum face. For angles under say 30°, the horizontal pitch of grooves needs not be more than ¼ to ¾ in greater than rope diam; for large angles, pitch may be 2 to 2.5 times rope diam, to make grooves deep enough to prevent displacement of rope. Number of grooves = rope length divided by mean circumference of drum. Cylindro-conical drums (Art 5) may be used for deep shafts, to avoid excessive width of drums (Fig 17). This is at expense of equalization.

Passing point of cages. Let S = distance of passing point below surface, ft; l = distance between top and bottom landings, ft; d and D = respectively small and large diam of drum at bottom of groove, ft; a = rope diam, ft; n = number coils of rope on one drum with cage at top s (2 or 3 extra coils at each end being usually allowed).

$$\frac{l}{\left(\frac{D+d}{2} + a\right)} = \frac{2\,l}{\pi\,(D + d + 2\,a)} \qquad (9)$$

elow middle of shaft) the number of rope coils on *abce* = ssing point is reached when drum has made half the revolu-

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tions required for a complete hoist, and both ropes are winding on me
of rope unwound from $bfgc = S$, whence:

$$S = \frac{\pi\,(bc + fg)}{2} \times \frac{n}{2}, \quad \text{and} \quad \frac{bc + fg}{2} = \frac{\frac{d+D}{2} + D}{2} = \frac{3D}{}$$

Substituting, $$S = \frac{\pi n}{8}\,(3D + d)$$

If D and d be taken as diam of rope coils,

$$n = \frac{2l}{(D+d)}, \quad \text{whence,} \quad S = \frac{l}{4}\,\frac{(3D+d)}{(D+d)}$$

To increase rate of acceleration, and attain max speed sooner, dru
coned steeply for a few rev, remainder of drum being cylindrical. T

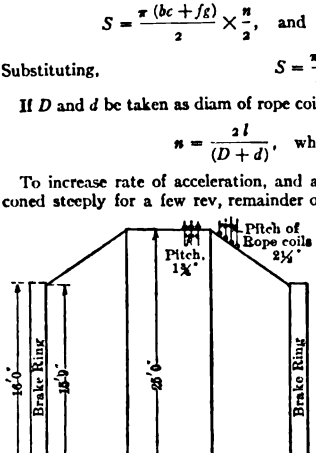

Fig 17. Cylindro-Conical Drum, Tamarack Shaft No 5

for electric than steam hoist heavy starting current re with induction motors. Fi sions of cylindro-conical dr Tamarack Mining Co, Mich weighs 300 000 lb. Each co 1.5-in rope; the center, 3 87 of each rope, 6 000 ft.

Reels. Flat rope (Ar creases the winding radiu an increment equal to hence, the result is theore and same calculations ap rope and conical drum. flat rope, practical consi thickness for a given widt winding increment so tha even more imperfect th drum. Advantages of r and lower first cost than for drums; elimination of fleet angle vantage: lack of the steadying effect on engine produced by the l

Design. Diam of reel hub d should be at least 60 times the rope th selected rope of proper strength, following are the relations betwee winding diam d and D, length of rope l, and its thickness t, all in between inner and outer surface of coiled rope $= lt$; whence,

$$lt = 0.25\,\pi\,(D^2 - d^2)$$

and, if $D = xd$, $$lt = 0.25\,\pi\,d^2\,(x^2 - 1)$$

whence, $$d = \sqrt{\frac{lt}{0.25\,\pi\,(x^2 - 1)}}$$

in which $x = D \div d$, found from Eq 8. Knowing d, $D = xd$.

Example. Rope, 2 000 ft = 24 000 in of 4 by ¾ in, 2.65 lb per ft 6 000 lb; skip, 3 000 lb. Then $x = (2 \times 3\,000 + 2 \times 5\,320 + 6\,00$ 6 000) = 1.88. Substituting in Eq 13:

$$d = \sqrt{\frac{24\,000 \times \frac{3}{4}}{0.7854\,(1.88^2 - 1)}} = 67.2 \text{ in} = 5.6 \text{ ft}$$

Hence, $D = 1.88\,d = 1.88 \times 67.2$ in $= 10.52$ ft.

Tapered rope (Art 14) assists more in solving problem of deep hoisti ing the load. Theoretically, the rope is of constantly decreasing s maximum diam and total wt are smaller than for rope of uniform se

ced load at beginning of hoist is decreased; with conical ed.

ndrical drum, gives perfect equalization through- loop in the shaft, its ends being attached to bottoms move independently, as for hoisting from different in shafts where hoisting is from one level only (5). bight of the tail rope, and supported in sliding bear- teadies the rope; but in very deep shafts vibration lty.

Drum is replaced by a large, single-groove driving sheave. asses around say 190° to 200° of sheave circumference, . This system is suited to mines of one level only, and a aves are placed in tandem, so that both parts of hoisting To prevent slippage of hoisting rope on driving sheave, elerating force must not exceed wt on empty side minus its a certain percentage, depending on arc of contact and coef to the smaller pull must not exceed $e^{f\theta}$, in which e is the e angle of contact and f the coef of friction (= 0.18 to he sheave there is less mass to be accelerated and retarded may be smaller, and on the large sheaves wear of rope is Cages can not move independently of each other; if rope f rope over sheaves is likely to prevent operation of safety ited to electric hoists, which have constant torque (4).

on of the Koepe, has two 3 to 5-grooved driving sheaves, ngine by connecting and parallel rods. Fig 18 is a dia-

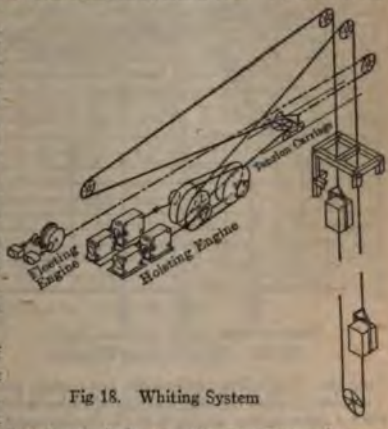

Fig 18. Whiting System

fric- times sting es, or ened, ngine sses, and com- oves nder ough VAN- Koepe ry to sink- riodi-

s of tatic pths, oo-ft

o lb; hoisting in balance. Working stress in rope, taken at 50 000 lb per sq in of steel section, as near ected. Conditions: Cylindrical drums, 9 ft diam, Conical drums, 9 by 18.45 ft, rope 1.25 in, 69.6 coils. diam 144 in, rope ¾ by 5 in @ 3.25 lb per ft, 112.4 t diam, rope 1⅛ in @ 2 lb per ft. Koepe sheave, lb per ft, with 1-in tail rope.

equires largest engine, because of its large diameter; t, but is uniform. Curves for conical drums and

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reels show good equalization when hoisting from shaft bottom; lines, giving moments when hoisting in balance from half the de

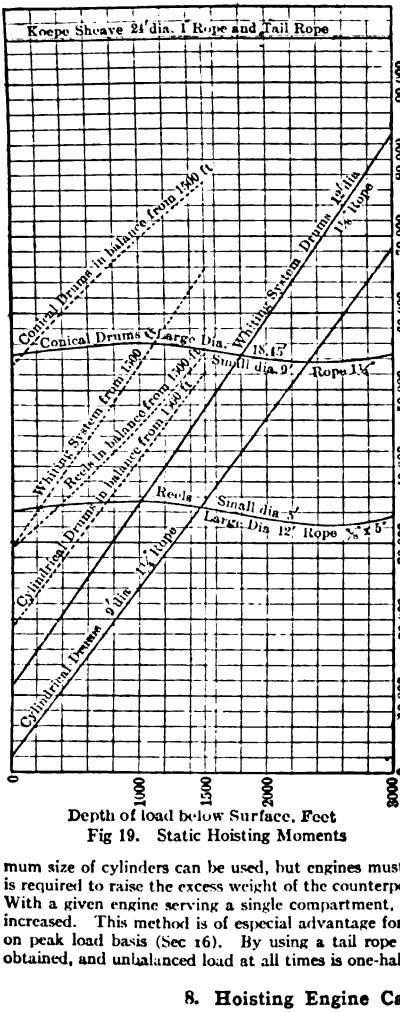

Fig 19. Static Hoisting Moments

larger engine for hoisting f ate levels in b hoisting from

7. Balance balanced

Hoisting i 2-compartme cylindrical dr plest and mo tem. Drums shaft for hoi level only, o for balanced several levels the rope is not forms a cons ing load fron end. Advant plicity, low adaptability t ditions of hoi vantage is gre load in deep

Over-balanc system is appl compartment sl is sufficient roo weight and its hoisting in a ment, unbalan start is: $W +$ the end of the which W = wt or skip, W_r = and w = wt of terweight equal be employed, t balanced load i and at end of t With this arr

mum size of cylinders can be used, but engines must be reversible, beca is required to raise the excess weight of the counterpoise as the unbalanc With a given engine serving a single compartment, net load and daily increased. This method is of especial advantage for electric hoists, if p on peak load basis (Sec 16). By using a tail rope perfect equalizatio obtained, and unbalanced load at all times is one-half net weight of ore.

8. Hoisting Engine Calculations

Factors for general case: 1. Daily output required governs nu per day or shift. This, in turn, depends on load per trip, as dete of car (or other local conditions), depth of shaft, type of plant, and delays and handling men and supplies. 2. Gross load = wt of o

determined by wt of ore, car and cage, multiplied by vert dead into live load, plus weight of rope. Rope ned, and one or more approximations made. (For ; ropes, see Art 14.) 4. Max unbalanced load, when shaft, equals wt ore + rope. 5. Total time for a e factors must be harmonized for selecting type and r conditions; geared hoist for low, direct-acting for

For SINGLE-COMPARTMENT SHAFT, this covers down bottom and top, for caging cars; or, with skips, for is also holds good, in general, for double-compart-ent drums operated by clutches. For DOUBLE-COM-in balance with fixed (keyed) drums, or single drum the total time per load is the elapsed time from the om on down trip until it stops at top on up trip.

th best track arrangements, takes 7 to 8 sec; better , which may be greatly exceeded when loading facil- and power-operated loading chutes, delays may be 10 sec minimum should be allowed. Larger allow-

NOTE. — In hoisting with fixed drums, since load-imultaneously, the longer time, whether at top or r, and one allowance for delays suffices. Besides the m 20 to 40% of each shift is usually occupied by supplies.

od. Time per trip includes periods of acceleration, tion. Acceleration and its period vary widely. For depth, acceleration is from 2 to 10 ft per sec per and large tonnages, it may be 12 to 15 ft. At latter ay run into retardation period, with practically no with conical drum or reel). Uniform speed period and hoisting speed. With geared hoists max speed min. With direct-acting hoists, 4 500 to 5 000 ft ned. Speeds of 2 000 to 3 000 ft are common. Re-ceeds that of acceleration.

per sec per sec; v = mean velocity, ft per sec; v_1 = = hoisting time, sec; t_1 = time of acceleration, sec; c; h = depth of shaft, ft; s = distance passed over hen, $t_1 = v_1 \div a$, and $s = v_1^2 \div 2a$; and if time and d retardation be assumed equal,

$$t_2 = \frac{h - 2s}{v_1} = \frac{h}{v_1} - \frac{v_1}{a}$$

$$+ t_2 = \frac{2v_1}{a} + \frac{h}{v_1} - \frac{v_1}{a} = \frac{v_1^2 + ah}{av_1}$$

$$\frac{ahv_1}{v_1^2 + ah}, \quad \text{whence} \quad a = \frac{vv_1^2}{h(v_1 - v)} \qquad (14)$$

3 shaft, Mich, 4 800 ft in 75 sec, or 3 880 ft per min aver, nin. Whiting shaft, Calumet & Hecla mine, from 4 900-ft . Kimberley diamond mines, South Africa, from 1 560 ft ax 3 770 ft; acceleration period 16 sec, retardation 13 sec, 2 sec. Hence, acceleration is about 3.5 ft per sec per sec. Dowlais shaft, Cardiff, Wales, 2 220 ft in 52 sec, an aver Moss colliery, England, 1 500 ft in 35 sec. Denaby &

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Codely, 2 289 ft in 55 sec. Rhodes Rotherham, 1 650 ft in 45 sec. At Rosel England, speed of 5 100 ft per min is on record.

Usual ranges of hoisting speed are as follows: SMALL GEARED HOISTS, 45 min; large, 900 to 1 200 ft. SMALL DIRECT-ACTING HOISTS, 1 000 to 1 5 large, 2 500 to 3 500 ft.

With fixed output, speed and load are dependent upon each other. I hoisting speed per min; T = output, tons per hr; W = net load, tons; shaft, ft, and t = caging time, min; then,

$$V = \frac{hT}{60W - Tt}$$

Engine details. Minimum diam of drum depends on diam of rope (maximum diam, on hoisting speed required and piston speed of engine. R ING, from 1 : 3 to 1 : 6. PISTON SPEED rarely exceeds 650 ft per min; fo 500 ft. RATIO OF CYL DIAM to length of stroke varies for geared hoists 1 : 1.67; ratios of 1 : 1.25 to 1 : 1.33 are common. For direct-acting ho usually from 1 : 1.3 to 1 : 2.66. FRICTION ALLOWANCE for engines, shea guides is generally lumped to cover all these items, including windage. Futers (4) assume starting friction of engine as 25% of wt of one cage + co running friction as 0.6 of this or 15%, and shaft friction and windage as 10 all these items are not effective until after starting, it is customary to take ance at 20% for direct-acting and 25% for geared hoists. Following frictio were measured at three Butte (Mont) shafts, when hoisting from depth max speeds of 3 000 to 3 500 ft per min (6, p 837).

	In balance	Out
Speculator shaft	21.1%	
High Ore shaft	23.0	
Diamond shaft	29.0	

These percentages are in terms of indicated work of the engines, the diffe values when hoisting in and out of balance showing that shaft friction at speed is much greater than engine friction. In these cases the engine fric probably less than 6% (6).

Effective crank radius is taken at 0.58 of full crank radius for engines 87.5% of stroke, which is about as late as is practicable. This enables c start the load, when cranks at 90° apart are in their most disadvantageous

Steam pressure. INITIAL PRESS is assumed for small hoists in at 80 to 100 lb. For simple, non-condensing hoists, assume 100 pound, 125 to 150 lb. (NOTE. — Hoist should be designed to start pressure, to take care of emergencies.) MEAN EFFECTIVE PRESS not affect size of cylinder necessary to start the load, but must when engine is up to speed with shortened cutoff. I indicator cards, approxima be calculated by:

$$p = 0.9\,[c\,(p_i + 14.7)$$

in which: p = m e p (gage steam press (gage); b = a press (for hoisting engines c = constant depending on off; 14.7 = atmos press at 0.9 is the diagram factor

Table 5. Values of "c" for Engines with 7% Clearance

Point of cutoff, % of stroke	Ratio of expansion	c
100	1.00	1.000
83.3	1.18	0.986
75.3	1.30	0.969
62.5	1.54	0.925
50.0	1.88	0.860
37.6	2.40	0.766
25.0	3.35	0.637

Force required to ac and moving parts can be di slow hoisting speeds, and in general for geared hoists, but must b for rapid hoisting.

Let M = mass of one cage and empty car, and W their weight;

ıass of one rope and W_r its weight; F_1 = accelerating
+ accelerating force in descending rope.
M_3), and $F_2 = aM$.
tion, load on ascending rope is $W + w + W_r + F_1$,
$- F_2$, assuming empty cage at surface. Unbalanced
glecting changing lengths of ropes, is:

$$) - (W - F_3) = w + W_r + a\,(2\,M + M_1 + M_3) \qquad (17)$$

ıte the masses of drums, sheaves, and reciprocating
ssumed as concentrated at its equivalent radius of
hen, if M_d is mass of drum, and neglecting sheaves
ting force required is $F_d = 0.7\,M_d a$, since surface of
city as the rope. Weight of drum is calculated, or
ıt of a similar one. A drum 8 ft diam by 8 ft long

on one piston, due to initial steam press,
n press, lb per sq in,
roke, ft,
inder, in,
iciency or factor, to allow for starting friction of engine,
nd cages, assumed at 0.7 to 0.83,
ım, ft,
on, sq in $= 0.7854\,d^2$,
load, lb,
oke to diam $= 12\,l \div d$,
unbalanced load about center of drum,
steam press on one piston about center of crank shaft,
dvantageous position of cranks.

$LD \div 2$, and $m_P = 0.58\,Pl \div 2$
equal to or greater than m_L, or $0.58\,ePl \div 2 = LD \div 2$,

$$d^2 = 2.19\,\frac{LD}{ep_i l} \qquad (18)$$

rms of d, $d^2 = 26.3\,\frac{LD}{ep_i k}$ (19)

ton rod. For a geared hoist, if q be the gear ratio,

$$d^2 = 26.3\,\frac{LD}{ep_i kq} \qquad (20)$$

mply large unless rapid acceleration and high speed are
acceleration, the engines develop greatest power just at

, ft per min; $p = m$ e p, lb per sq in; N = revs of drum
d load, lb, including wt of ore and rope, and total force
ges, ropes, and drums. Then the required h p $= L_T V \div$
ders $= 2\,eplAN_2 \div 33\,000$. Equating and reducing to

$$d^2 = \frac{L_T D}{epl} \qquad (21)$$

$$d^2 = 12\,\frac{L_T D}{epk} \qquad (22)$$

alanced load only, at uniform speed, the m e p for any
, and minimum point of cutoff found for constant speed,
or for running engine friction, shaft friction, and windage,
ssame value as in Eq 19, since increase in shaft resistance
ing engine friction to running engine friction.

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Table 6. Examples of Hoisting Engines (McCulloch and Futers)

	Case.....	I	II	III	IV	V and VI	VII	VIII
1	Type of engine...	Non-condensing, duplex cylinders. Cyl drums, no tail rope	Non-condensing, duplex cylinders. Conical drums	Non-condensing, duplex cylinders. Cyl drums, tail rope	Non-condensing compound. Cyl drums, no tail rope	Compound condensing. Cyl drums, no tail rope	Compound condensing. Conical drums	Compound condensing. Cyl drums, tailrope
2	Diam of drums.....	8 ft	6×14 ft	8 ft	8 ft	8 ft	6×14 ft	8 ft
3	Cylinders.....	21⅜″×48″	22″×48″	16¼″×48″	18⅜″ & 30¼″×48″	18⅜″×32¼″×48″	.....	.....
4	Distance during acceleration.....	21½ ft	.....	41½ ft	As in Case I	As in Case I	As in Case II	As in Case III
5	No revs during accel	⅞	1.13	1.66	"	"	"	"
6	Time of acceler'n, sec	1.87	3.1	3.6	"	"	"	"
7	Distance during retardation.....	200 ft	.....	43½ ft	"	"	"	"
8	No revs during retardation.....	8	2.1	1.74	"	"	"	"
9	Time of retard'n, sec	25	7.3	4.8	"	"	"	"
10	Max shaft velocity, ft per sec.....	18.5	24.5	17.34	"	"	"	"
11	Max h p, at full speed	365	190	170	"	"	"	"
12	Min h p at full speed	48	190	170	"	"	"	"
13	Steam consumption, acceleration period	36 lb	38 lb	20 lb	36 lb	36 lb	38 lb	20 lb
14	Steam consumption, full speed period...	278 "	252 "	200 "	218 "	143 "	126 "	123 "
15	Total steam.....	314 "	290 "	220 "	254 "	179 "	164 "	143 "
16	Steam per shaft hp-hr, entire trip.....	61.6 lb	56.8 lb	43 "	50 "	35 "	32 "	28 "
17	Safety factor of rope	10	10	10	10	10	10	10
18	Per cent of economy, comp'd with Case I	.....	8%	30%	19%	43%	48%	55%
19	Per cent extra rope stress due to accel..	43%	18%	25%	43%	43%	18%	25%
20	Starting crank radius	0.38 r	0.38 r	0.38 r	0.8 r	*0.8 r	*0.8 r	*0.8 r
21	Condensa'n loss fac-							

Valves			Corliss				Corliss	
Cutoff at start & at full [speed	0.85–0.25	.....	.....	.....	.....	.....	0.85–0.25	.....
Steam consumption, shaft hp-hr	.....	45 lb	.....	28 lb	.....	45 lb	39 lb per i h p hr	.....
No and type of drums	2 cyl	2 cyl	2 cyl	2 cyl	1 cyl	1 cylindro-con- [ical	2 cyl	1 cyl
Size of drums, dia × face	12′×5.5′	10′×6.5′	6′×3.5′	10′×5.5′	12′×6′	(a)	10′×5.5′	15′×15′
Keyed to shaft or loose	loose	loose	loose	1 keyed, 1 loose	keyed	keyed	1 keyed, 1 loose	keyed
Type of brake	post	post	post	post	2 post	post	post	2 band
Power for auxiliaries	steam	steam	steam	oil under press	steam	steam	(b)	steam
Unbalanced rope pull, lb	34 000	40 680	19 000	26 000	11 000–16 000	42 200	14 800	.....
Balanced or unbalanced	both	both	both	balanced	balanced	balanced	balanced	balanced
Aver hoisting speed	2 300	1 500	1 000	3 000	2 100	5 000	2 700	.....
Max hoisting speed	3 500	same	1 900	.....	3 000	6 000	3 000	.....
Depth of shaft	3 500	6 000	1 800	2 700	700	5 300	1 610	4 000
Inclination of shaft	90°	15° to 35°	90°	68°	90°	90°	47.5°	90°
Time for caging	8 sec	10 sec	.....	5 sec	.....	2 min	5 sec	.....
Size of rope, in	1.5	1.5	1⅛	(c)	2	1.5	1⅛	1.5
Skip or cage and weight, lb	skip 7 200 cage 3 400	12 cars, 1 150 ea	cage 2 630, 2 decks	skip 4 500	cage [2 decks, 7 750 [3 decks, 9 700	cage, 6 200 2 decks	skip, 5 500	.....
Weight of ore	10 000	26 880	3 000	13 440	(d)	12 000	12 200	.....
No of levels served	several	3 [hr	4	.....	.....	8	ore 3, men 6	.....
Tons hoisted and time	.....	1 100 gross, 9	400, 16 hr	.....	see Art 18	19 hr	1 650, 8 hr	.....
Time of compl hoist, min	ab 1 min fr 2 400	7	.....	1	.....	1	35 to 50 sec	.....
Weight of hoist, lb	520 000	540 000	153 000	550 000	248 200	1 122 000	250 000	.....
First cost, fob factory	.....	.....	$15 000	.....	$20 000	$75 000	$26 500, erected	.....

(a) 15.75′ and 25′ [Fig 14]. (b) Reverse and clutch, air; brake, gravity and air. (c) 1⅞″ flattened strand. (d) 2 cars, 8 000; 3 cars, 12 000 lb.

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Comparison of performance of different types of hoistin in Table 6. These engines are designed for max depth of 3 o 3 360 lb; skip, 2 240 lb; initial steam press, 150 lb; outpu balanced hoisting. (NOTE. — "Shaft horsepower-hour" denc of ore only × total aver depth in ft hoisted per min ÷ 33 000.)

Examples and costs of hoisting engines. An Eastern manu geared hoists, 1914: cylinders 14 × 18 in, single rigid drum 54 in dia motion reverse, slide valves, hand brake, $1 760; cylinders 12 by 15 drum, 48 in diam by 40 in face, link-motion reverse, hand brake, $1 figures from an Eastern factory, 1914.

Table 8. Geared Hoists, Single Friction Drum, Reversibl

Rated hp	Cylinders, in	Drum, in	Wire rope on drum in single coil	Aver hoisting speed, ft per min	Bed-plate, in	Weig hoist
		diam lgth	ft		width lgth	lb
10	5 " by 6	16 by 20	150	225	57.5 by 41.5	1 50
16	6.25 " 8	24 " 24	240	265	66.5 " 50	2 00
30	8 " 10	29 " 25	276	350	73.5 " 59.5	3 50
50	10 " 12	42 " 34	426	400	89.75 " 68.75	4 50
75	12 " 16	48 " 40	500	450	105.25 " 81.5	7 00
100	14 " 18	54 " 48	602	450	117 " 92.25	9 00
150	16 " 18	60 " 60	754	450	120 " 94	10 00

Note. — For double-drum hoists add 50% to weight and 45% to p

A Western manufacturer quoted for geared hoists, 1914: cylinder rigid drum, link-motion reverse, balanced slide valves, drum 48 in dia gears, post brakes, indicator, $1 450; 10 by 14-in engine, with 48 b brakes, and band friction clutch, $1 760; 12.25 × 14-in engine, drum face, with reverse, band brake, etc, $1 650. Table 8 gives data of large with approx costs.

9. Electric-driven Hoisting Engines

(See Sec 16, "Applications of Electric Transmission for M

10. Hoisting with Compressed Air (See also Sec

Compressed-air hoists have same general construction as s small-scale work standard geared hoists are used (Table 8). operation, large expansive-working engines require cylinders Cutoff is later, clearance volume is reduced to minimum, and ports are required, since at same pressure the density of ai steam. SMALL HOISTS, running intermittently, give no trou of moisture in exhaust ports, as cylinder walls and passages h normal temp. But, for economy, the air should be reheated. acceleration, air is admitted at practically full stroke, and pc throttle. IN LARGER HOISTS, with expansion gear, expansio because, since there can be no condenser, the terminal press for proper exhaust. Owing to rapid drop in temp of expand done is less than with steam; that is, the adiabatic curve of a steam, with corresponding decrease in mean effec press. Hen cutoff must be later (Sec 39). On the other hand, the theore expansion is never reached in practice, because of transm cylinder walls, compression in clearance spaces, and presence air. Loss in efficiency due to incomplete expansion can be red expans[illegible] in high-press cyl at 0.9 stroke (mi

ing between cylinders to initial temp, the loss in t one-half that of a simple cylinder, or a saving of entering high-press cyl.

l air. See Sec 39 for theory, and results of work with . Table 9 gives relations between initial and terminal nts of cutoff (9). Corrections are necessary for clearance off being found by dividing the sum of cutoff plus clear- earance. Thus, if stroke is 5, with cutoff at 0.2 and clear- of cylinder plus clearance is $(5 \times 0.05) + 5 = 5.25$; the $+ 0.25 = 1.25$, and actual cutoff is $1.25 \div 5.25 = 0.23$. volume of air at cutoff is increased by the clearance, the culated on basis of nominal cutoff.

Ratios of Pressures and Temperatures Due to Compressed Air in a Motor Cylinder

of to abs for re ke	Ratio of mean to total abs press, during expansion only	Ratio of initial to final temp	Ratio of initial to final abs temp, due to expansion only	Ratio of initial to final abs press for ratio of expansion
19	0.166	0.513	0.513	0.039
48	0.233	0.460	0.578	0.069
36	0.295	0.518	0.627	0.104
15	0.353	0.568	0.669	0.142
85	0.408	0.612	0.705	0.184
47	0.460	0.652	0.737	0.228
06	0.510	0.688	0.767	0.275
57	0.558	0.722	0.794	0.325
02	0.604	0.754	0.818	0.378
42	0.649	0.784	0.841	0.433
77	0.692	0.812	0.862	0.487
07	0.734	0.839	0.882	0.545
32	0.774	0.865	0.902	0.605
54	0.814	0.889	0.920	0.667

Due to Clearance, for Nominal Cutoffs in Column 1

	Percentage of clearance				
	0.05	0.06	0.07	0.08	0.10
	0.143	0.151	0.159	0.167	0.182
	0.191	0.198	0.206	0.213	0.227
	0.238	0.245	0.252	0.259	0.273
	0.286	0.293	0.299	0.305	0.318
	0.333	0.340	0.346	0.352	0.364
	0.380	0.387	0.392	0.398	0.409
	0.429	0.434	0.439	0.444	0.455
	0.477	0.481	0.486	0.490	0.500
	0.524	0.528	0.533	0.537	0.546
	0.571	0.576	0.580	0.585	0.591
	0.619	0.623	0.626	0.630	0.637
	0.667	0.670	0.673	0.676	0.682
	0.714	0.717	0.720	0.722	0.727
	0.762	0.764	0.766	0.768	0.772

d from G. D. Hiscock, "Compressed Air, Its Production, Also, see Peele's "Compressed Air Plant," Chap 17.)

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Cylinder volume. Work per stroke is calculated from ft-lb of wor revolutions of engine. This, with initial and final pressures is subst for partial or for complete expansion (Sec 39), which is solved for initi pressed air per stroke = theoretical cylinder volume. This is correc according to type of engine, and the proper ratio of stroke to diam is de volume of air ÷ volume at end of expansion = per cent of cutoff.

Volume of compressed air required = volume of air per st above) × number of strokes, remembering that there are 4 str tion in a duplex hoist. Volume so found is reduced to free air (Se gives volume of free air per min per indicated hp, for different reheating, and at 60° initial temp. These values do not incl corresponding to piston clearance (see above).

Table 11. Cubic Feet of Free Air per Minute per Engine I H P

Point of cutoff	Gage pressures, lb							
	30	40	50	60	70	80	90	100
1	23.3	21.3	20.2	19.4	18.8	18.42	18.10	17.8
¾	18.7	17.1	16.1	15.47	15.0	14.6	14.35	14.15
⅔	17.85	16.2	15.2	14.5	14.2	13.75	13.47	13.28
½	16.4	14.5	13.5	12.8	12.3	11.93	11.7	11.48
⅓	17.5	15.2	12.9	11.85	11.26	10.8	10.5	10.21
¼	20.6	15.6	13.4	13.3	11.4	10.72	10.31	10.0

Quantities of free air in Table 12 are based on actual runnin hoists. As such engines seldom run more than ¼ to ½ the tin capacity of ¼ to ½ the volume given is sufficient, provided the age capacity, and the time per trip is short.

Table 12. Volume of Free Air for Duplex Hoists (@ 6

Diam of cyl, in	Stroke, in	Rev per min	Normal hp	Actual hp	Wt lift singl rope,
5	6	200	6	11.8	1 000
5	8	160	8	12.6	1 650
6.25	8	160	12	19.8	2 500
7	10	125	20	24.2	3 500
8.25	10	125	30	33.6	6 000
8.50	12	110	40	37.8	8 000
10	12	110	50	52.4	10 000
12.25	15	100	75	89.2	.....
14	18	90	100	125.0	.....

Reheating (theory and results given in Sec 39 and 15) reduc required proportionally to the ratio $T_2 \div T_3$, where T_2 = absolu and T_3 = absolute temp to which the air is reheated. Or, the in is expressed by:

$$T_2 : T_3 = 1 : V_3$$

V_3 being the volume at temp T_3. The added volume is obtain the cost of producing the same volume in the compressor itself

Practicable increase of temp is up to 400° F; higher temp lubrication difficult and loss in heat between heater and cylinde sive. Reheating to 400°, and allowing a transmission drop to 30 oretical gain of 40% in practice, say 20 to 30%. Using dry

per added hp-hr due to reheating is 0.2 lb coke. Corliss engine, with air at 95.5 lb abs, reheated to nption of 0.176 lb per added hp-hr. At Anaconda nples below), air is reheated by steam at 200 lb press ni, Ariz, the 75° F. For 20% is gen-

nodifying the mine, and in or using com- utilize energy etardation to rage system, ting. This is d of hoisting. Anaconda air functions (6, le is opened of stroke and troke, giving ng 1 to 3 rev ing off the air ases (card *b*). d the point of anced to such ce air to be s. As speed hortened by card of series d to atmos- tically work- naintain this in the card s press) free all points of . This stage

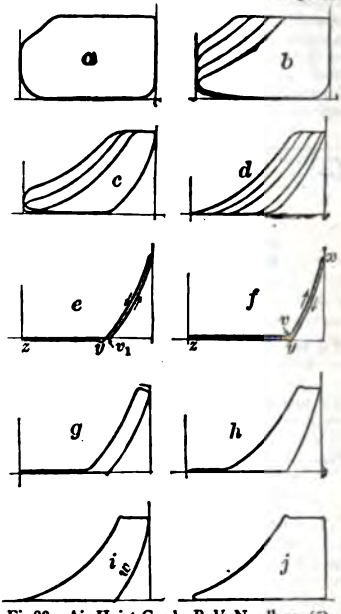
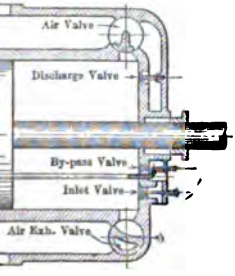


Fig 20. Air Hoist Cards, B. V. Nordberg (6)

e, the last of the group, air is cut off practically on center, which is then maintained to end of stroke. Also, atmos troke to the point where recompression begins. This card corresponds to work of ± 0. At this point maximum speed is attained, and the hoist runs by momentum of the moving masses.



Cylinder, B. V. Nordberg (6)

To retard without applying brakes, the regulating lever is moved to retarding position, causing admission valve to remain closed during forward and return strokes, as in card *e*. Exhaust valve now opens, air is ejected during return stroke, and the process of producing cards *e* and *f* is

er controls point of closure of exhaust valves during the e shaded area of *f*, which represents negative work. By from its neutral position, cards *g*, *h* and *i* are made succes-

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sively, i representing compression of a full cylinder of air. The effect of t
exhaust all air not to be compressed, and then compress the remainder. By
lever to its extreme position, an auxiliary valve opens communication betwee
cylinder, while the piston passes its center with both exhaust valves closed
pressed air then flows to other side, which is filled with free air, causing increa
pounds press in front of piston, the disappearance of the re-expansion line zr i
a much higher mean effec press of compression, which stops the hoist. Card
at this time. Fig 21 is a diagrammatic section of the cylinder and valves.
inlet valves prevent air in cylinder from expanding below atmos press. Disc
operate automatically when air is compressed back into the system; by-pas
opened positively and closed automatically. Fig 22 shows the operation

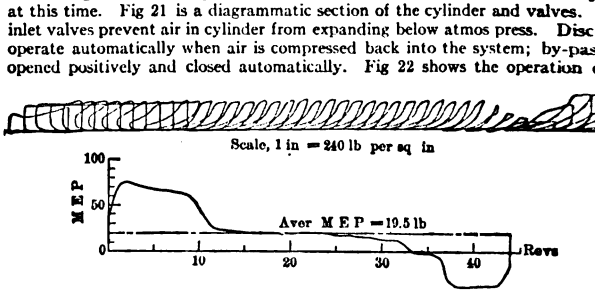

Fig 22. Hoisting Card (Leonard Shaft, Butte) from Head End of 34 × 72 Ai
Drum, Running Balanced. Hoisting Speed, 1 800 ft in 45 sec (B. V. Nor

For first 5 strokes, cylinders are completely filled; next 5 strokes are made wi
shortened cutoff, after which the clearance compression gear comes into act
is admitted during last 6 strokes, air being compressed back into the sys
represented by this compressed air is shown by the area below atmospheric

Examples of Compressed-air Hoists

I. Anaconda Copper Co, Mont (6). Main hoists and several auxiliaries
by air at 90 lb. Cylinders of main hoists are 34 by 72 in, with Nordberg-
gear of special design. When lowering a load, or "plugging" instead of br
retardation period, air is compressed in cylinders and returned to the system
paragraphs). Load of ore, 5 to 6 tons; rope, 4 000 to 12 000 lb; combinatio
cage, 11 000 lb aver. Hoisting is usually in balance. Maximum speed, 4 50
average, 2 000 to 2 300 ft. Air is reheated indirectly by steam to 250 to 3

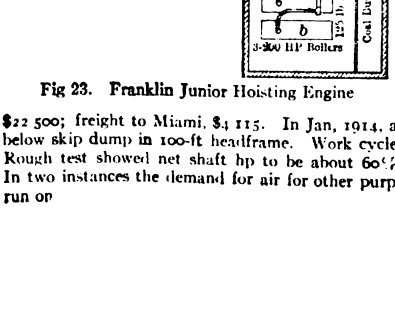

Fig 23. Franklin Junior Hoisting Engine

made in 1911, air rehe
gave net shaft hp as 3
trical hp supplied to m
the compressors. Mec

II. Miami Copper
Nordberg 20 by 48-in
Corliss valves. Drums
by 55 in face, holding
1⅛-in rope; one drum k
the other driven by fri
speed, 72 rev per min.
is reheated to 370° F; c
about 2 275 cu ft free
for depth of 675 ft.
750 ft per min. Capac
per hr from 540 ft.
250 000 lb; cost, f o b
$22 500; freight to Miami, $4 115. In Jan, 1914, all ore came from 420-ft
below skip dump in 100-ft headframe. Work cycle, 94 sec, of which engine
Rough test showed net shaft hp to be about 60% of the indicated hp of
In two instances the demand for air for other purposes was so great that t
run on

ne, Mich. Combined air and steam hoist (Fig 23). The unbalanced, and compresses air back into a system of steam r cylinders *g*, 36 in diam, are tandem to steam cylinders *h*, th Corliss valves. Drum, 15 ft diam by 15.5 ft long, holding t dips 47°. Of the 3 boilers *b*, 200 hp each, only 2 are used. ks *c* are each 10 by 32 ft. When hoisting, the ends of air cyls ompressed. When lowering, exhaust valves of steam cyls are in air cyls is discharged into the receivers. Reducing valve 5 lb. Skip is lowered with receivers full of steam at 75 lb, the press to 95 lb, which is then available to start hoisting. valve *i* opens to admit steam to complete the hoist.

Bisbee, Ariz. Five direct-acting hoists and 2 geared hoists ir. The larger direct-acting hoists require 1 400 to 1 600 cu quivalent to a consumption of 50 to 55 lb of steam at com-

and Gasolene Hoisting Engines

igh freight rates make coal expensive, where there nerating steam, for isolated mines, or for temporary re not so well suited for hoisting as steam or air hoists e of time required to start from rest, the engine must n being operated by friction clutch. Hence, in inter- is low. Since the engine can not readily be reversed, ary; but, by special arrangement of clutches and gears, drum can be reversed and hoisting done in balance.

Producer or city gas, gasolene, distillate, kerosene, or Producer gas, or waste gas from coke ovens, furnishes ts require 0.10 to 0.15 gal gasolene per hp-hr; with to 0.85 lb per hp-hr.

ginia mine, Idaho City, Colo, hoist was driven by two e engines (11). 50 gal gasolene @ 20¢ did same work as nder boilers; a saving of $2.50 per day, aside from fireman's

mon form for small hoists is a friction drum, geared to a ingle-cylinder engine; made in standard sizes up to 25 to

e Gasolene Hoist, Single Drum (Western make)*

2 and 15-hp engines have 2 cyls)

n	Ft of rope	Hoisting speed, ft per min	Rope pull, lb	Approx wt, lb	Approx price
	900	50	800	1 175	$180
	900	100	900	1 250	225
	900	100 150	1 500 1 000	1 475	290
	900	100 150	2 250 1 500	2 700	400
	900	100 150	3 000 2 000	3 000	460
	900	100 150	3 500 2 300	3 600	565
	900	100 150 200	4 300 2 900 2 000	3 800	635

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30 hp. Engine runs at constant speed, and is controlled by a governor, admission of gas at every alternate revolution, or shuts off gas entirely as sp larger capacity, 4-cycle engines with 2 to 8 cylinders are employed (12). T cal and similar to the ordinary automobile and marine type. Since rot high, double-reduction gearing to drum shaft is desirable. Speed may be v tling the gas, and advancing or retarding the spark. Chief advantages of m engine are: variable speed, thus allowing slowing down between hoists an up the load; power is more uniform throughout each revolution; large fly nated; engine is more easily started from rest; self-starters, as for autor may be applied; less danger of stopping while hoisting.

Double-drum gasolene engines for hoisting in balance have sp and gears. In one form, crank shaft is at right angles to drum sha it by diametrically opposite bevel pinions, meshing with a gear or Each pinion is loose on shaft, and is driven by friction clutch.

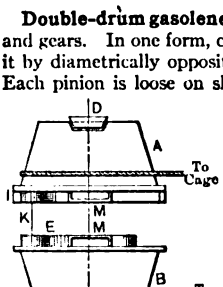

Fig 24. Arrangement of Clutches, Double-drum Hoist

locking device only one clutch at a thrown in gear; this rotates drum shaft tion, and throwing in the other clutch opposite direction. Drums are keyed controlled by clutch.

Fig 24 shows diagrammatically an arrange ing in balance with 2 drums, the engine ru ously in one direction (13). Each rope wi drum, the drums being thrown in alternate D and D_1. Internal gear I, driven throug drum A, is connected by pinion shaft K with which is thrown in or out of gear with drum With drums at rest to hoist cage A and low is thrown in and D_1 out, while both clut gear. Drum B will then rotate in opposi drum A. Clutches M may be dispensed necessary to change relative positions of ca is used to pick up load while engine is runni

Rating and capacity. Rating is us of indicated hp. Lifting capacity and h also depend on quality of engine and ca nance. Let L = load or lifting capacity; HP = rated hp of engine efficiency of hoist; S = hoisting speed, ft per min; then,

$$L = 33\,000\,eHP \div S$$

Up to 10 hp, e may be assumed as 0.5 to 0.6; for larger engines, 0.6

At Blackwood Colliery, L. V. Coal Co, Penn, a gasolene hoist hauls c slope, at 40°. Wt of car and coal is approx 4 000 lb, rope speed, 200 ft per loose on shaft, and driven by band friction clutch. Engine has 4 cylind with gears for reversing direction of drum rotation.

12. Hoisting from Deep Shafts

Deep hoisting is taken to mean hoisting from depths greater tha Because of the time required to hoist from great depths, and inc of rope weight to net load, both speed and load must be increased given tonnage. Great length of rope requires large drums, the mas of which make special problems in the design. The highest grade are requisite. For depths to say 4 000 ft, cylindrical drums with form section are suitable, but better economy is obtainable with ho having special features for deep winding. In each case the resul adjustment as to operating expense, efficiency, and capital outlay,

sed for deep hoisting are: (*a*) HOISTING IN A SINGLE l ropes of uniform section or tapered; (*b*) WHITING KOEPE SYSTEM WITH TAIL ROPE; (*d*) HOISTING IN h rope, one suspended at half the depth and one at aded and dumped simultaneously, lower skip dis- h the upper skip is filled; (*e*) HOISTING IN TWO OR r stage hoist either on surface or underground.

t. For depths of 4 000 to 4 500 ft, conical drums may be used without exceeding practical limits for 000 ft taper ropes become necessary, with conical m section rope, the slope of drum face for equality angerously steep. Also, the weight and moment of ery great. For hoisting 50 tons per hr from 5 000 ft, 42) calculate an engine of following dimensions: small m, 24 ft; cyls, 38 by 48 in; max speed, 60 ft per sec; g skip, 30 sec; net load, 6 100 lb; taper rope in 4 sec- 375 in diam, the other 3 of 1 400 ft each being 1.0625, pectively; total weight of one rope 11 600 lb, with rm-section rope were used, diam would be 1.4375 in; im pressure is assumed at 150 lb. For same output, 4 times that required for a depth of 3 000 ft. Hans sting engines more than 3 times as large are required om a depth of 6 000 ft as from 3 000 ft.

d rope for hoisting in a single lift from vertical depth of C. W. Comstock (15), as follows: wt of ore, 15 000 lb; h wires of tensile strength of 270 000 lb per sq in, 1.13 in t upper end; wt, 41 240 lb. ROPE has a varying section, t all points, and is calculated by formula

$$d = d_0 e^{\left(\frac{\delta x}{f}\right)} \tag{25}$$

from lower end; δ = wt per cubic unit of rope; f = unit e, or total area of the circumscribed circle; e = base of d. Weight of a length x of this rope is:

$$W = (Q + q)\left(e^{\left(\frac{\delta x}{f}\right)} - 1\right) \tag{26}$$

wt of skip. The smallest diam of drum is taken as 1 500 s (0.084 in), or 10.5 ft. Allowing for angular position of on, acceleration, and bending stress, the total unit stress equivalent to net safety factor of 2.88. DRUM is designed stant, when winding a rope of constant unit stress. Equa- surface is:

$$\frac{{}^{n}+1] - 2A^{2n}\left\{A^{y_n} - 2\,(y_n - 1)\,A^{3(\quad+1)}\right.}{{}^{n}+1] - 2A^{2n}\left\{A^{y_n} - (y_n - 1)(A^{2(\quad+1)} + 1)\right.} \tag{27}$$

bove or below mean radius; if below, n is considered nega- n radius of drum, in; δ = 288 lb per cu ft for steel rope with in; f = unit stress of nominal cross-section of rope, or cle circumscribing rope; y_n = ratio $\rho_n \div R$, in which ρ_n is or below mean radius. Mean radius is radius of coil or passing point of skips in the shaft; $y_{(n+1)}$ = ratio of radius to the mean radius.

landing and sheave is neglected:

$$R = \rho_{-n}\frac{Q + 2q + p_{-N}}{Q + 2q} \tag{28}$$

end of drum, and p_{-N} = weight of one rope equal to depth

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Since R is the arithmetical mean of any two radii, at same number of rev
and below R, then, $2R = \rho_N + \rho_{-N}$

in which: ρ_N is the radius of large end of drum and ρ_{-N} is fixed by assum
diam of large end of rope to diam of the small end of drum on which it wind
from Eq 28, and ρ_{-N} from Eq 26. With R and ρ_{-N} known, ρ_N is found fr
calculating the intermediate radii from Eq 27, start at mean radius R, as
to simplify calculations. For first revolution, $\rho_n = R$ and $y_n = 1$; for
tion the value of $y_{(n+1)}$ found for the first revolution becomes the value of
Only half the number of radii need to be calculated, because those on the o
R may be found from Eq 29, which may be written, $2R = \rho_n + \rho_{-n}$.

Total number of revolutions for one trip is the same as for a cylindrical drum
or if N is the total number, then, $N = H + 2\pi R$

in which H = total depth of shaft.

Detailed design is required to determine exact width of drum. In this c
(15) calculates drums 14.14 ft diam at small end, 49.5 ft diam at large en
diam of 31.82 ft. Width of each drum is approx 24 ft, requiring that th
behind the other, and connected by vertical walking beams or operated
cranks. An average speed of 3 000 ft per min is assumed; also an acceleratio
ing skip of 16 ft per sec, corresponding to 4.5 ft per sec acceleration for loaded
starting moment to be overcome by engines is calculated at 3 324 300 ft-lb
unbalanced load times mean radius, or 240 000 ft-lb; moment due to accele
which consists of 2 skips, 1 rope, and ore, or 176 200 ft-lb; and moment du
tion of 2 drums and 1 rope on one drum, or 2 908 100 ft-lb. Each drum
weigh 193 000 lb. Compound cylinders are calculated: hp cyl, 32 by 96 in
96 in, a pair of cylinders being at each end of drum shaft. Initial steam pre
ing, 150 lb. Acceleration requires 5 sec, the engine being worked compou
mainder of hoist. Total time required per trip, from start to stop, is 4
made up of 5 sec acceleration, in 1.25 rev; 63 sec retardation, 16 rev; 205.
speed, or 102.75 rev at 30 rev per min. Allowing the remainder of a 5-r
25.5 sec, for loading and dumping, 12 trips per hr are made, or 12 × 7.5 to
Form of drum, as calculated above for a taper rope, approaches a true con
closely that a cone may be used without seriously affecting conditions.

Whiting system with tail rope (Art 6) requires rope of uniform s
is not applicable when the depth is such that the useful strength of th
small for a load required to give the output. With rope of high tensile
system has advantages to a depth of say 6 000 ft, especially if hoistin
bottom only. A tail rope (Art 6) is necessary, giving constant load
the hoist. This has the advantage of lower first cost; the engines are
less energy is required to accelerate the comparatively light driving

Koepe system with tail rope (Art 6) has same advantages as t
as to low first cost and elimination of fleet of the rope. The large s
eter requires larger engines to overcome the static moment on start
Koepe sheaves are better adapted to constant-torque electric ho
steam hoists. With large cylinders and relatively low rotative speed
of large drums is an advantage in steadying the engine impulses.

Tandem system (Fig 25) consists in having 2 skips in the same com
upper skip with a rope equal to half the depth of shaft, the lower with its
to bottom of upper skip. Upper skip receives its load from pockets at m
shaft, into which the lower skip dumps. Both are loaded and dumped si
The upper rope carries its own weight and that of both skips. Both ropes m
The drums need be only large enough to hold rope equal to half the total
equation for the ratio of end diams of conical drums (Art 6) must be modifi

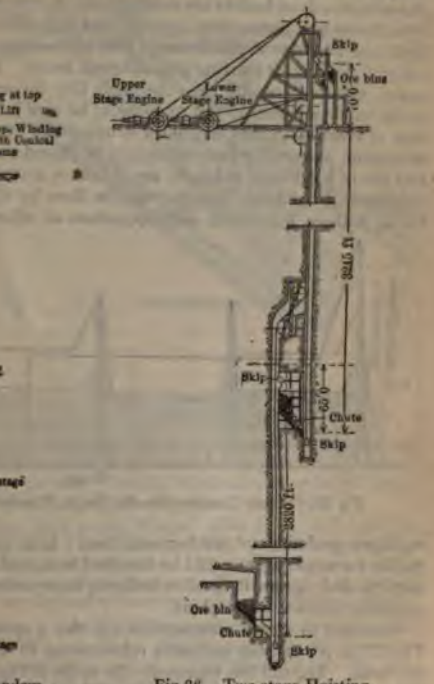
Let W_r = wt of whole rope; W_l = wt of lower section; w = wt of ore
and W = wt of both skips; then, for equal static moments at beginning and

$$r(W_r + W + w) - R(W + W_l) = R(w + W + W_l) - r(W + W$$

whence $$R = \frac{r(w + 2W_r + 2W)}{w + 2W_l + 2W}$$

smaller than for hoisting with a single lift, and moment of . There is an economy in steam consumption of about , and the hoist is smaller, with consequent lower first of extra loading pockets and shaft arrangements.

3) requires 2 engines, one hoisting from the bottom intermediate station, from which the other engine



ndem

Fig 26. Two-stage Hoisting

ngines may be duplicates, by which first cost is nical drums this condition may make it necessary noments. As the rope for lower lift is larger, this e drums are to be kept of same width. Taper ropes es. The two lifts must overlap by about 65 ft. am economy than the single lift, by about 10.5%, ed cost of shaft construction and extra attendance ation.

oist would be necessary at the intermediate station, ion. This arrangement would effect some saving eration, also in cost of rope.

two-stage systems compared. SINGLE LIFT. ibility, absence of attendance at intermediate t of this station. Disadvantages: high first cost,

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on account of large engines and drums; excessive ratio of end dian of whipping of the ropes; increase in cost of headframe; engin tionately less economical when handling men, tools, and supp SYSTEM. Advantages: smaller size of engines and very much l end diameters of drum, thus reducing first cost; engines more e whipping of ropes; may be used for greater depths than sing steam demands on boilers are more equally distributed than in sin Disadvantages: extra first cost of, and attendance at, interme high cost and size of headframe; lack of flexibility, so that the suited to sinking, or hoisting from different levels. TWO-STAGE SYS tages: greater flexibility than the single lift; favorable drum prop to the tandem system); tail ropes could be used in both upper and rope of lower stage may be guided in upper half of shaft and dang overcome; sheaves for lower stage may be placed at the ground leve and size of headframe reduced; use of brakes is attended with less is suitable for sinking; hoisting may be done by either stage from levels, without interfering with operations of other stage; first c

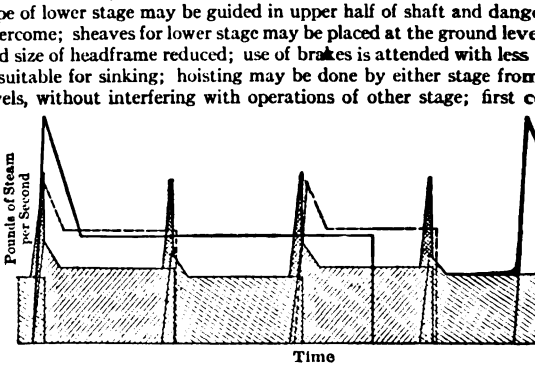


Fig 27. Steam Consumption for Single, Tandem and Stage Syste

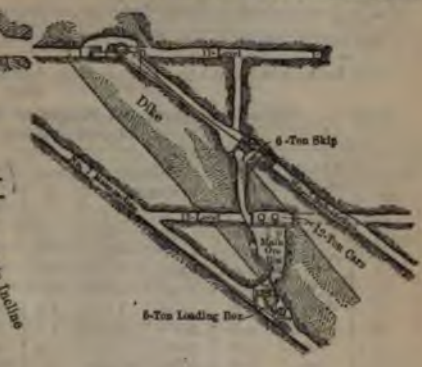
engines is probably not much greater than 1 large engine, as for sing engine for upper stage could be installed first, and second engine i until needed; steam demand on boilers is more uniform than in the requiring smaller boiler plant.

Variation in steam consumption for the 3 systems is shown Fig 27 (4, p 157), the ordinates representing lb of steam per sec the time. The full line represents the single-lift system, dotted lin the shaded diagrams the two-stage, all for depth of 6 000 ft. In diagrams the load and time of hoisting are made equal, so that the stage occur in rotation. Points of maximum demand for two-stage s to overlapping, not to acceleration. If period of rest were increase would not occur.

Examples of Deep Hoisting

I. Three-stage hoisting is in use at Crown Mines, Ltd, Transvaal (1 vertical shaft. Previous experience on the Rand proved the inefficiency of shafts, if hoisting is in 1 stage. Straight vertical shafts with crosscuts to the methods of transportation installed, but it was found advisable to inclined portion of shaft. The vertical portion, 1 085 ft deep, was deepened room for storage pockets between inclined shaft and the loading pockets o Each stage has 4 hoisting compartments. The second stage, 2 004 ft lo part of the incline, its hoist being near the intersection. Fig 28 shows g ment at bottom of vertical shaft. Below 13th level, at foot of second sta stage, with its hoist in the level above. Mode of transferring ore from stage is shown in Fig 29. These 2 shafts replace 11 old shafts formerly i

net, Mich, is equipped with Whiting hoists. Shaft is o driving sheaves are 19 ft diam, driven by a pair of condensing engines. High-pressure and intermediate crosshead, and l-p cylinder on another. The two cross- rods to opposite ends of a triangular walking beam, ides. From third point of the beam, a connecting rod



n of Fig 29. Transfer from Third Stage to Second Stage (16)

ave. Cranks are 90° apart, connected by parallel rods inders are 20.5, 31.75, and 50, by 72-in stroke. Steam ev per min. Use of a tail rope greatly reduced swaying Main rope, 1.75 in, tail rope, 1.375 in diam.

ne, Mich, is 4 935 ft deep. Hoist has 4 simple, non-con- by 60 in. (See Table 8, No VI.) Cylinders are set at 45° ch end have center lines at right angles and are connected hen one cylinder is on center, the others act at 45°, 90° and starting and uniform torque are thus obtained (Fig 17).

ine, is 4 990 ft deep, equipped with a duplex, horizontal, inders, 42 by 84 in. Drum is a steep double cone, large in. Central part of drum is cylindrical for a few turns is and not on conical surface.

ROPES AND SHEAVES

Vegetable Fiber Ropes

ngth, are: manila hemp, or fiber from *abaca* plant l aloe), Baden split hemp, Russian hemp, Italian and Cotton. Many grades of each material are ength, length of fiber, and care in cleaning. Price ranged from 6 to 16¢ per lb.

made of 3, sometimes 4 or more, strands. Each strand fibers; which, in turn may be individual fibers, as for a om a number of fibers. Original fibers rarely average gth of the rope depends upon the binding action of the friction between them. For same diam, 4-strand rope strength than 3-strand.

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culty of constructing the rope and attaching fastenings so that each its proportionate load. In standard round rope each wire at some an angle of about 28° with the rope's axis, so that the useful streng

Table 17. Hoisting Ropes with 6 Strands of 19 Wir

Diam, in	Wt per ft, lb	Iron			Cast steel			Extra cast steel			Plow steel		
		Approx breaking strength, tons	Minimum diam of sheave, ft	Cost per ft, ¢	Approx breaking strength, tons	Minimum diam of sheave, ft	Cost per ft, ¢	Approx breaking strength, tons	Minimum diam of sheave, ft	Cost per ft, ¢	Approx breaking strength, tons	Minimum diam of sheave, ft	Cost per ft, ¢
3/8	0.22	2.4	2.2	3.4	4.8	1.5	4.2	5.3	1.5	4.8	5.85	1.5	5 5
1/2	0.39	3.9	3.0	4.1	8.4	2.0	4.8	9.2	2.0	5.5	10.0	2.0	6 2
9/16	0.50	4.7	3.5	4.9	10.0	2.2	5.3	11.2	2.2	6.2	12.3	2.2	7.0
5/8	0.62	6.0	4.0	5.8	12.5	2.5	6.1	14.0	2.5	7.2	15.5	2.5	8.3
3/4	0.89	8.5	4.5	7.8	17.5	3.0	8.3	20.2	3.0	9.7	23.0	3.0	11.4
7/8	1.20	11.8	5.5	9.8	23.0	3.5	10.5	26.0	3.5	12.7	29.0	3.5	14 9
1	1.58	14.5	6.0	12.7	30.0	4.0	13.7	34.0	4.0	16.3	38.0	4.0	18 9
1 1/8	2.00	18.6	7.0	16.1	38.0	4.5	16.7	43.0	4.5	20.3	47.0	4.5	23.8
1 1/4	2.45	22.8	7.5	19.5	47.0	5.0	20.2	53.0	5.0	24.6	58.0	5.0	28.6
1 3/8	3.00	28.0	8 5	23.9	56.0	5.5	24.6	64.0	5.5	30.0	72.0	5.5	34.8
1 1/2	3.55	33.0	9.0	27.8	64.0	6.0	29.1	73.0	6.0	35.2	82.0	6.0	41 0
1 5/8	4.15	38.0	10.0	31.7	72.0	6.5	33.8	83.0	6.5	41.4	94.0	6.5	47.5
1 3/4	4.85	44 0	11.0	39 0	85 0	7 0	39.5	99.0	7.0	48 3	112.0	7.0	57.0
2	6.30	55.0	12.0	45.7	106.0	8.0	50.9	123.0	8.0	58.8	140.0	8.0	69.3

* Different manufacturers have different trade names for this qualit

Note. — Minimum sheave diameters are those recommended by manufac sizes should be used. Costs given are average of several makers for 1913

Table 18. Haulage Ropes with 6 Strands of 7 Wire

Diam, in	Wt per ft, lb	Iron			Cast steel			Extra cast steel			Plow steel		
		Approx breaking strength, tons	Minimum diam of sheave, ft	Cost per ft, ¢, 1913	Approx breaking strength, tons	Minimum diam of sheave, ft	Cost per ft, ¢, 1913	Approx breaking strength, tons	Minimum diam of sheave, ft	Cost per ft, ¢, 1913	Approx breaking strength, tons	Minimum diam of sheave, ft	Cost per ft, ¢, 1913
3/8	0.22	2.2	4 0	2.2	4.6	2.7	2.4	5.2	2.7	2.6	5.9	2.7	3 0
1/2	0.39	3 7	5.5	3.2	7.7	3.5	3.5	8.8	3 5	4 2	10.0	3 5	5 0
9/16	0 50	4.8	6.0	4 0	10.0	4.0	4.4	11.0	4.0	5.3	12.0	4 0	6 4
5/8	0 62	6.0	7.0	4.9	13 0	4.5	5.3	14.5	4 5	6.4	16 0	4 5	7 7
3/4	0.89	8.8	7.5	6.8	18.6	5.0	7.5	21.0	5.0	8.8	23 0	5 0	10 7
7/8	1.20	12.0	9.0	9.0	24.0	6.0	9.9	28.0	6.0	11.8	31 0	6 0	14 0
1	1.58	15.0	10 5	11.7	31.0	7.0	12.7	35.0	7.0	15.4	38.0	7 0	18 0
1 1/8	2.00	19.0	12.0	14.6	37.0	8.0	15.8	43.0	8.0	19.3	47.0	8.0	22 5
1 1/4	2.45	23.0	13.0	17.6	46.0	9.0	18.9	54.0	9.0	23 3	60 0	9.0	27 3
1 3/8	3.00	28 0	15 0	21.0	53 0	10 0	22.5	63 0	10 0	28 2	72.0	10.0	33.5
1 1/2	3.55	32.0	16 0	25.0	63 0	11.0	26.4	73 0	11.0	33.0	82.0	11.0	39.5

plied by *cos* 28°, or 0.883. Strength of the rope is regate strength of the wires. Tests made by U S strength of 6 × 7 rope is 90 to 92%, and of 6 × 19 strength of the wires. Epton (19), from a large of various types, found that a difference of 3 to 16% strength of wires and breaking strength of rope. after ropes had been used a short time to tighten makers revised their lists May 1, 1910, and con- breaking strengths. Breaking strengths given in based on actual tests of rope. Tables 17 and 18 aking strength, cost, and minimum diam of sheave ropes. Sheave diameters specified are based on a ending stresses are considered as part of working s' lists. Strength of 6 × 7 rope is from 10 to 14% o 5% less for the larger sizes, than standard 6 × 19 to 13% less strength than standard 6 × 19.

lat Rope. Weights and Strengths

Crucible steel		Plow steel	
x breaking stress, 2 000 lb)	Proper working load, tons (2 000 lb)	Approx breaking stress, tons (2 000 lb)	Proper working load, tons (2 000 lb)
27	5.4	33	6.6
36	7.2	43	8.6
41	8.2	49	9.8
50	10.0	60	12.0
54	10.8	65	13.0
63	12.6	76	15.2
68	13.6	81	16.2
77	15.4	92	18.4
81	16.2	97	19.4
45	9.0	54	10.8
52	10.4	63	12.6
60	12.0	72	14.4
69	13.8	82	16.4
83	16.6	99	19.8
90	18.0	108	21.6
98	19.6	118	23.6
05	21.0	126	25.2
28	25.6	153	30.6
68	13.6	79	15.8
79	15.8	92	18.4
91	18.2	105	21.0
02	20.4	119	23.8
14	22.8	132	26.4
25	25.0	145	29.0
48	29.6	171	34.2
70	34.0	197	39.4

WORKING LOAD is discussed in Art 8, including rope. Additional BENDING STRESS depends on m of individual rope wires, and is independent of rope should never exceed elastic limit of wires. ce must be made for unknown factors, as shaft

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friction, wind resistances at high speed, and lashing of rope. The
minate, and must be covered by safety factor. In deep shafts, ela
takes care of jerk due to slack rope, but in shallow shafts this stress
sidered. Sticking of cage in guides, or jamming of skip or cars in
stresses which can not be covered by any practicable safety facto

Compensation for slack rope by the use of springs is of littl
lowing tests show effect of slack: total length of rope below sheav

FIRST TEST. Empty cage, 4 000 lb
Lifted gently .....
" with 2.5 in slack .....
" " 6 in " .....
" " 12 in " .....

SECOND TEST. Cage and 4 empty cars, 6 375 lb
Lifted gently .....
" with 3 in slack .....
" " 6 in " .....
" " 12 in " .....

THIRD TEST. Cage and loaded cars, 11 300 lb
Lifted gently .....
" with 3 in slack .....
" " 6 in " .....
" " 9 in " .....

Bending stress is expressed by: $S = E_R (d \div D)$, in which $S =$
due to bending, lb per sq in; $E_R =$ modulus of elasticity of rope
largest wire, in; $D =$ diam of bend over sheave or drum on center l
If A be aggregate area of wires (sq in), and S_R bending stress in rop
any rope the total bending stress is:

$$S_R = E_R \div (dA \div D).$$

Table 20. Approximate Diameters and Areas of Wire in W

Diam of rope, in	6 × 9 hoisting rope			6 × 7 haulage
	Diam of wire = $\frac{1}{15.52}$ diam of rope, in	Area of 1 wire, sq in	Aggregate area of wires, sq in	Diam of wire = $\frac{1}{9}$ diam of rope, in
¾	0.0241	0.00045	0.0513	0.041
½	0.0320	0.00080	0.0912	0.055
9/16	0.0360	0.00102	0.1162	0.062
5/8	0.0404	0.00128	0.1456	0.069
¾	0.0480	0.00181	0.2063	0.083
⅞	0.0564	0.00250	0.2850	0.097
1	0.0644	0.00325	0.3705	0.111
1⅛	0.0720	0.00407	0.4640	0.125
1¼	0.0805	0.00509	0.5802	0.139
1⅜	0.0890	0.00622	0.7090	0.153
1½	0.0970	0.00739	0.8400	0.167
1⅝	0.1045	0.00857	0.9769	.....
1¾	0.1130	0.01003	1.1434	.....
2	0.1290	0.01307	1.4900	.....

Table 20 gives values of A for standard 6 × 7 and 6 × 19 rope.
and Wire Co give $E_R = 12\,000\,000$ maximum for new 6-strand
aut[1] 35% of the modulus of elasticity of steel wi

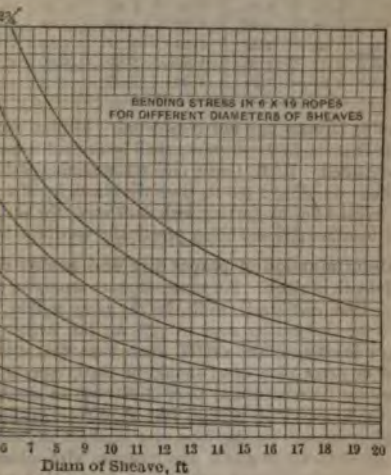
(20), 11 180 000 is a good average of E_R for the in use long enough for the strands to bed into the odulus for old rope. It is inaccurate to use the ires, because of their spiral twist in the rope, which ng stresses. In Eq 32, $A = 42 \times 0.25\,\pi d^2$, and if $d_R \div 9$ for 6×7 rope. Substituting, and taking c, $S_R = 542\,880\,(d_R^2 \div D)$.

$0.25\,\pi d^2$, $d = d_R \div 15.50$ and $S_R = 288\,000\,(d_R^2 \div D)$.

$0.25\,\pi d^2$, $d = d_R \div 21.7$ and $S_R = 204\,000\,(d_R^2 \div D)$.

$500\,(d_R^2 \div D)$.

el and Wire Co, 1913) shows bending stresses in 6×19 ula.



ending Stresses in Hoisting Ropes

retardation due to load and rope must be added to pre- ge, or skip, and contents, lb; w = wt per ft of rope, lb; t; a = acceleration, ft per sec per sec; P = total stress due to acceleration, lb,

$$= S_R + W + wl + F \tag{33}$$

$$= S_R + W + wl + a\left(\frac{W + wl}{2g}\right) \tag{34}$$

$\overline{C}$, where C represents rate of increase in the load due to d cage and rope as 1. Table 21 gives values of C from per sec, distance traveled due to acceleration, and total by weight of cage and contents plus rope, gives total g. An acceleration of 12 ft per sec per sec is rarely ex- ore common.

on condition of service. If all determinate stresses 5 to 4 is ample for hoisting ore and 6 for men. ires a safety factor of 5 for ropes hoisting coal and nd materials exclusively; in which, however, the

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Table 21. Stresses of Acceleration and Retardati

C $\frac{a}{32}$	a Ft per sec^2	S Ft per sec	$C+1$ total stress factor	Safety factor 10 for quiet load	C	a Ft per sec^2	S Ft per sec
0.	0.	0.	1.00	10.00	1.00	8.02	4.01
0.10	2.54	1.27	1.10	9.09	1.25	8.97	4.48
0.20	3.59	1.79	1.20	8.34	1.50	9.82	4.91
0.25	4.01	2.01	1.25	8.00	1.75	10.61	5.31
0.30	4.39	2.20	1.30	7.70	2.00	11.34	5.67
0.40	5.07	2.54	1.40	7.15	2.50	12.68	6.34
0.50	5.67	2.84	1.50	6.67	3.00	13.89	6.94
0.60	6.21	3.11	1.60	6.25	3.50	15.00	7.50
0.70	6.71	3.36	1.70	5.88	4.00	16.04	8.02
0.75	6.94	3.47	1.75	5.72	4.50	17.01	8.50
0.80	7.17	3.58	1.80	5.66	5.00	17.93	8.96
0.90	7.61	3.81	1.90	5.27	....	.....	.....

bending stresses are not considered. Mining regulations of the quire that the factor of safety shall be at least 6, bending stresses a disregarded. R. B. Greer (16) recommends a sliding scale facto depths exceeding 2 500 ft, since the elasticity of a long rope comp for stresses due to starting with slack rope, the actual weight of cag and bending stresses being practically constant for all depths.

Determination of proper size and quality depends upon th and depth of shaft. For same ultimate strength, plow-steel rope c more than cast-steel; while, for the same weight, plow steel is st 10% for the smaller sizes to 35% for the larger. Hence, it is e heavy loads and deep shafts, where rope weight is a large part of steel or extra plow steel should be used, especially as bending s same for both cast and plow-steel rope of same diameter. Rc To the known load stresses add wt of rope assumed to be the co bending stresses for this rope over smallest advisable sheave or dr by safety factor required, and compare with ultimate strengths in ing a rope the strength of which is nearest that calculated. With recalculate bending stress and weight of rope, and check result fixed economical ratio between bending stress and load. In sha bending stress may economically be 0.33 to 0.40 of total rope stre shafts such a condition leaves too little of the rope strength for weight of rope is deducted. SELECTION OF ROPE requires a know tions under which it is to operate. Steel is always better than and plow or extra crucible steel is generally preferable to ordinary For same strength, a plow-steel rope may be employed on smalle wrought-iron rope, and on sheaves of same diameter as ordinary s slopes, inclines, and rope-haulage systems, 6×7 rope is better th Lang lay is preferable for endless-rope haulage where car grips vertical shafts 6×19 ordinary lay is standard American practice. with a greater number of smaller wires is not justifiable except ir where the size of the drum or sheave is limited. In a shaft in regularly hoisted, it is generally advisable not to use rope smaller t for psychological reasons, regardless of actual strength required.

Wear and durability of rope depend upon diameter of sheav speed, conditions causing wear or abrasion of the outside wires,

lities are practically independent of the load, so tic limit of the wires. Sheaves or drums of too outer wires. If wires show clear fracture before h, either the sheave is too small or the wires are ure. Tables 22 and 23 (21) show effect of diam- f rope. High speed causes rapid wear; excessive ystallization of the steel. If sheaves have rough de, wear on surface wires is severe, due to chafing and between the rope coils on ungrooved drums. pend upon efficiency of lubrication, and whether y shafts, if rope is well lubricated, and sheaves corrosion and wear of interior wires is negligible. cid, corrosion of wires determines the rope's life.

pe, 6 Strands of 7 Wires, Hemp Core (21)

n ft, showing per cent of life for different diam

80%	75%	60%	50%	25%
12	11	9	7	4.75
10	8.5	7	6	4.5
8	7.25	6	5.5	4.25
7.75	7	6	5	4
6.75	6	5	4.5	3.75
6.25	5.75	4.5	3.75	3.25
5.5	5	4.25	3.5	2.75
4.5	4	3.25	3	2.5
4	3.5	2.75	2.25	1.75

pe, 6 Strands of 19 Wires, Hemp Core (21)

n ft, showing per cent of life for different diam

80%	75%	60%	50%	25%
10	8.5	7	6	4.5
8	7	6	5.25	4.25
7.5	6.75	5.5	5	4
6.5	5.5	5	4.5	3.75
6	5.5	4.5	4	3.5
5.75	5	4.25	3.5	3
4	3.75	3.25	3	2.25
3.75	3.25	3	2.5	2
3	2.75	2.25	2	1.5
.....	2	.....	1.5	.....

ing ropes, each of 1¾-in diam, at Robinson Deep Gold pes were 6 × 19 Lang lay. Diam of wires, from 0.072 97 to 0.735 sq in; tensile strength of wires, 235 200 to 509 lb. Average aggregate strength of wires was in all ntee, average excess being 7.2%. Initial safety factors .15. Ropes were discarded when factor fell below 6. ft. After 6 months use reduction in breaking strength, of wires, varied from 9.3 to 33.5%. WORKING LIFE ork in ft-tons, from 341 058 789 to 1 551 805 722; dis- s. Lowest cost per ft-ton, 0.00008246, was with a me- showing SHORTEST LIFE were identical, each Lang lay,

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6 × 19; 2 diams of wire were used in the strands, 0.114 and 0.076 in; te wires, 257 600 lb per sq in; safety factor, 11.2. Work done by each rope ft-tons; distance run, 21 790 miles; cost per ft-ton, 0.0001708¢. Bes was by a rope not included in the above. This was a 6 × 19 Lang lay 0.094 in diam, with total area of 0.7912 sq in. Tensile strength of wire sq in; reduction of initial strength after 6 months was 35%. Initial sa At end of 357 days the work done was 1 669 935 634 ft-tons; distance miles; cost per ft-ton, 0.000073¢. Lowest cost per ft-ton is apparently high-class heavy rope, carrying a heavy load and working steadily (22).

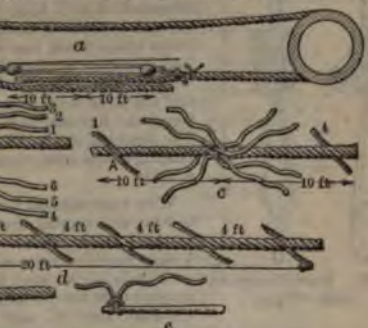
At a slope in West Pittston, Pa (23), 1 800 ft long, with pitch varying 21 ropes were used between 1893 and 1903. They were from 1.25 to 1⅜-i ent grades of steel and number of wires, some Lang lay and some of ordina life was 6 months; coal hoisted per rope averaged 61 500 tons. 6 × 19 r or better service than 6 × 7, probably due to the fact that at vertical an rollers and sheaves of too small a diam were used for a 1⅜-in. 6 × 7 rop the rope was bent 15° over a 5-ft sheave, at another 24° over 4 2-ft rollers.

Care of hoisting ropes. Avoid kinking or nicking the wires tion and use. Frequent inspection is essential. Where men are inspection should be made daily, and a thorough inspection w rope socket or capping is attached directly to cage, and landing c the rope close to the socket will often first show wear or broken slack rope may cause sharp bends at that point. When this occ of rope should be cut off at regular periods, and a new socket j there is enough headroom, a few feet of chain between rope so draw-bar is advantageous, and in some cases required by law. rope end for end, at the expiration of half its useful life, is also so that if wear occurs at any particular point, it will be distributed length of rope.

Lubrication must be efficient, to prevent wear and corrosion minimize surface abrasion. In severe service proper lubrication rope's life 75 to 100% (24). The core should never be allowed enough to absorb moisture. A good lubricant should be chem and of such character and consistency that it will penetrate the core; it should not run nor drip off, nor be so thick and sticky as on the rope or in sheave grooves. Some lubricants, apparently soon harden and flake off, especially in cold climates. Raw linseed runs off easily. More body is given by adding lampblack or fine A common mixture is 1 bbl pine or coal tar plus 1 bushel slake hot. Lime gives body and neutralizes acid often present in c pine tar, tallow may replace the lime. This mixture is sticky an in cold climates does not penetrate well between the wires. Pet vaseline mixed with fine flake graphite and applied hot is good. facturers have rope dressings for different conditions. Lubri applied at intervals of say 1 to 3 weeks, depending on conditions; first cleaned of dirt and gummed lubricant, by passing it through kerosene and scrubbing with stiff-wire brushes.

Splices are not permitted in hoisting ropes used for raising or lowerin haulage, aerial tramway, and power transmission rope, splices may be as strong as the rest of the rope (Fig 42). For a running rope, the splice s 20 ft long for 0.5-in rope, increasing to 30 or 40 ft for 1.25 and 1.5 required are: hammer and sharp cold-chisel, pair of strong nippers, st 2 rope clamps (or small hemp rope slings with sticks, Fig 42, *e*), a knife a copper or lead mallets. A bench vise is convenient. To splice (Fig the rope 20 ft or more, and mark center of lap on each end with string or ch lay each end to center mark, and cut off hemp core (*b*). 3. Interlock the of each end alternately, and draw together until center marks meet

it closely with opposite strand 1 of other end, laying
nd proceeding thus until all but 12 in of strand 1 are
l length and tie the strands temporarily in place. 5.
l so on for each pair of opposite strands, stopping each
e short of the preceding pair (*d*). 6. Bend the rope
e set in place and have equal tension. 7. Wrap ends
ips of sheet lead, and straighten them. 8. With the



12. Splicing Wire Rope

the rope at one strand of the end pair; cut the hemp
wly and follow it up with the strand until the latter
he core at end of this strand. In same manner tuck
ng careful that their ends do not cross over each other.
up the rope, and hammer the strands with the mallet
Shift the clamps and repeat operation at the other
mplete.

p should develop as nearly as possible the rope's full
pplied LOOP is made by bending back the end of the

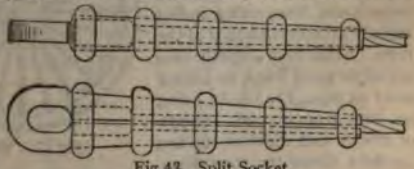


Fig 43. Split Socket

common use in U S, and if properly put on will develop
CKET with rings (Fig 43), often used in England, is more
d not so good. The ends of the strands are turned back
into the bottom recess, and the rings driven
down to clamp the socket tightly on the rope.
It should be forged from the best wrought iron,
and the groove accurately bored for the rope
diam.

Open
ket

Coned socket is of 2 types, closed (Fig 44)
and open (Fig 45). They should be of best
ls, and accurately bored out. For flat rope, the socket
cular. Bending back some of the wires to help fill the

Leschen & Sons Rope Co). At a distance from end of
of socket "serve" the rope securely. Open rope and

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cut off hemp core. Separate individual wires (for this a piece of small p'pe Clean well with turpentine and wipe thoroughly dry. Soak for 5 min i 0.5 water, 0.5 HCl. Bind the loose wires together at top (*do not bend bac* over them and release ends of wires. Close base of socket around the rope

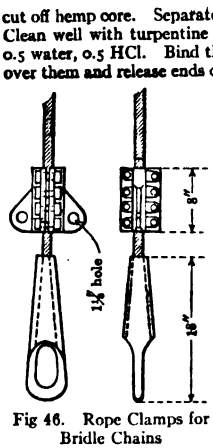

Fig 46. Rope Clamps for Bridle Chains

pour the basket full of molten spelter, **not** Another method (25) consists in dusting the instead of cleaning with dilute HCl, and usin composed of 60% lead, 30% tin, 9% anti bismuth. Antimony increases hardness and fusion temperature, also expanding slightly i heat of molten white-metal does not affect st W. Routledge (26) on testing 28 wires remove and the same wires from a different part of found the strength to be unaffected by ab The breaking of the wires close to socket bending at this point caused by slack rope (s rope").

Wire rope clamps, for attaching bridle c to rope, above socket, are more commonly us at metal mines; laws of most coal district Vulcan Iron Works makes the clamps shown i are usually of cast steel, but forgings are be grooved to fit the rope closely, and the bolts s tightened after the rope is under tension. clamps (27) show the ultimate strength against slipping to be about 10 c of area of the bolts securing the clamp.

15. Hoisting Sheaves

Construction. For diameters up to say 7 ft, sheaves may be of C I in one piece; larger sizes have C-I rim and hub, with W-I spokes. Tread or bottom of groove must be smooth and true, with a radius slightly greater than that of the rope. Grooves of large sheaves are often wood lined, to reduce wear on rope, the blocks being set with grain on end. Depth of groove and width at rim are 3.5 to 4 times the rope diam.

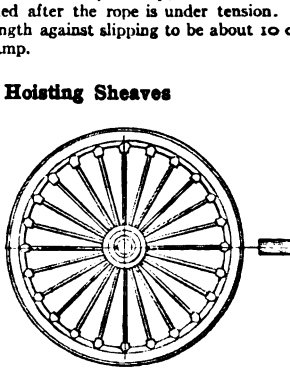

Fig 47. Wrought-iron Spoke

Spokes of W-I sheaves are cast into rim an and 48), making an angle of about 7° with pl Sheave shaft or arbor is solidly keyed in the ported in heavy bearings.

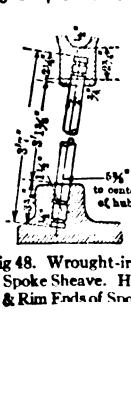

Fig 48. Wrought-iron Spoke Sheave. Hub & Rim Ends of Spoke

Bearings are of the post type (Fig 49), with th plane of base, or of the ordinary horizontal type design of headframe (Art 16). Post bearings either vertical or horizontal position, having the advantage that the resultant of the rope pull always falls approximately in bottom of bearing. Journals should be so proportioned that the unit pressure does not exceed 200 lb per sq in, to permit proper lubrication.

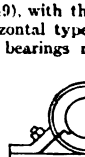

Fig 49.

e not direct quotations of any one maker, but are
s, and are probably correct within 5% (1914).

4. **Sheaves for Round Rope**

of n	Journals		Wt, sheave only, lb	Wt complete, with shaft and boxes, lb	Approx cost complete (1914)
	Diam, in	Length, in			
	1½	.....	58	92	$ 5.00
	.....	.....	86	120	.....
	1½	.....	85	139	6.00
	.....	.....	115	190	.....
	1¾	.....	115	165	8.00
	.....	.....	150	220	.....
	.....	.....	165	315	.....
	2	.....	150	200	10.00
	.....	.....	175	300	.....
	3½	8	350	607	.....
	.....	.....	275	400	.....
	.....	.....	155	225	.....
	2	.....	271	321	15.00
	.....	.....	440	665	.....
	.....	.....	272	375	18.00
	.....	.....	470	690	.....
	4	8	600	961	.....
	.....	.....	460	750	.....
	.....	.....	368	460	25.00
	.....	.....	565	690	.....
	.....	.....	900	1 200	.....
	4½	8	1 050	1 467	.....
	.....	.....	540	725	45.00
	3⅜	.....	.....	1 200	85.00
	.....	.....	1 200	1 800	100.00
	4½	10	1 500	1 973	.....
	3⅜	.....	1 440	2 040	120.00
	.....	.....	1 530	2 400	126.00
	5½	10	2 000	2 764	155.00
	.....	.....	1 535	2 300	.....
	4	.....	2 000	2 800	.....
	.....	.....	1 950	3 030	.....
	6	12	2 700	3 914	.....
	4	.....	2 450	3 450	180.00
	7	13	3 500	5 160	.....
..	4	.....	3 600	5 000	240.00
	8	15	6 100	8 070	.....
..	5	.....	4 500	6 000	375.00

ured at bottom of tread, depends upon the diam and
s large as conditions will permit, considering econ-
ct to life of rope. Manufacturers' lists advise a
pe to diam of sheave of 1 : 48 for 6 × 19, and 1 : 84
eel rope. With these ratios, bending stresses are
ate strengths of rope for crucible cast steel, and 8%
ical wear of rope, the ratios should not be less than
: 125 for 6 × 7 rope; these being equivalent to a
ndividual wires and sheave diam. For flat ropes,
least 80, and of sheave 150, times rope thickness.

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Table 25. Sheaves for Flat Rope, W-I Arms (Wellman-Seaver-

Diam of sheave, in	Style of box	Width of rope, in	Journals		Wt, sheave only, lb
			Diam, in	Length, in	
60	Post	3½	3.5	8	862
60	Flat	4	3.5	8	890
72	"	3½	4.5	10	1 520
72	"	4	4.5	10	1 560
72	"	5	4.5	10	1 600
72	"	5½	4.5	10	1 625
84	"	5	4.5	10	2 000
84	"	6	4.5	10	2 100
96	Post	5	5.5	10	2 400
96	"	6	5.5	10	2 500
120	"	7	7	12	5 400
120	"	8	7	12	5 500

HEADFRAMES

16. Design of Headframes

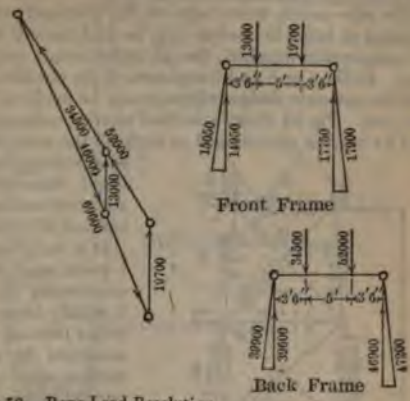
Types. Headframes are of the A-type or modifications of it (Fi
or of the 4 or 6-post type (Fig 58 and 55). The A-type is more e
material, and the stresses are all determinate. The 4 or 6-post typ
when the shaft compartments are not in line with engine; that
of drum is normal to long dimension of shaft. They are con
Penn coal fields, where the compartments are large and so require
overhang of sheave to line up its circumference with center of shaf
there are often 4 compartments in a row. Where a rock house or t
bined with the headframe, the 4 or 6-post type lends itself a little
to the construction.

General principles of design are the same as for any fram
Severe conditions of mining and great variation of load require
factors than ordinary structures, and ample thickness of section fo
bers, because of the exposure to corrosion at mouth of shaft, fr
air. Minimum thickness of section is usually specified as ¼ in to

Height is fixed by: elevation of cage landing or skip dump al
over-all height of cage or skip, when in dumping position; and allow
winding (Art 27). OVERWINDING ALLOWANCE depends chiefly on s
ing, type of engine, and drum diameter. It is usually specified
drum circumference, but with drums of 20 ft or more diam, and wh
discharged to bins or crusher at a considerable height above shaf
may give a prohibitive height of frame. For speeds under 500 ft
10 ft is ample, although it is better to allow more if convenient. Fo
a clearance of 25 ft, from rope socket to sheave center, is rarely exc
the landing is above the ground, and 35 ft where the landing is at
At the landing portals, bracing is arranged to allow ample room fo

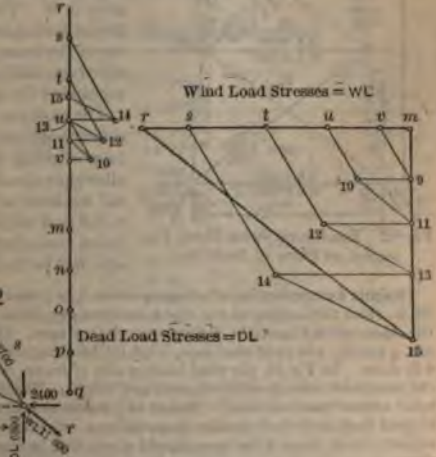
Calculation of stresses. For A or 2-post frames, all stresses are
and are conveniently found graphically. Total stresses are due to: (*a*
or weight of the structure, and sheaves with their bearings (Art 15).
frame is approximated from that of a similar headframe (by a prel
mate) and divided equally among panel points. (*b*) WIND LOAD. W

or headframes which carry a housing, and at 50 lb
ection of the members if not housed. Wind loads
he joints of members. (c) LIVE LOAD, assumed as



50. Rope Load Resolution

ng strength of the rope (see also Art 17 and 18).
ne takes half the live load. With 2 ropes (hoisting



Headframe Stress Sheet. Side Elev

ame takes the full live load of one rope, for 2 and
-post frame, having a central set of posts, the middle
e rope, and each side takes half the load, provided

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the sheaves are equidistant from the center and side posts. In th frame, it is sufficiently accurate to consider the live-load stresses as intersection of front post and back stays; although, in the structu resultant of the rope stresses may not pass exactly through this poi in the side panels takes none of the live-load stresses. Diagon assumed as being in tension only, in steel frames, whence the hori are compression members, though under certain conditions they ma sion. In the 4 or 6-post frame some of the stresses are statically in and for accurate design reference may be made to standard work ject (30). Fig 54 shows a method meeting ordinary requirements. and its bracing are assumed to carry all the wind and dead load, wind stresses in transverse bra stays. Since diagonals are cons sion members only, redundant omitted in calculations.

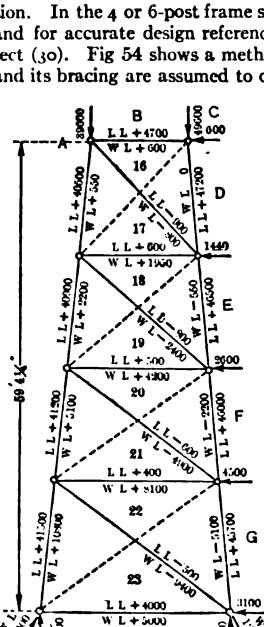

Fig 52. Headframe Stress Sheet, Projection on Plane of Back Stay

It is to be remembered that a whole must be sufficiently st overturning by wind pressure. is usually best to depend on w rather than on anchor bolts, w weakened by corrosion. Ma turning effect of wind will be i parallel to the shortest dimensi area. The total area in sq ft o exposed to wind, *on both windward sides* of frame, multiplied maximum horiz press per sq f press in lb. This may be taken trated horiz load, acting at the ure of the frame. It is resisted structure, including sheaves a multiplied by half the width o

The position of the center of closely approximated in any giv proportioning the width of ba allowance should be made to ca nal resultant of wind press and to fall well within the base area. lation is similar for a structure attached to the frame.

Graphic determination of stresses for an A or 2-post frame is shown in 53, in which the loading on one rope is assumed as half the rope's breaking st the other double the weight of skip, contents, and rope. Shaft, 1 200 ft deep; ore, 4 500 lb; 1-in plow-steel rope, 1 896 lb; breaking strength of rope, 76 o 6 ft diam. In Fig 53, the wind load on side of frame, combined with the the maximum stresses in front posts and bracing. Bracing in front and b takes none of the dead load. Weight of frame is estimated as 50 400 lb, ar to dead load, and wind pressure from the rear, as shown in Fig 51, give m at foot of back brace if the ropes should at any time be removed from the sh shows stresses in back brace due to live load and to wind pressure on one s posts are usually made normal to plane of front and back of frame. Combi tions at foot of one back post gives maximum uplift normal to the base maximum pressure of 61 900 lb, and a shear on anchor bolts at right an pressure (or tension) of 10 000 lb. Similarly for the front posts maximum sure on the base is 48 200 lb and shear 8 900 lb.

eadframe Stress Sheet. Front Elev

mum Stresses in Headframe Members

Maximum compression	Member	Maximum tension	Maximum compression
+45 700	11–12	0	+ 6 900
+63 000	12–13	− 8 000	0
0	13–14	0	+10 300
+ 2 250	14–15	−11 500	0
0	16–17	− 1 700	0
+ 4 750	17–18	0	+ 2 550
0	18–19	− 3 200	0
+ 8 700	19–20	0	+ 4 700
0	20–21	− 5 500	0
+ 4 300	21–22	0	+ 8 500
+ 4 300	22–23	− 9 900	0
0	23–H	− 4 000	+ 5 000

rames. In these, since live and wind-load stresses embers, either the tower, or the A-frame made by ce, may be considered as carrying all wind load from diagonal bracing between back brace and tower is s transmit wind pressure from back brace to tower; n front and middle bents transmit wind pressure ure from the side is carried by each bent and its surface supported by each, and the stresses are . In a 6-post frame the center post is redundant,

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Fig 57 shows the Montana type, which may be modified for almos
tion; Fig 58, a 4-post and back-brace frame. Fig 59 shows an A-f
inclined shaft at West Dome mine, Porcupine, Ont (28), combined

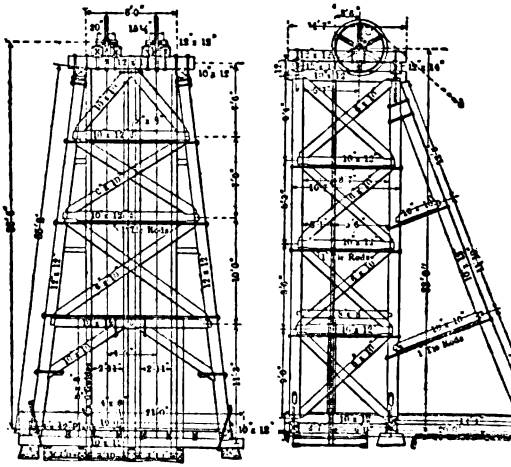


Fig 58. Four-post Frame, with Back Brace

bin; height, 40 ft; hewed posts, 10 × 10 in, with a batter of 1 :
Posts are tied with 3 girts and a cap, back brace with 2 girts, 1-in
rods. Front posts are assun
no load, being simply exten
bucket skids. This frame
$150, exclusive of bin.

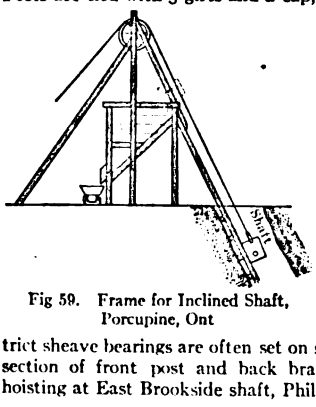
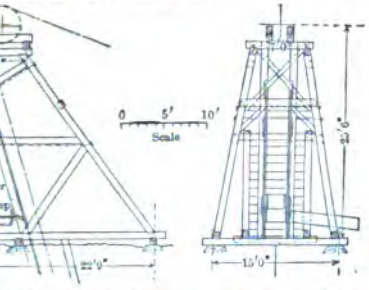


Fig 59. Frame for Inclined Shaft, Porcupine, Ont

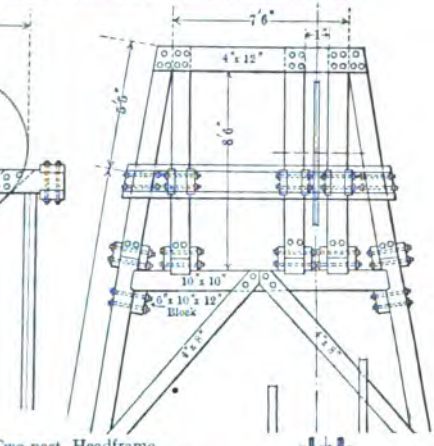
Design of timber headfr

4 and 6-post frames the to
designed to accommodate the
tirely within the top frami
Otherwise, the sheave girders a
short posts set on a girt in pla
bent. For 2-post frames the
supported as in steel fram
Fig 61 shows top of 2-pos
single-compartment shaft, a
load of 8 000 lb. In Penn ar
trict sheave bearings are often set on special castings, which cap the
section of front post and back brace. Fig 62 shows headfram
hoisting at East Brookside shaft, Phila & Reading Coal & Iron Co.

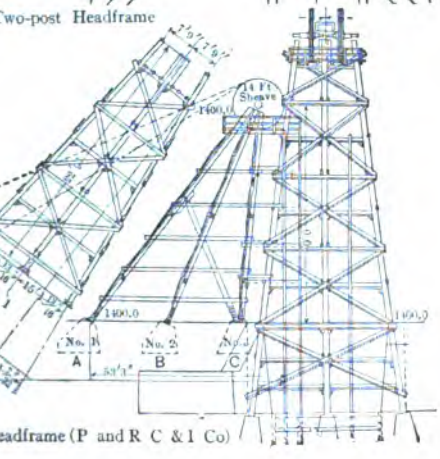
Fig 63 shows details of a timber headframe, with tandem sheaves, workin
on each rope. Mortise and tenon trenailed joints, with tie rods, make neat
but reduce strength of timbers, and labor cost is greater than for gained jo
connections. For the latter, the braces are gained 1 to 2 in on the posts,
bolts. Strength of tension members depends more on bearing area of t
sectional area of timber. Malleable or steel castings at joints make stron
(Fig 62). Steel-plate brackets are not so good as castings. Columns sta



and Ore Bin for Development (*Mines & Min*)



Two-post Headframe



eadframe (P and R C & I Co)

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on the foundation are anchored by angles on opposite sides of the post a
If resting on sills, columns are similarly anchored to the sill, which in
foundation. A cast pedestal, recessed to receive foot of post, is a

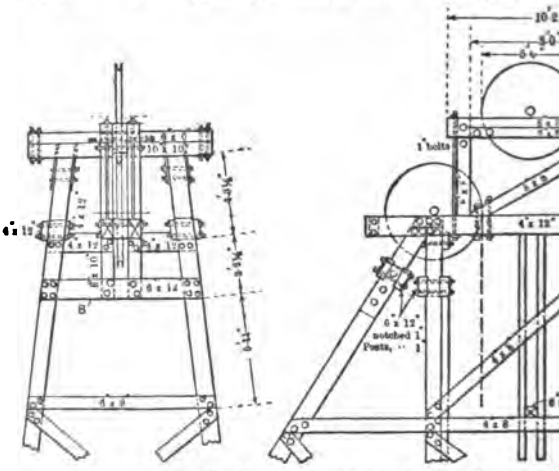

Fig 63. Headframe with Tandem Sheaves

Solid timber is best for main posts, but built-up columns may be use
reduction of section by mortises and bolt holes must be made.

Factor of safety should be double that allowed for ordi
taking into account total rope load from all causes. Or, since h
be stronger than the rope, ultimate strength of frame is made
strength of rope (Art 18).

Allowable unit stresses and strength of members. F
frames M. S. Ketchum (33) gives Table 27. Formula 35 gives a

Table 27. Allowable Working Unit Stresses for Dead Loads,

Kind of timber	Transverse loading, S	End bearing	Columns under 10 diams, C	Bearing across fiber	Shea	
					Parallel to grain	Longitudi-
White oak	1 200	1 200	1 000	450	200	
Long-leaf yellow pine	1 300	1 300	1 000	300	180	
White pine and spruce	1 000	1 000	800	200	100	
Western hemlock	1 000	1 000	800	200	160	
Douglas fir	1 200	1 200	1 000	350	180	

for dead load, which applies to live load when properly convert
(Art 8). Length of columns should not exceed 45 times least di
stress for lengths of more than 10 times least dimension is found

$$P = C - (Cl \div 100\,d),$$

hort columns (Table 28); P = allowable unit stress, olumn, in; d = least side of column, in. Association s of Bridges & Buildings recommended (1895) the n Table 28.

28. Average Safe Unit Stresses

Tension with grain, lb	Compression			Transverse extreme fiber stress, lb
	End bearing, lb	Columns under 15 diams, lb	Across the grain, lb	
10	5	5	4	6
1 200	1 400	1 000	500	1 200
1 200	1 400	1 000	350	1 200
700	1 100	700	200	700
800	1 100	900	200	800
600	.....	800	150	600
700	.....	800	150	750

mns, is based on that of the Division of Forestry, U S Dept

$$P = F \times \frac{700 + 15C}{700 + 15C + C^2} \qquad (36)$$

h, lb per sq in; F = ultimate crushing strength of timber; nn, in; d = least diam, in; F = 5 000 lb for white oak and for Douglas fir and short-leaf pine, 4 000 lb for red pine, wood, and 3 500 lb for white pine and cedar.

ames (Table 29) varies more widely than for steel, timber, which ranges from $20 to $60 per M bd ft. day, cost of erection will be from $25 to $30 per M bolts and nails, depends on design, averaging say used.

od and steel headframes can be sharply drawn for special re cheaper in first cost and total erection time, considering aterial, while steel frames are fireproof and much more nd when life of mine will exceed that of a wooden frame epending on climatic conditions), the greater first cost of gth of steel is known within much closer limits than that le with more certainty as to safety of the structure. Large the variable strength of even the same kind of timber. ll strength of a timber member at the joints, without use make the cost approximate that of an all-steel structure. e that damage due to overwinding or other causes can be ch accidents generally cause local damage only, and are e frame as in case of steel. The Philadelphia & Reading ous accidents with steel frames, replaced them with wooden ooden frames tend to loosen, from the action of moisture, es a more rigid structure for high frames. First cost of d by 20 to 80%. A wooden frame can be erected by local requires services of expert riveters.

18. Steel Headframes

ss-sections of posts and main bracing are determined girders, post bases and guide supports are designed ost frames sheave bearings are usually supported by

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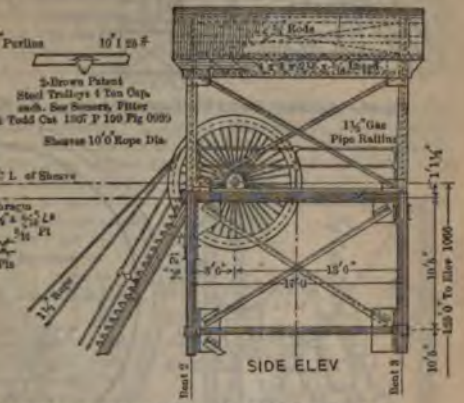
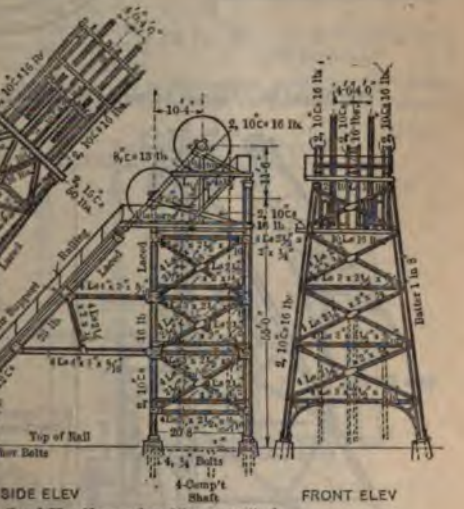
Table 29. Examples of Timber Headframes

Mine and location	Year erected	Shaft					Dimensions of posts, in		Dimensions of bracing, in		Total bd ft timber	Wt of iron work, exclusive of sheaves, lb	Dimensions of base, ft		Height of landing above collar	Distance from center of shaft to center of drum	Cost, $			Total cost, exclusive of sheaves and foundations
		Inclination, deg	No hoisting compartments	Depth, ft	Weight of cage (or skip) and contents, lb	Size of rope, in	Diam of sheave, ft	Type	Height to sheave center, ft	Tower	Back brace	Horizontal	Diagonal			Width	Length			Lumber	Iron work	Labor	
Rose Mohawk, Nev								A	36	10×10	10×10												
Gold Field, Nev								4 post	36	10×10	10×10	6×10	6×10	7 500	2 650					404	133	390	927
N Amer Sm Co, Ont	1913	90	1	275	6500	7/8	6	A	73	10×10	10×10	4× 8	4× 8	7 900	2 200	35	47	45	60	280	135	250	665
Park City, Utah	1902					2¼	6	4 post	40	12×12	12×12	10×10	10×10			17	35.5	at surface	56				
Joplin type, Mo		90	1					4 post	30 and up	6× 8 or 8× 8	none	6× 8	2× 6	1 500 and up		12	12	12 to 35		20 and up	5 and up	20 and up	50 and up
Kanopolis, Kansas	1895	90	2			1¼	12	A	96	4 pcs 6× 8	2 pcs 6× 8	6×6 & 8×10	6×6 & 8×10	26 470		40	72	55				250	750
Stratton's Independence, Colo			2				6	4 post	52	16×16	16×16	12×12	8×10			26	59	at surface					
Portland, Colo		90	2				6	4 post	60	14×14	2 pcs 10×14 spaced 10 in	10×10 and 8×14	10×10 and 8×14			16	56	21	60				
Tesla Mine, Cal		60	3				6	A	44	16×16	16×16	14×14	6×12			20	60						
East Brookside, near Pottsville, Pa	1907	90	2	1800			14	6 post	from 78 back 103	16×16	16×16	12×12	14×14			35	60	at surface	84				
Hiawatha Shaft, Iron	1909	90	1			1¼	8	4 post	80	12×12	12×12	12×12	12×12			30	51						

ms give a broader and more stable support than one igned for the bending moment due to vertical com- n sheave (Art 8), plus wt of sheave. For the front



eave Supports, Ralph Shaft, H. C. Frick Coke Co



Steel Headframe for Gilberton Shaft

porting sheave girders, channels work best into the d top of frame are often covered by a steel-plate riz component of rope load goes to intersection of cross girders on back brace are omitted (Fig 64).

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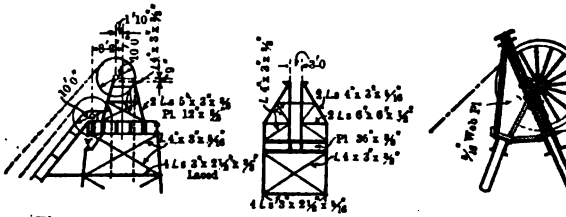

Fig 66. Auxiliary Tower for Tandem Sheaves Fig 67. Sheave Suppo

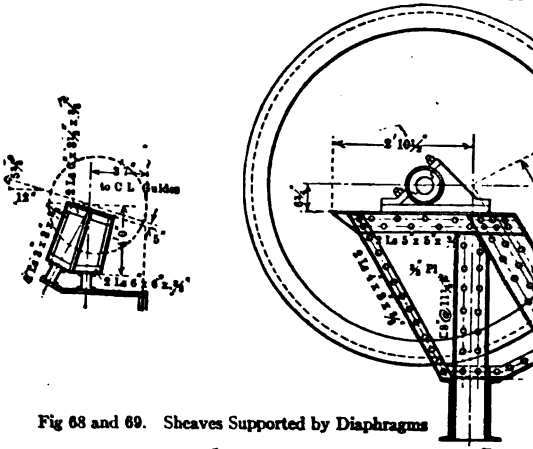

Fig 68 and 69. Sheaves Supported by Diaphragms

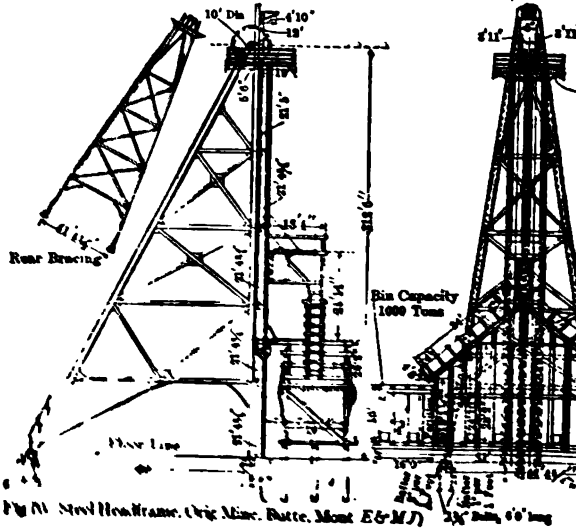

Fig 70. Steel Headframe, Orig Mine, Butte, Mont E&MJ)

th large sheaves, projecting over back of frame, back cross
r sheave, and sheave girders then rest on plate diaphragms,
up from cross girder. Fig 64 shows details of this standard
. Frick Coke Co, by American Bridge Co. Other designs
increase in wt of material, but at increased cost for shop
eep plate girders sometimes support sheave bearings, resting
but with no special advantage. When sheaves must be in
are carried at the back by auxiliary posts in plane of back
heave is supported on an auxiliary tower on main frame

aves may be supported by plate-girder diaphragms, carried
and back brace. If diam of sheave permits, its bearings are
s; if not, they are placed on top. In first case (Fig 67),
the posts by front and back plates. Another cross girder
e sheaves, and the
it. In second case
sts need not extend
s they are to carry
ging sheaves. Fig
No 7, 5, 12, and 11,
modern A-frame,

re generally bolted
from shaft collar to
s are supported in
nels or angles, from
nt; they are more
uxiliary girt across
uts (Fig 71). In 4
uides are not com-
ons, but supported
eams or channels,

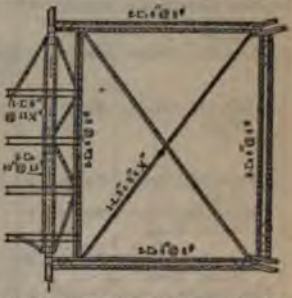

Fig 71. Horiz Sec of Frame, Showing Guide Supports

support is needed at intermediate points, by angle braces
ig 73 shows details of fastening the guides to steel supports.

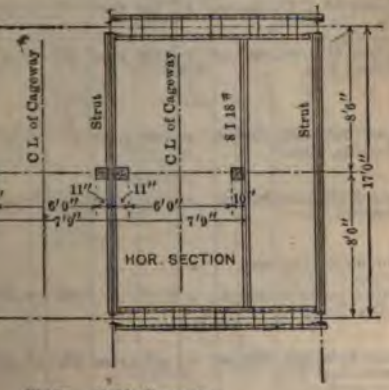

Fig 72. Guide Supports

nsmit load to the foundation within the allowable safe unit
tion. This, for best Portland-cement concrete, is 500 lb per
atone masonty, 400 lb per sq in. There should be at least

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...es, with Costs

	Dimensions horiz and diag bracing				Distance, back post to brace, ft	Height, landing above collar, ft	Dist, center of shaft to center of drum	Weight, without sheaves, lb	Cost erected, without sheaves and foundations
	Side horizontal struts	Side diagonals	Front and back horizontal struts	Front and back diagonals					
	2 12 in Cs	2 9 in Cs	2 12 in Cs	2 9 in Cs	....	....	....	117 000	$3 920
fe	2 12 in Cs	2 10 in Cs	2 12 in Cs	2 10 in Cs	....	....	....	56 000	.....
	2 9 in Cs	2 Ls, 5×3×5/16	2 9 in Cs	2 Ls 5×3×5/16	....	....	....	74 700	2 755
e s 1/2	2 Cs, 15 and 12 in			.....	61	38	....	292 000	8 940
Cs	2 8 or 9 in Cs	2 6 or 7 in Cs	2 6, 8 or 9 in Cs	2 6 or 7 in Cs	....	....	83	45 000	2 327
e	2 9, 10, 12, or 15 in Cs				....	....	....	183 000	7 320
Cs	2 7 in Cs	2 6 or 7 in Cs	2 7 in Cs	2 6 or 7 in Cs	44	31	110	79 000	3 857
Cs	2 5 in Cs	Same	Same	Same	30	....	75	42 200	2 500
5/16; s, 5/16	None	None	Front 1 L, 4×3×5/16; Back 2 Ls, 3 1/2×2 1/2 ×5/16	1 L, 3×2 1/2 ×5/16	30	....	....	42 000	.....
in rope ys in Cs	1 8 in C 2 6 in Cs	1 L, 3 1/2×2 1/2 ×1/4 2 Ls, 3 1/2×2 1/2 ×1/4	1 8 in C, and 1 L, 4×3×5/16 Front 2 Cs, 10 or 6 in; Back 2 Cs, 6 in	1 L, 3 1/2×2 1/2 ×1/4 2 Ls, 3×2×1/4	 34	31 15	80 100	23 000 35 250	1 198 .....
in Cs	2 Cs, 5 or 6 in	2 Ls, 4×3×5/16 or 3 1/2×2 1/2 ×5/16	2 6 in Cs	2 Ls, 4×3×5/16 or 3 1/2 ×2 1/2×5/16	34	39	150	79 000	5 000
in Cs	2 6 in Cs or 2 Ls, 2 1/2×2 1/2 ×1/4	2 Ls, 2 1/2×2 1/2 ×1/4	Front 2 Ls, 5×3×5/16; Back 2 6 in Cs	2 Ls, 2 1/2×2 1/2 ×1/4	39	40	100	65 000	6 920
.....	.....	.....	.....	.....	....	....	....	60 000	4 385
.....	2 Ls, 6×4×7/16	Same	Same	Same	....	20	116	.....	.....
Ls, 6×1/2	Cs	1 L, 6×3×3/8	Cs	1 L, 6×3×3/8	41	67	60	350 000 includ-ing trestle	16 175 includ-ing trestl
ume	2 Pls 8×3/8; 4 Ls, 3×2 1/2×3/8 battened	2 Ls, 3×3×3/8	same as side	2 Ls, 3×2 1/2×3/8	75	80	84	.....	.....

Table 30. Examp

Number	Name and location	Year erected	Shaft			Wt of cage (or skip) and ore, lb	Diam of rope, in	Diam of sheave, ft	Type	Height to sheave center, ft and in	Hoisting speed
			Inclination, deg	No hoisting comparts	Depth, ft						
1	St Lawrence, Mont	1897	90	2	2100	21 000	7×1½ flat	10′–0″	A	97–0	2
2	West Colusa, Boston & Mont Co, Mont	1897	90	2	....	.....	....	.....	A	70–0	..
3	Anaconda, Mont	1898	90	2	2400	21 000	7×1½ flat	.....	A	58–8	2
4	High Ore, Butte, Mont	1898	90	2	2800	19 000	7×1½ flat	10′–0″	A	100–0	2
5	Stewart, Butte, Mont	1898	71	2	....	.....	7×1½ flat	7′–6″	A	55–0	..
6	Mountain View, Butte, Mont	1899	90	2	....	21 000	7×1½ flat	.....	A	80–0	2
7	Basin & Bay State, Basin, Mont	1899	90	2	....	.....	.....	10′–0″	A	70–0	..
8	Second Speculator, Butte	1899	90	2	....	21 000	Round	.....	A	50–0	2
9	Union shaft, Virginia City, Nev	....	90	2	2000	4150	1	7′–0″	A	50–0	1
10	St Louis S & R Co No 5, St François Co, Mo	1906	90	2	500	6100	⅞	6′–0″	6-post guyed	70–0	
11	Copper Queen, Bisbee, Ariz	1906	90	2 Main 1 Aux	1700	9700	1¼	7′–0″	A	60–0	1
12	Inland Steel Co, Hibbing, Minn	1908	90	3	185	10 400	1⅛	6′–0″	A	76–0	
13	Tonopah-Belmont Tonopah, Nev	1911	90	2	1700	10 700	1	7′–0″	A	75–0	
14	Gwin mine, Calaveras Co, Cal	1904		2	....	.....	.....	.....	A	100–0	..
15	Prospect colliery, Wilkes-Barre, Pa	1898	90	2	....	.....	1¼	6′–0″	4-post	61–0	..
16	Lehigh & Wilkes-Barre Coal Co, No 9, Sugar Notch, Pa	1899	90	2	315	13 500	1½	12′–0″	6-post	.....	2
17	Phillips mine, S W Pa	..	90	2	268	.....	1⅜	10′–0″	6-post	107–0	..

horiz and diag bracing			Distance, back post to brace, ft	Height, landing above collar, ft	Dist, center of shaft to center of drum	Weight, without sheaves, lb	Cost erected, without sheaves and foundations
le nals	Front and back horizontal struts	Front and back diagonals					
n Cs	2 12 in Cs	2 9 in Cs	....	....	....	117 000	$3 920
n Cs	2 12 in Cs	2 10 in Cs	....	....	....	56 000	.....
s, X5/16	2 9 in Cs	2 Ls 5X3X5/16	....	....	....	74 700	2 755
nd 12 in		.....	61	38	....	292 000	8 940
7 in	2 6, 8 or 9 in Cs	2 6 or 7 in Cs	....	....	83	45 000	2 327
12,	or 15 in Cs		....	....	....	183 000	7 320
7 in	2 7 in Cs	2 6 or 7 in Cs	44	31	110	79 000	3 857
ne	Same	Same	30	....	75	42 200	2 500
ne	Front 1 L, 4X3X5/16; Back 2 Ls, 3½X2½ X5/16	1 L, 3X2½ X5/16	30	....	....	42 000	.....
2½	1 8 in C, and 1 L, 4X3X5/16	1 L, 3½X2½ X¼	....	31	80	23 000	1 198
s, 2½	Front 2 Cs, 10 or 6 in; Back 2 Cs, 6 in	2 Ls, 3X2X¼	34	15	100	35 250	.....
4X or 2½ 16	2 6 in Cs	2 Ls, 4X3X5/16 or 3½ X2¾X5/16	34	39	150	79 000	5 000
s, 2½	Front 2 Ls, 5X3X5/16; Back 2 6 in Cs	2 Ls, 2½X2½ X¼	39	40	100	65 000	6 920
....	.....	.....	....	....	....	60 000	4 385
ne	Same	Same	....	20	116	.....	.....
X¾	Cs	1 L, 6X3X⅜	41	67	60	350 000 including trestle	16 175 including trestle
s, X⅜	same as side	2 Ls, 3X2½X⅜	75	80	84	.....	.....

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2 anchor bolts at foot of each post. Fig 74 shows details of base for a colu
latticed channels, for a back brace, and Fig 75 an anchorage for front post

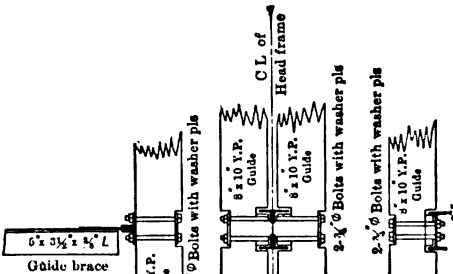

Fig 73. Details of Fastening Guides to Steel Supports

Sometimes placed bet foundation advantage

Safety be larger nary stru of varyi indetermi (Art 16). recommer for dead and 8 for load being the wt of tents, plu plus load rope fricti tor of 2.6

strength of rope or ropes be used instead of live load. If unit str dead and live loads, are added the total unit stress should not exceed

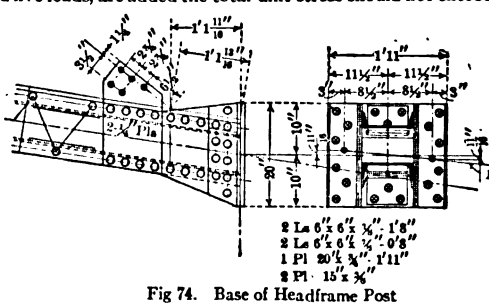

Fig 74. Base of Headframe Post

unit stress for dead load by more than 25%; that is, the combine should not be less than 3.2. If the basis of breaking strength of the wind-load stresses may be neglected, except as regards overturning

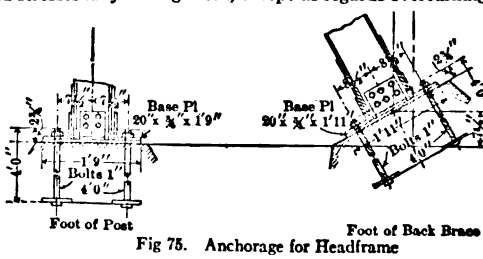

Fig 75. Anchorage for Headframe

A rational method is to design for an ultimate strength equivalent to strength of rope. Then, by assuming the live load as half the rope's bre

ll stresses, the live, dead, and wind loads may be combined.
g proper size of member from structural steel handbooks,
r calculated with a safety factor of 4.

s, with a safety factor of 4 (live load being converted to
ension, 16 000, and for compression, $16\,000 - 70\,(l \div r)$,
ere l = length of member, in, and r = least radius of
ompression members should not exceed $100\,r$ for main
condary members.

commended:

aring	22 000 lb per sq in
ear	11 000 " " " "
treme fiber	24 000 " " " "
hear on net section	10 000 " " " "

lframe (Fig 64), the American Bridge Co used following unit
king load of the ropes from sheave supports to headframe
n, 25 000; rivets, single shear, 15 000; rivets, bearing, 35 000;

f steel headframes are given in Table 30. In 1909
Mach'y Co quoted approx price of steel headframes
and foundations, as follows: for weight of 20 to 25
tons, 5¢; 75 tons, 4.5¢ per lb.

STING MISCELLANY

Cage Guides and Skip Tracks

tical shafts are the rule in America. Long-leaf yellow
se of its strength, hardness, and straight grain. Oregon
inclined to splinter, warp, and
g. Red or Norway pine is the
ze of guide depends on weight
g speed. For small cages and
fficient, 6 by 8 in being about
hoisting. Guides must be
oughout, thus reducing actual
from the rough; 4 by 6-in
by 5.75 in. LENGTH should
at the shaft sets; though if
ot so essential, provided the
han say 4 ft apart. CLEAR-
cage shoe depends upon the
sets and guides are lined up.
d on the face than the edges,
lt to maintain if timbers bend
on edges and 0.5 in on face is
NING TO SHAFT SETS is com-
ws, with countersunk heads
on of lag screw into bunton
o thickness of guide. Use of
itates removal and replacement of guides. Iron angles
s, bolted to both guide and shaft timber (Fig 78).

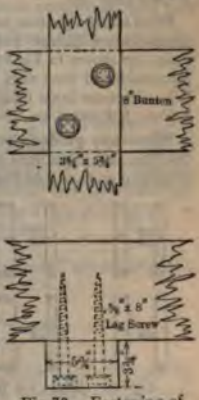

Fig 76. Fastening of Wood Guides

ides. At No 2 shaft, Hancock Consol Mining Co, Mich.
screw heads countersunk in slot (Fig 77); if a screw becomes

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loose it will work down this slot, instead of sticking between guide a
guides are backed by studdles between the dividers. Fig 78 shows faste
Tobin & Dunn mi
screws in guide are
3-in angle is placed
bolted to guide by
and to divider by
Fig 79 shows me
mine, Mich. Guid

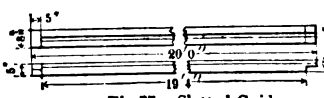
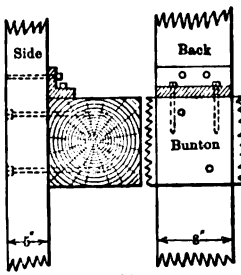
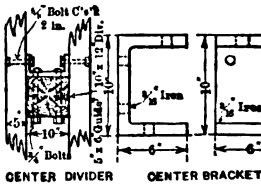


Fig 77. Slotted Guides

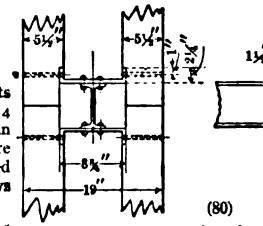
by 22 ft long, with butt joints. Brackets are 3/16 by 5 in. Fig 80 show
to steel work of headframe No 10, Table 30.



(78)



(79)



(80)

Fig 78, 79, 80. Fastenings for

Guide joints. Fig 81 shows joints used on the Rand and elsewhere, for 4 by 8, 5 by 6, 4 by 6, and 4 by 5-in guides. At Butte, Mont, guides are usually 4 by 9 and 5 by 9 in, fastened by two ⅞ by 8 or ¾ by 9-in lag screws at each shaft set.

Steel slotted guides, as used in No 5 shaft, Crown Mines, Ltd, Witwatersrand, are shown in Fig 82. The brackets are bolted to divider with o. three-cornered fang nuts. Oval-necked bolts and spring nuts fasten gu Cage shoes are brass lined, to reduce wear on guides. RR RAILS are also used as guides.

Wire-rope guides, preferably of locked-coil type, are often used in Europe, for shafts of circular section. At Dalton colleries, Roth-

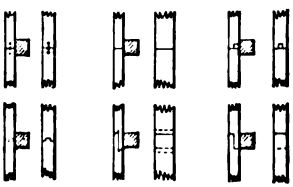


Fig 81. Joints for Wooden Guides

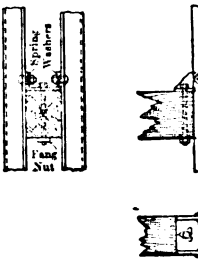


Fig 82. Bracket for Steel C

erham, England, the shaft is 21 ft diam and 2 238 ft deep. Cages are 3-d and carry 7 tons of coal. At each corner of the cage is a locked-coil r 14 tons tension; and between the cages, to prevent collision at passing p bing guide ropes under similar tension.

Skip tracks for inclined shafts, if at a pitch not exceeding 30 surface tracks, except for measures to prevent downward cree

es at 2 or 3 places in a rail length is generally sufficludes sills, rails rest on these with 1 or 2 intermediate wedges. In absence of timbering, every third or 1" into side walls of the shaft. Rails not lighter even for light skips and slow speeds; 45 and 50-lb CRETE STRINGERS for skip tracks are replacing sills

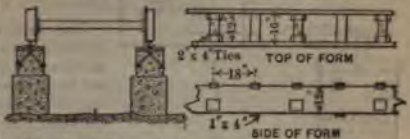


Fig 83. Skip-track Concrete Stringer and Form (*E & M J*)

Bolt holes are made by inserting rods ⅛ in larger than when concrete is partly set. Fig 83 also shows the form for molding upper part of stringer, with blocks attached for forming the recesses for nuts and washers. 2 by 4-in creosoted blocks are set in the concrete, 1 in deep and 18 in apart, to cushion the rail. Concrete of 1 : 2 : 5 is mixed on the level above, and poured down troughs to desired point. In 24 hr 6 or 8 men can build 100 ft of

ip Track (Longit Sec)

P SLOPES it is difficult to hold the rail. At Copper Range shown in Fig 84 and 85 was devised (38). Rail rests on r, set in the concrete base. Three notches on each side

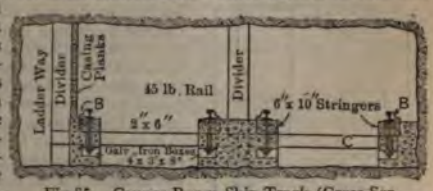


Fig 85. Copper Range Skip Track (Cross-Sec on E F, Fig 84)

building the forms 2 by 6-in crosspieces *C* are set to grade, are laid. Concrete mixture is 1 : 2 : 6.

Hoisting Signal Systems

rated by hand is cheap and positive for a few levels, or 500 ft. Beyond this the springs or weights to e require a heavy pull; and the man giving the signal rly received or understood by the engineer. Light 5 to 0.25 in diam, makes the best bell wire, though of direction should be made by bell cranks, not by e better than iron staples to hold the bell wire to s usual arrangement.

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Pneumatic signals are largely used in collieries in eastern and
Fig 87 shows a station outfit. Ordinary pipe and fittings are
stations. A stroke of the ha
cylinder *A* slightly compresses
causing all the gongs of the sy
and the whistles to blow. The
be used as a speaking tube.

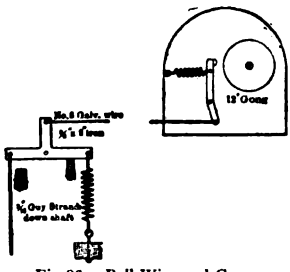

Fig 86. Bell Wire and Gong

This method is reliable, with
maintenance expense; not affecte
accidental signals cannot be given as in some electrical systems. No indicator can be used, as in electrical systems, to supplement bell signal. Cost for each station outfit (Fig 87), not including piping between stations, is: 300 ft signaling distance, $20; 800 ft, $25; 1 500 ft, $30; 2 000 ft, $40. In the Missouri lead district, a small whistle is often placed on the air line at top of shaft, which is blown by a hand wire from the shaft stations; a bell in engine room is sounded from the shaft mouth.

Fig 87. Pr
Gong an

Electric signal systems are of general application, but for reliability require careful installation and maintenance. There are 3 classes.

A. Low-voltage direct current, operated by wet, dry or storage batteries. Annunciator bells should be iron clad and waterproof, and heavier and more durable than house bells. Only the best rubber-covered double-braided wire should be used, and al down the shaft in metal conduit. Double-conductor twisted wire is good. are cheap and convenient, but are a source Small single-throw knife switches are bet buttons; or specially designed contacts, dirt and water, may be used. All the bel one circuit (Fig 88, *B*), so that the signal s tions and in the engine room. If only the an I engine room bells ring (Fig 88, *A*), co cause l by two stations attempting to si ously. Engine room and station bells a that the signal may be repeated from en necessary to signal from the cage at any po wires *a* and *b* (Fig 88, *B*) may be bare and lators on shaft timbers, the signal being m contact with a piece of iron or by pressir gether. In wet shafts this plan gives t batteries are rapidly exhausted. Fig 89 SIGNAL AND LIGHTING SYSTEM, at Geduld m (19), as arranged for sinking. Shaft sig ordinary bell cord, operating an electric rings a bell at top of shaft and in the engi also upon a relay and semaphore. Throu dent battery, the relay rings the bell on th The relay insures that, whatever may happ ing lamp and its bells, the engine room sign The white and red semaphore disks show ment from which the signal was given, in addition to the bell signal. T showing the connections represent double conductors.

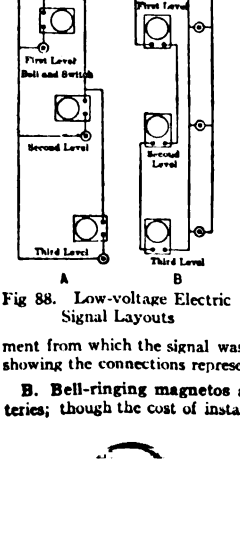

Fig 88. Low-voltage Electric Signal Layouts

B. Bell-ringing magnetos are excellent, as they do not become exha teries; though the cost of installation is much higher. Loud-ringing wea

t for one installation, consisting of one station outfit of 1 -in gongs, with a similar engine room outfit so that engineer can return signal, 1 extra bell at shaft mouth, and 200 ft of double-braid rubber-covered No 16 conductor wire, cost $35. Besides this, 150 ft of 0.75-in pipe was used for conduit, the shaft being very wet; also a few insulators on the surface. Labor cost of installation, about $17. The 3 gongs and 2 magnetos were connected in series.

C. Power or lighting circuits (direct current) not exceeding 220 volts, where

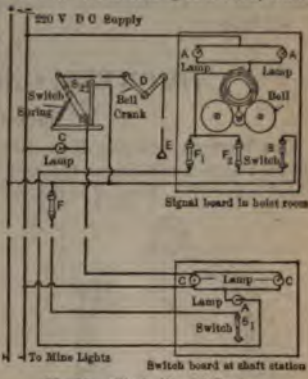

Fig 90. Shaft Signal System, Rosiclare, Ill

red, may be utilized for arranging signal systems, both bells . Alternating current, of 220 volts is dangerously high, and ; volts.

osiclare, Ill. All wiring is double braid, rubber covered, and conduit. For signaling, S_1 is closed, ringing engine room bell al is returned by pulling E, thus closing switch S_2, which pened. stand. s. To moved h S is l lights are so-urrent. onclad, ns and

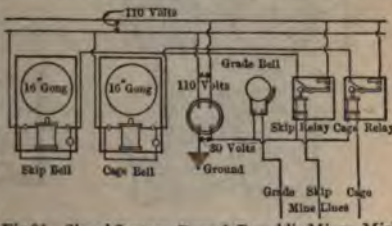

Fig 91. Signal System, Penn & Republic Mines, Mich

ic iron nating down grounded on one side, the other leading to a relay for each p, cage and one side of a grade bell, in shaft house. The the grade bell are connected to one of 3 No 4 bare copper rs in the shaft. By grounding any one of these wires a cur-ade bell or the relays in engine room. The relay then rings 110-volt circuit. A heavy single blow is struck by the A-C

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solenoid, an indicator registers number of bells, and a lamp is lighted. ... may be grounded from the cage at any point in shaft.

At Raven Mine, Butte, Mont, 110-volt A C is used for signaling. In ... and *O* are for lighting, *R* burning constantly. Wire *N*, for signaling, is c...

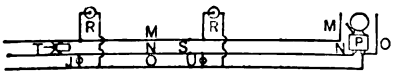

Fig 92. Signal System, Raven Mine, Butte, Mont

single-stroke solenoi... room. *S* is a push ... levels, and *T* a spec... wet levels. *U* and *J* ... signal lamps, but w... buzzers. When *S* o... *P* and buzzers *U* and *J* ring. Bell and buzzers being in parallel should ... same resistance. By a switch in line near *P*, the engineer can return th... double-braid okonite, and rubber-covered wires are used, without enclos... both kinds proving satisfactory. Pull box and buzzers, made at mine, ... $5 and $3.15 each.

BUCKETS, CAGES, AND SKIPS

21. Buckets and Crossheads

Ore buckets, unless made at the mine, are of steel (see also A... shows a bucket made from the lower three-quarters of an oil ba... about 4.5 cu ft. Band *B* is 0.25 by 2.5 in, riveted to bottom ...

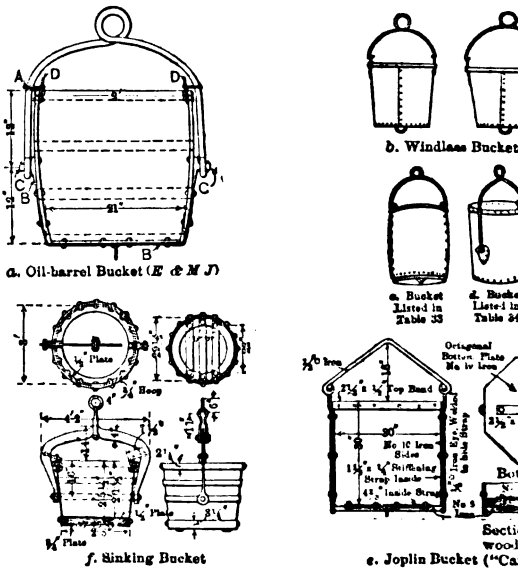

Fig 93. Hoisting Buckets

forged into bail pins *C*. Rings *A* and lugs *D* prevent accident... This bucket is too large for a windlass; it requires a horse whim, ... Fig 93*b* shows light steel windlass buckets, as made by mining ... (Table 31). Stock sizes of large buckets for heavy work, wit...

Table 31. Windlass Buckets

Gage of steel	Capacity		Wt	Cost per lb
	Cu ft	Lb		
No 16	1¾	200	28	10 to 12¢ for smaller, 8 to 9¢ for larger
16	2	250	28	
14	3	375	55	

by makers. Large mines often have buckets for sink-
order. Bail may be attached to ears at top, or to side
enter of gravity. The bottom is best dished, with a
aching the dumping hook. Self-dumping buckets are
o and clean themselves better, but with some danger
hile being hoisted. Buckets with straight or flaring
iable to catch on shaft timbers.

Capacities
93*c*)

Wt, lb	Capac-ity, cu ft
270	5
270	5.5
280	6
300	7.5
340	8.5
350	10
400	12.5

Table 33. Sizes and Capacities of Buckets (Fig 93*d*)

Diam bot-tom, in	Diam top, in	Depth, in	Wt, lb	Capac-ity, cu ft
18	24	26	260	5
20	26	28	300	7
22	28	32	350	9
24	30	36	400	12
28	34	36	450	16

o, buckets (locally "cans") are used commonly for regular
opment work. Largest size is 34 by 34 in, called a "1 600-lb
h about 1 200 lb. Other sizes are: 30 by 30, 30 by 28, and
rst dimension. Smaller sizes are: 26 by 28 and 24 by 28 in.
No 8 steel, with various modes of stiffening with horizontal
ms. Fig 93*e* shows typical Joplin bucket; ears are welded
y hook under bottom flange (33). At iron mines of Mine-
ket is used for sinking (Fig 93*f*). It has flaring sides, with
s.

designed so that they can not accidentally unhook, due to
ing shaft timbers. They are best made of Norway iron.
3) is safe, but with a spring strong enough to be safe it is
ine hook *b* is safe and convenient for rope up to 0.5 in diam,
th heavy, stiff rope. Chain hook *c*, in which only one part
gh the hook opening, is safe, but not convenient. DOUBLE,
fe and convenient (Fig 94, *d* and *e*). Any liability to acci-
d by a ring surrounding both hook shanks, which must be
ate. Fig 94*d* shows use of 2 pairs of sister hooks and chains
g bucket at Hancock No 2 shaft, Mich. Two forms used in
in Fig 94*e*. Fig 94*f* shows a simple safety hook. Gap *C*,
et bail, is closed by thimble *D*, made of heavy pipe, or a piece
BINED HOOK AND SWIVEL, used at the Rand Collieries, Ltd,
94*g*. Dimensions given are for a hook for a 2-ton bucket.
tch thread, so that 3 or 4 turns by hand will allow bucket

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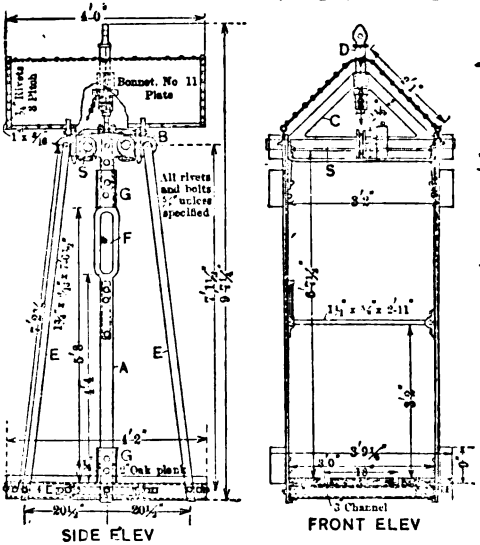

Fig 100. Light Metal-mine Cage (wt, 800 to 1 000 lb, for 2 500 to 3 00

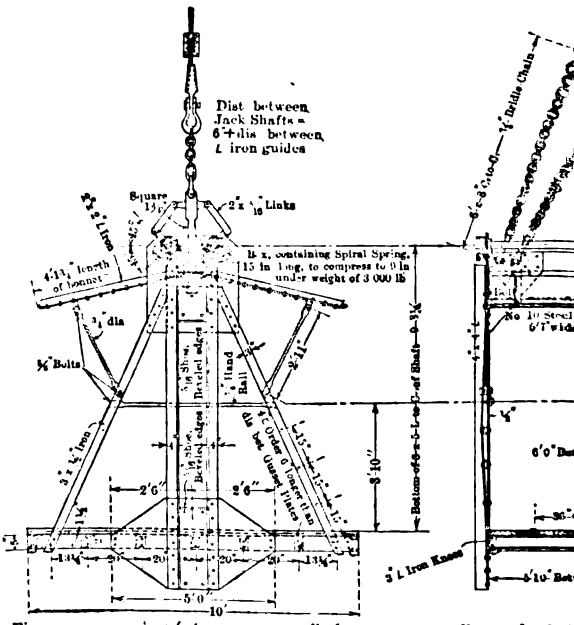

Fig age (wt, 2 500 to 4 000 lb, for 7 000 to 9 000 lb gross load),

Cages: General Construction

ince the cage is a dead load, its weight should be as e useful load as is consistent with safety. Fig 100 and or light and heavy loads, respectively. Platform frame ral shapes (channels for side members and I-beams or . For HEAVY CAGES (Fig 101) the center suspension single channel, or a pair of angles, riveted to a plate gs and pins for bridle chains. In this case, upper cross tural shapes. For LIGHT CAGES (Fig 100) center sus- ing guide shoes *G*, may be a flat, welded to forging *B*, rings. To *B* is bolted or riveted the triangular forging -bar *D*. Inclined members *E*, from *B* to the cage deck, t or round bars. If members are narrow bars, bolted tructural shapes), rivets are used, with at least 2 rivets

ig 106 and 107) are used more in Europe than in America, to some extent in Western U S, for increasing the shaft capacity e. In Butte, Mont, and on Comstock lode, Nev, cages have . 4-decked Michael col- each deck d 2,300 or oisted from 3). Upper ge (though ty gear and ist of plat- ly, and are above, for ined guides. loss of time deck must receive or time there cks, so that decks; but e of instal- r handling deck cages dling men, These are rtments, or age substi- f each shift.

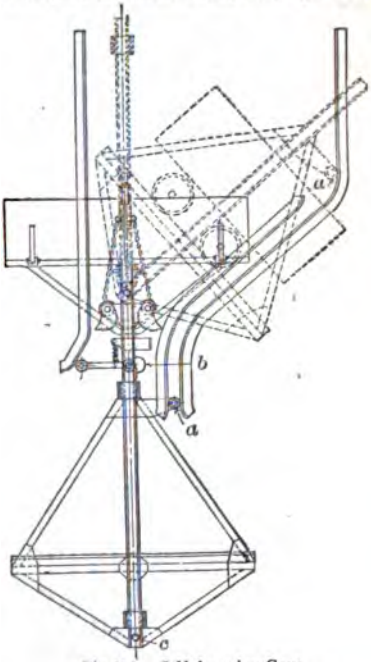

Fig 102. Self-dumping Cage

(Fig 102) inous and e coal dis- Cage con- ter runs in l to this at e and plat- l fast by a oing point, tward, by roller *a* engaging in curved guides, thus sup- g angle. Meantime, the outer frame runs up on main 92), to prevent accidental tipping of inner frame before

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should be sharp and deep enough to bite well into guide, with an angle a
less than 45°. Width of cam is about 2 in for small, and 3 in for larg
connection between draw bar and chain wheel on jackshaft (Fig 100) is si

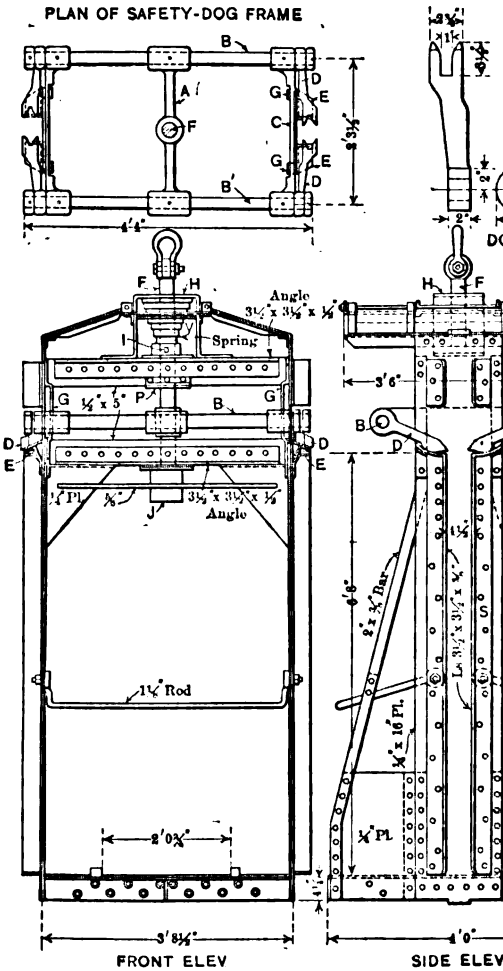

Fig 104. Munzner Safety Device

tion for light cages. In coal districts the cams are sometimes placed at
3 ft above platform, and connected with jackshafts by levers and lo
designs of flat spiral or helical SPRINGS, encircling the jackshafts or draw
Fig 100, 101 and 104. When on jackshaft, one end of spring is fast to

ide of cage; or, short levers keyed on each shaft are connected For heavy cages, a powerful spring encircling the drawbar ncidentally eases the shock when starting to hoist. An objec- kshafts by one spring or set of springs is that, if one shaft o act against an unusually hard or smooth place on the guide, ing it into action. They are best operated by independent

ch. This is an old device. Fig 104 shows one form, as used S E Missouri. It is claimed that the cage is stopped more d cams. Dogs *D* have toothed or split sharp ends, which are loose on shafts *B*. While cage hangs on drawbar, collar *I* ring *V* until frame *B* strikes plate *P*. Points of the dogs are les. If rope breaks, *B* is thrust down by the spring, and the e forced against guides. Frame *B* works in vertical guides *G*. hen gripping guides, is stopped by the shoes. Spring is a 0.375 in thick, and is supposed to take 80% of weight of cage. costs about $425. For description and tests of a number of of the Transvaal Commission" (19).

cages. Plain steel colliery cages, for a gross load of 4 000 lb; for heavier loads, up to 6 000 lb. A combined anthracite district, weighs 5 600 lb, has a deck 6 ft by 500-lb car and 6 000 lb of coal. Cage at Sugar Notch s 5 000 lb, car 2 500 lb, and contents 6 500 lb. Drop- weighs 2 680 lb, car 1 270 lb, contents 2 000 lb. Large ges weigh 6 000 to 8 000 lb. Single-deck cages, with n gross load, weigh 900 to 1 400 lb; for 3 000-lb load, eavier loads, up to 2 800 or 3 000 lb. Each additional the weight. Self-dumping cages are 30 to 40% heavier cages of standard design cost 8 to 12¢ per lb; heavy e from special designs requiring new patterns, cost is

24. Design of Cages

ould be 10, whence allowable unit stresses for iron and lb; and for compression, $S = 6\,000 - 70\,(l \div r)$; where nd r least radius of gyration, both in inches. Rivets 5 000 lb; double shear, 10 000 lb. Bolts, when used in pins, single shear, 3 500 lb, double shear, 7 000 lb. If car and contents, and the dimensions of hoisting com- esign is a matter of mechanics and knowledge of the l auxiliary parts, as determined by practice. Clearance eck and shaft timbers is 2 to 4 in. Width is fixed by tance between guides, less thickness of shoes and side between shoes and guides (Art 23).

nsions of cage deck, and place crosspieces equidistant. Small ading the ends; long decks may have 5 to 7 crossbars. Max- l its point of application on the deck rail are determined, and nbers are then calculated as beams (Sec 43). It is to be noted dy is not placed symmetrically on its truck; one pair of wheels r. Throughout the calculations the dead load is multiplied l (Art 8). If l = length of rail between crossbars, and W = ment $M = 0.25\,Wl$; also, $M = \phi S$, where ϕ = unit stress = ctural shape is used, $S = 0.25\,Wl \div 6\,000$, and proper size is . For a rectangular bar, $M = \phi bd^2 \div 6$; in which b is width ssume b and solve for d, in $d^2 = (3\,Wl) \div (2 \times 6\,000\,b)$.

DECK CROSSBARS is determined by relation between their dis- of car, remembering that front wheels may carry more load

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in which, P = load on each spring; S = max unit stress = 80 000 lb; R = r
center of coil; l = developed length of coil; G = modulus of elasticity in te
12 000 000; d = diam of spring rod; f = deflection of spring in inches under
In this case, pull from drawbar to chain wheel or lever must be converted into co
load in direction of axis of the spring. (See also Sec 41, Art 10.)

25. Caging Devices and Landing Chairs

Caging of small cars is nearly always done by hand. Large cars a
often caged by hand; but mechanical devices are common, especially in co

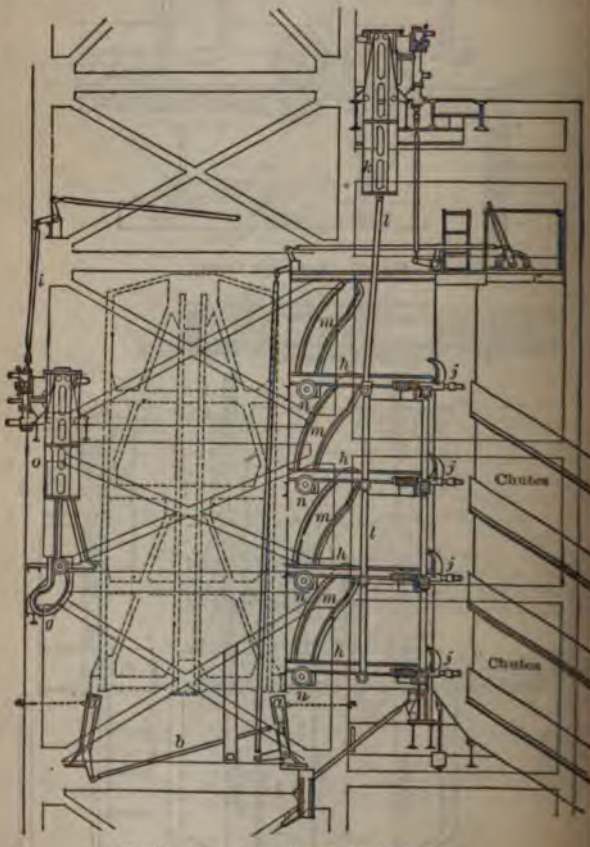

Fig 107. Surface Landing, St Michael Colliery

and increase capacity and safety. For hand caging, where station layou
mits tracks on both sides of shaft, the track approach to cage is laid with
2% grade, so that cars will run on by gravity, the loaded car pushing

Car checks, placed at a safe distance back from the ever worked by the station tender. They should be a car passes, the check is raised into position to hold face landing, if point of dumping be some distance be pushed off by the empty; but, with hand caging, ause more effort is required than at shaft stations.

Rams, operated by steam or compressed air, are often ct against the empty car, which pushes loaded car the ram should be operated by levers interlocking g chairs, so that it cannot be opened unless cage is own.

is made by the Mining Safety Device Co. When cage ight turns crank *A* and opens horns *B*. This allows first same time closes horns *C*, thus holding back second car. are opened and *B* closed by weight *D*. For intermediate is replaced by a hand lever, which is locked in position

deck cages are loaded simultaneously (35) (Fig 106). At rs *a*, checked by horns *c*, approach shaft on a down grade. ting cage *d* is a 4-deck auxiliary cager *e*. While main cage oisted, dumped and lowered, cager *e* is lowered so that each he landing and receives a loaded car. In balance with it is er compartment, which rises to position for transferring its he cager on empty side moves uniformly at all times with hoisting cage is opposite cager, the decks of the latter are rled cars run simultaneously onto cage, pushing off empties. held in position on the cage by spring stops and buffers *h*.

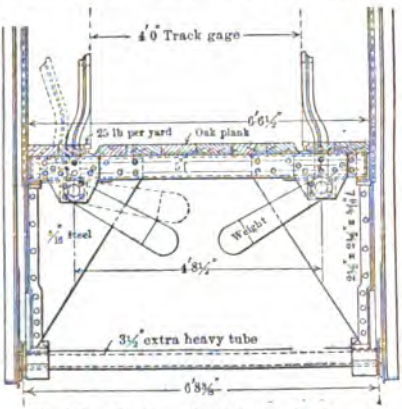

Fig 108. Car Stops for Self-dumping Cages, Jed Coal and Coke Co

(*b*) pair of bent bars, 0.375 or 0.5 by 1.5 in, hinged of cage, and resting in brackets on the diagonals. small clearance over ends of car body.

to car wheels or axles, whenever cage is not on the chairs. They are of 2 kinds: (*a*) HORNS raised in front of wheels, ed by weight and levers when cage leaves landing. Fig 108

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shows stops for a self-dumping cage. For ordinary cages, the horns are often by a lever at corner of cage, within reach of cage tender at landing. (*b*) Drop CAGE. In this an independent transverse section of deck and rails, on which stands, drops a few inches when cage is lifted from chairs. Wheels are thus h fixed rail ends (Fig 109). In Jeffrey drop-rail cage, rails are pivoted at outer

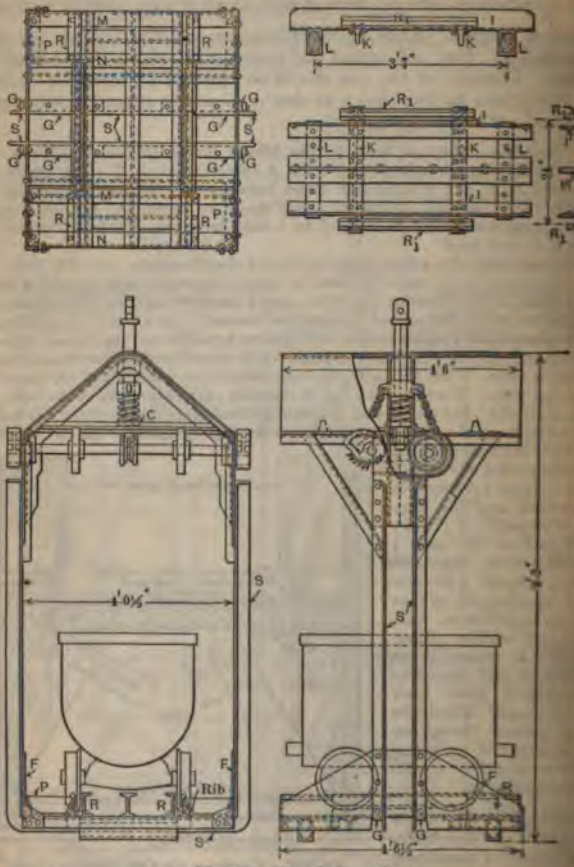
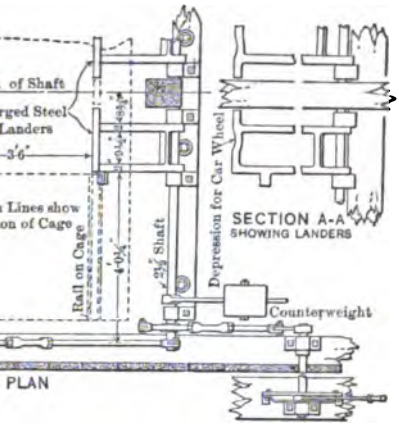
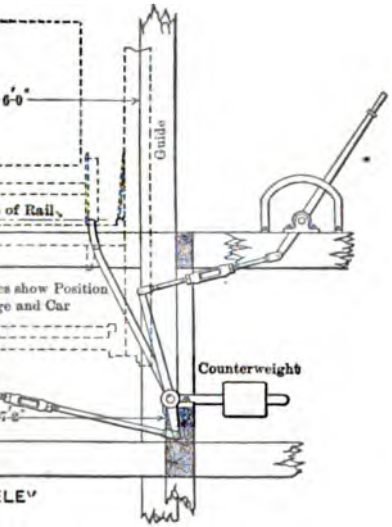


Fig 109. Drop-bottom Cage

divided in the middle. When cage is lifted from chairs, inner ends of the r and car axles fall into notches in a pair of rigid skids. Car is thus held in place w is landed at another station. In another design (Fig 110) two transverse s cage rail and deck, each slightly wider than the car wheel diam, are cut out entir at shaft stations are special chairs or "landers," the tops of which just fill thes the deck. By a hand lever the chairs are thrown in under the cage as it is land lifting car wheels to the rail level.

eps). Fig 111 shows a common form, installed at supported at its corners by 4 fingers *a*, pivoted at *b*, hand lever *d*; which is pivoted at *e*. The cut shows





mping Cage, H. C. Frick Coal and Coke Co (Ketchum)

rs *a*, which are pushed back by the rising cage, are deck by weight *h*. For UNDERGROUND LANDINGS, of *e*, thus always holding the chairs back out of th

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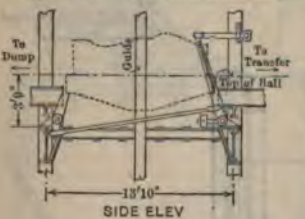
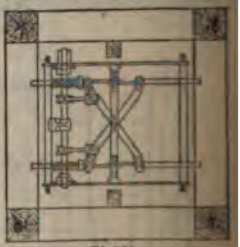
way of a passing cage. To support cage at a given landing, the station t pushes lever *d* into the position shown. A spring may be used instead weight. Especially designed chairs, with 8 fingers instead of 4, are required for drop-bottom cages.

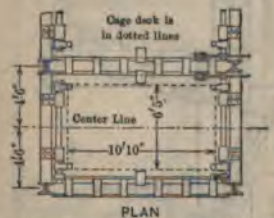
Fig 112 shows chairs supported by steel construction at mouth of Annabelle shaft, Four States Coal and Coke Co. Fig 113 shows a simple design used by Desloge Consol Lead Co, S E Missouri. Parallel rails are caused to slide in and out by links

Fig 111. Chairs for Surface Landing

Fig. 113 Desloge Landing Chairs (E







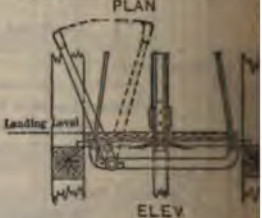


Fig 112. Surface Chairs, Annabelle Shaft, Four States Coal and Coke Co

Fig 114. Gray's Cage Chair

and levers, serving both compartments. CHAIRS ATTACHED TO CAGE DECK (Fig 11 advantages for shafts with many levels, since only one set of chairs is required f cage. But they are not so safe as those at each level, as they may be accidentally out while hoisting or lowering, and they are forbidden in some districts.

26. Skips

Skips may be used in either vertical or inclined shafts (as employ colliery slopes they are called "gunboats"). ADVANTAGES over cages: le required for loading and dumping; large capacity in shafts of small cr

g at shaft top; may weigh less than combined cage
difficulty in keeping separate different grades of
ossible to distribute output into a long storage bin
; men can not be so conveniently raised or lowered;
re danger of ore dropping down shaft; in collieries,
due
oat.
ARE
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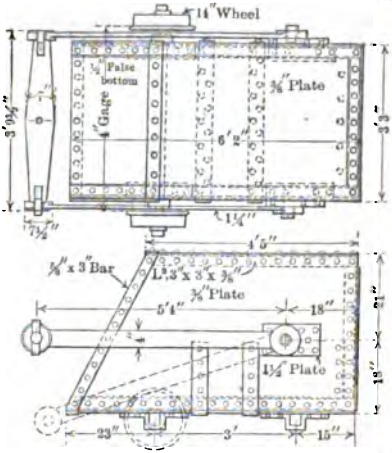

Fig 115. 3.5-Ton Skip, Mineville, N Y (*E & M J*)

xles
on

castings bolted to sides (Fig 117). They must be in accu-
s cost more, but reduce headroom required in shaft com-
below rail heads, derailment is almost impossible. Axles,
laced than trunnions. Wear on wheels and axles is severe,
e of acid mine water; hence, end of axle should be capped
and liberal lubrication provided
for. These features are shown
in Fig 116, illustrating rear wheel
of skip in Fig 115. Manganese
steel wheels are good; or a steel
or chilled tread, with soft cen-
ter bored for a bronze bushing.
Front wheels have treads of
standard width; rear wheels are
extra wide (5 to 6 in) for dump-
ing the skip (see below). BAIL
side bars are attached to lugs
bolted to side plates, as shown;
or better, to pins supported in
double shear by a flat bar, pass-
ing around back of body. Pins

Mineville Skip

ue. Sectional area of metal through center of eyes of bail
er than that of the bar. Length of side bars should be
erfere with loading, as skip hangs on the rope. Crossbar
d at ends and loaded at center with total rope load, includ-
s service is severe, and a large safety factor is necessary.
eutral axis of bar. POSITION OF BAIL LUGS depends on dip
s, lugs are placed near the rear and as low as possible with-
e bars with hubs of wheels. For dips exceeding 60°, lugs
ital center line of skip body; and to secure stability th

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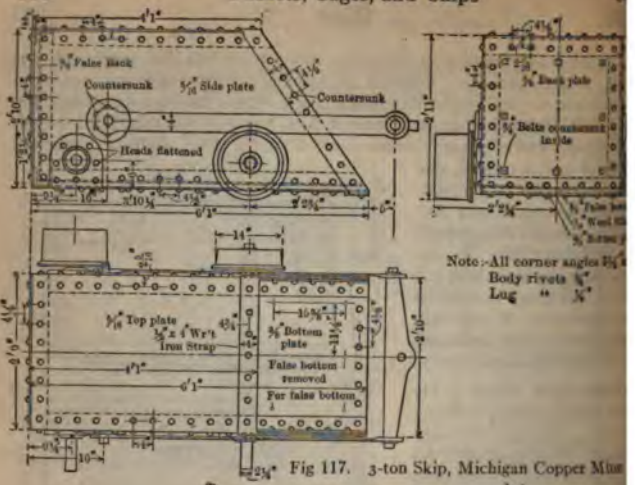

Fig 117. 3-ton Skip, Michigan Copper Min

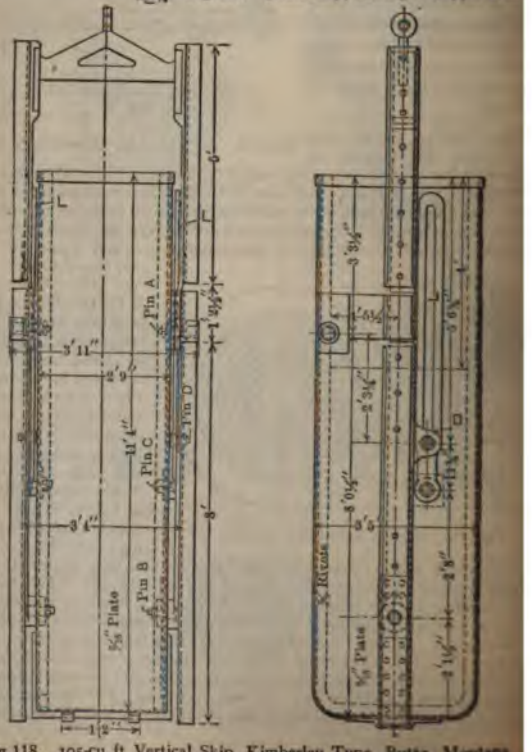

Fig 118. 105-cu ft Vertical Skip, Kimberley Type, Butte, Montana

t Vertical Skip, Copper Queen Mine, Ariz

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should be as far forward as satisfactory dumping will allow. LINERS, due to loading and dumping is on back and bottom of skip, which shou false plates, with wood fillers. For convenience of renewal, these plates riveted (Fig 117), the bolts having countersunk heads. When ore break is great. No satisfactory SAFETY APPLIANCE has yet been devised for inc

Vertical skips. For large outputs from vertical shafts of smal skips are often used instead of cars and cages (see par 1 of this shows KIMBERLEY TYPE of vertical skip. Fig 119 shows a 60-c vided with safety catches, as it is used also for conveying men. Fig designs, up to a capacity of 10 tons, have largely replaced 3 anc formerly used in deep vertical shafts of Butte, Mont, district. as 11 tons capacity are used by De Beers Diamond Mining Co, S

Design of vertical skips is largely a matter of experience and good pract being usually more than taken care of by proper allowance for heavy operation. Unless required capacity cannot otherwise be obtained, L should not exceed twice the least cross-sectional dimensions; otherwise t dumping, especially if the ore is sticky. Fig 118 shows an example of a lo

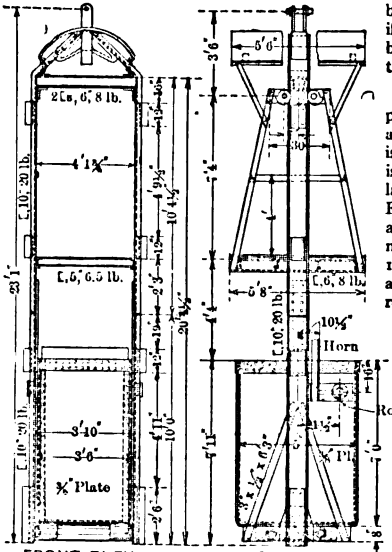


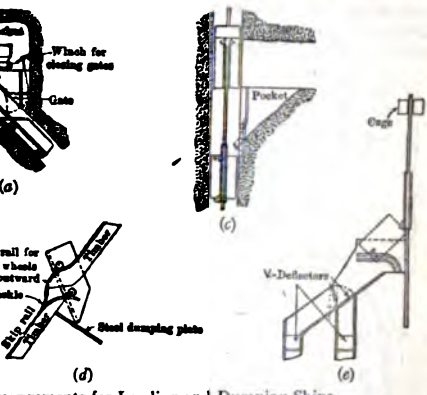
Fig 120. Combined Cage and Skip

Skip of Miami Copper C by 10 ft, capacity 7.5 to if employed, must be hig body to be out of danger to obstruct loading from

Combined cage and providing transit for me also, for sinking operation is to be deepened, provide is small. Fig 120 shows a lar hoisting, as used by F Flat River, Mo; capacit about 6 tons, though 5 nary load. Main verti 10-in, 20-lb channels, splic and skip for convenience renewal of parts.

Arrangements for loading. In INCLINED SHAFTS, skips are from ears, the skip holding 1, 2 or more car-loads; or (Fig 121, *a*) (Art 28), through a gate controlled from landing above. Latte since cars and skip are independent of each other, and on reaching the skip is loaded promptly, without waiting for cars. In VE a skip holding 1 or 2 carloads may also be loaded direct (Fi pockets are preferable for skips holding more than 1 carload (Fi

mping are more varied than those for loading. For uckle" is formed at dumping point by curving main The narrow-tread front wheels run forward on these



rangements for Loading and Dumping Skips

ad rear wheels continue up the regular slope on a pair der gage, beginning at the knuckle.

ion of this typical dump, as or an 8-ton skip. By curv- ils, the skip is brought more gle. Design must be such cannot go down the shaft the whole tread of the wide guard rails may be neces- ip from slipping sidewise. outer part of tread may be nner part, the shoulder so the auxiliary rails.

np at more than one point, s of ore into separate bins. in main rails at dumping wheels pass, are opened or ewise by an attendant, by power for intermittent ser- Copper Range-Mining Co, The wide-tread rear wheels *K*, and when skip reaches front edge is supported by of skip would drop. Points at *A*; and, by a system of ers, may be raised to dotted up the track to a higher

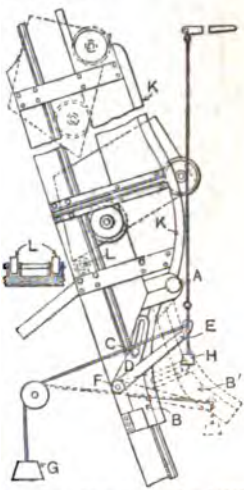


Fig 122. Skip Dump for Intermittent Service (*E & M J*)

e skip body, which is piv- d guide frame, is thrown a dumping angle by a pair of small wheels, near upper ich run into a curved slot-like track attached to head- nt.

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Fig 123 shows dumping track for the skip in Fig 120. Skip body rests on shafts bolted to guide frame, and far enough apart to insure stability of skip w hoisted (Fig 118 and 119). One shaft passes through 2 heavy pillow blocks bolted to skip body, thus forming a pivot on which the body rotates. The o rests in pillow blocks without caps, also bolted to the skip bottom. Track designed that skip body will not reverse, and go down head first when low Kimberly skip (Fig 118), this is prevented by the long links *L*, in which slide t thus limiting the angle through which swing outward. More common devic of roller wheels attached to side plates ing track (Fig 123) in such position horns (Fig 119, 120) projecting forw front end of skip body will just slide o these rollers thus carry weight of s dumping wheels are being lifted acro reversed portion of dumping track, an trip they insure that skip body will right side up. Instead of 2 rollers, a c bar or roller spanning gage of track is s used, but is liable to interfere with fr ing of ore. When dumping wheels are (Fig 119) they should be placed close ing side of skip, so that ends of pivot will have clearance space between the main and dumpin The farther this shaft is from center line of skip, the less th to accidental overturning, but the greater is the force re dump skip, thus throwing excessive stresses on guides. M are subjected to heavy side thrust at dumping point, a be well reinforced and bolted to headframe. Pivoting sh ally placed at 0.33 to 0.5 the distance from center line body, 0.375 being good practice. For small skips the s are ¼ to 5/16 in; front plate, ⅜ in; bottom plate, 5/8 to ½ large skips, the sides are ½ in, and the front and bottom 5/ in some of the Mich copper mines, even heavier.

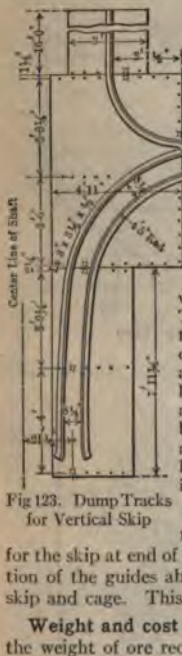

Fig 123. Dump Tracks for Vertical Skip

Handling men in vertical-skip shafts. F numbers of men, a double or triple-deck cage is sub for the skip at end of shift. The change may be made quickly by hingi tion of the guides above shaft collar, and providing a small crane for skip and cage. This method is practiced at Butte, Mont, and elsewhe

Weight and cost of skips. The weight of inclined-shaft skips is ab the weight of ore reckoned at 110 to 120 lb per cu ft. Cost is from 5 per lb at factory. For standard vertical skips Allis-Chalmers Mfg following weights, including dumping guides or cradles:

Table 34. Standard Vertical Skips (Allis-Chalmers Mfg Co)

Capacity, cu ft	Wt, without safeties, lb	Wt, with safeties, lb	Capacity, cu ft	Wt, without safeties, lb	Wt, safet
20	2 200	2 300	70	3 600	4
30	2 400	2 600	80	3 900	4
40	2 550	3 050	90	4 300	5
50	2 900	3 500	100	4 700	5
60	3 250	3 950	.....	.....	.....

Dumping cradles for 20 to 60-cu ft skips weigh 2 500 lbs; for 70 to 1 skips, 4 000 lb. For the 8 and 10-ton skips used in some deep mines, the are greatly in excess of above figures. Vertical skips with safety cate 7 to 8¢ per lb.

27. Overwinding

en for any reason the hoisting engineer fails to check
e to rest at surface landing. In such case, the cage
the sheave and its supporting timbers at top of head-
ne should be sufficient to allow reasonable clearance
ter of sheave, when cage is at landing. This clearance
ed.
less
fer-
rope
ver-
cage
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icts
men
ised
inal
s or

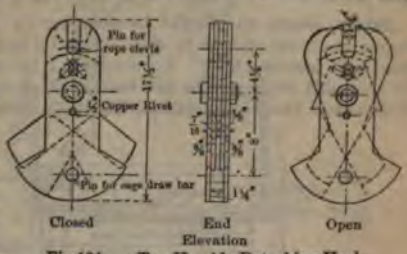
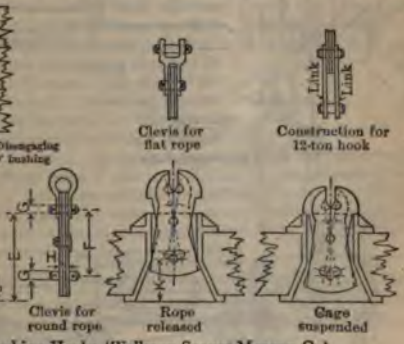


Fig 124. 4-Ton Humble Detaching Hook

is shown in Fig 124. Disengaging plate, into which hook
leasing the rope, is bolted to timbers in upper part of head-
steel, with a round opening of a diam about 0.5 in greater



aching Hooks (Wellman-Seaver-Morgan Co)

of the Ormerod hook (a modification of the Humble) with
-disengaging bushing, are shown in Fig 125 and Table 35.
be placed as high as possible in headframe, and solidly sup-

hing Hooks and Bushings (Wellman-Seaver-Morgan Co)

mensions, in						Shipping wt, lb			
E	*F*	*G*	*H*	*J*	*K*	Cast-iron	Forg-ing	Struct steel	Bolts& rivets
14	10½	1½	1¾	3	7	190	13	39	5
19	14	1¾	1⅞	4	11	325	13	96	8
25	19	2	2	6	13	875	32	183	28
33	23	3	2	9	17	1 325	100	351	67

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ported to withstand shock of cage, if overwound, and then to carry its weigh hook, well-known in Great Britain, is the WALKER. If a plain detaching ho ears or lugs, is used for holding the cage after rope is released, chairs or cat in upper part of headframe. They are so made that cage will pass up th and they are then thrown outward by springs. Detaching hooks do not gr chiefly because they are not effectual for high hoisting speeds. A run-away into the headframe at a velocity of several thousand ft per min, would pro itself and also the disengaging bushing, with its timber supports.

Safety stops, to prevent overwinding by automatically cont engine, generally operate by closing throttle and applying brakes. center the reversing links. Requirements: (*a*) simplicity; (*b*) all p be accessible for inspection and testing; (*c*) derangement or failure should cause the stop to operate; (*d*) steam should be cut off close t

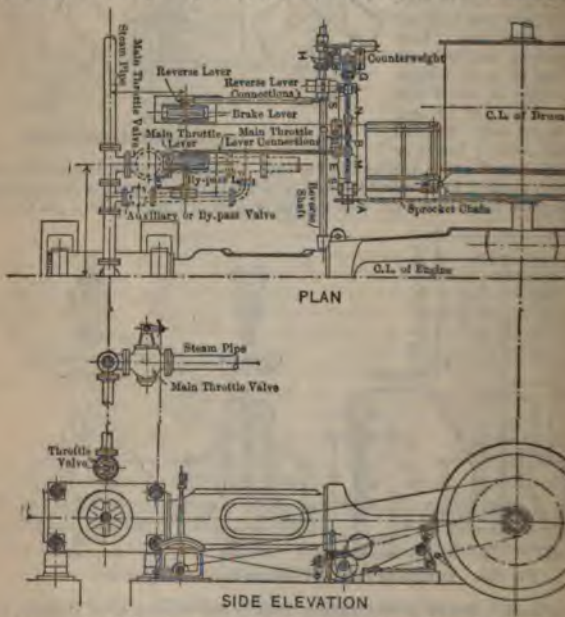
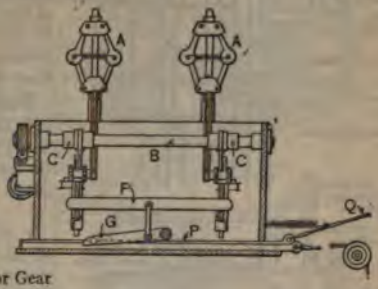


Fig 126. Automatic Hoisting Engine Stop (*Trans* L S M I)

chests; (*e*) steam should be throttled as cage approaches surface, a applied for bringing engine gradually to rest; (*f*) the apparatus sh into operation automatically, if the engineer fails to check the eng proper time; it should also be self-locking, so that engine can no started until gear has been thrown off and engine reversed.

Fig 126 shows safety stop, made by Sullivan Machy Co. It consists of closing device, operating on each trip at a predetermined point, without u engineer. Point of closure is such that only a slight use of the brake is requ the cage at proper point. To compensate for a heavier load or lower steam small by-pass valve admits enough steam to land the cage. The apparatus the drum in such manner that if one drum is unclutched the stop operated

ay proceed with the other (7). A threaded shaft *S*, sup- plate *E*, is driven from the drum by sprocket *A*. When endwise through bevel gears *H* and links *G*. This motion *N* to lift roller *B* and lever *C*, thus gradually closing throttle pen throttle again the engine must be reversed. FUTER's on the cutoff, shortening it as the cage rises above a fixed



Visor Gear

he valves at a predetermined the brakes. Its best features y at the cylinders, and grad- e safety stops are operated by releases a latch allowing d apply the brakes; but their with high speeds. "VISOR" gan, England) is shown in its Governors *A* and shaft *B* are m shaft. *B*, making approxi- ies the cams *C*. When speed levers *E* with their hooks *R* f cams *C*. If steam is not shut off at the proper point the ting frame *F* and pawl *G*, thus allowing weight *K* to close *P* and *Q* are parts of an auxiliary stop, to prevent starting By providing cams *C* with several beaks of increasing radii, ally as the cage nears the point where the throttle should for electric hoists see Sec 16.)

lition to safety stops are a necessary auxiliary for high- They are usually of the flyball type. With Corliss ening the cutoff; with slide and piston valves they

ETS AND SURFACE ORE BINS

28. Shaft Pockets

ng stations decrease time required to load a skip, and ty, so that hoisting and tramming are not directly . Filling a skip from a pocket also involves less own the shaft. CAPACITY of pockets depends upon re cars or trains of cars, and storage needed at the ammed while hoisting is being done from other levels. nore. As a rule, a measuring pocket is placed below olds one skip load and is filled from main pocket, thus danger of overfilling skip, if gate should stick or break.

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This also permits dividing main bin into 2 or more pockets fo
ore; then, with swinging chutes between main-bin gates and

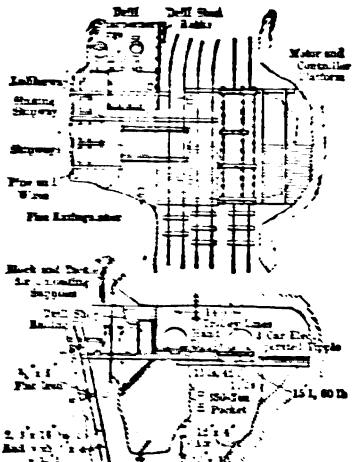


Fig 130. Pockets at 340-ft Level of Joker Mine (40)

any grade of ore
either skip. St
ets depends on ca
rock in which th
of the shaft. In
or steel only sh
perhaps a few la
front wall, lining

Pockets for
cept the measur
vated in the ha
timbering requir
gates.

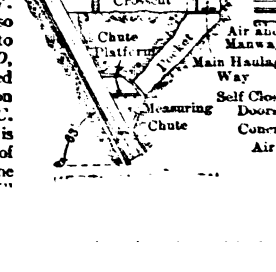
Ex

I. A 650-ton po
of Joker iron mi
shown in Fig 128
The magnetite or
skips. Three car
dumped simultane
revolving tipple.
ing chutes are ope
air cylinders. An
hand-operated tip
is placed near sta
main pocket is be

II. Proposed st
storage pocket for
northeastern N Y
Pocket is entirely

III. At Palme
Franklin Furnace
measuring pocket
steel (Fig 130).
chute are moved
taking its water f
the shaft. The

apparatus. Lower door is closed and upper undercut gate is op
measuring bin. When the angle of repose is reached the flow s
upper chute full. To load skip,
the ram moves out along the arc
shown until edge of undercut
gate is at A'. At the same time
rod G moves to position G'.
This moves levers B and C so
that pins H and I advance to
positions H' and I'. Door D,
held in position by arms pivoted
at O and E, is forced to position
D' by the movement of B and C.
The advance of cutoff gate A is
at first more rapid than that of
D. D closes first, so that the



ding chute and gate at Victoria shaft, Phœnix, B C. Load-
in pocket, without intervention of a measuring bin. The
bottom of chute are lined with castings. Gate is of finger

t Pockets, Franklin Furnace, N J (*E & M J*)

iron gate below fingers to catch fine ore. Fingers are raised
pper end; as they close by their own weight, and each finger
rs, and loose on the pivoting bar, large lumps of ore do not
o as
bars
ead-

In
con-
used
ined

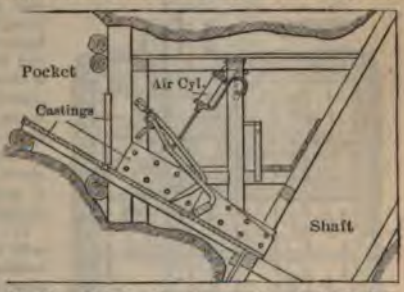


Fig 131. Skip Loading Device, Victoria Shaft, B C

lich,
ves 4
132)
es on
were
ed. The roof and floor over bins are supported by 70-lb
width of shaft and station between the compartments. Roof
downward; floor rails, with flanges upward. At station
as posts, attached by connecting angles to top and bottom
tion only is first cut out, and then the triangular space for
the shaft below the landing. Bin floor is formed by rails
Main gates are corrugated plates; they are raised by air
r own weight and weight of crosshead and piston. Gate
uides. Inner corrugated surface of gates decreases friction
opening. For loading the 8-ton skips, the chutes are fitted
l lowered by hand wheels. Measuring pockets are not used.
mine, Mich, is circular in section, with 2 skip compartments
34) are 12 ft 6 in wide, 18 ft 6 in deep, and 23 ft 3 in long

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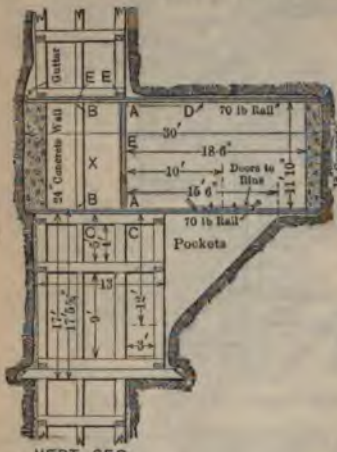
at the top; divided into 3 compartments for 2 grades of ore. Center compartme
2 gates, for loading into either skip. End compartments have 1 gate, and load entr
skip. Front wall of the pocket is supported by horizontal timbers, lined with 3 h
vertical timbers, covered with 0.5-in steel plate near bottom and 0.25-in plate a
Sides are concreted, and lined with 0.5-in and 0.25-in plates bolted to timbers emb
in the concrete; plates are put in pl
and concrete poured behind them
making forms unnecessary. Bottom
creted and lined with 2 layers of 3-in
wood planks, spiked to 5 by 7-in ti
embedded in the concrete. This
also is placed before concrete is p
thus serving as forms. Gates are v
finger bars, about 4 in square, suspen
chains from a crosshead attached to
of an air cylinder with 3-ft stroke.
measuring pocket holds a skip load.
gates, of 2 thicknesses of 0.5-in plate,
terweighted, and opened and closed
air cylinder, open away from the sh
that no part shall project into the ho
compartments. Ore drops through
tionary chute into skip. Measuring
ets are of steel plate, their fronts
supported by I-beams across the ope
in the concrete shaft lining.

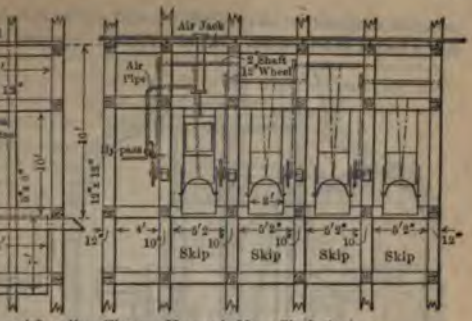


III. At the Grangerberg iron
Sweden, side-dump cars are tipped
loading chute by a long hook operat
a hydraulic cylinder controlled thro
3-way valve. Pocket holds 1 skip
only, as door can not be closed unle
is empty. Gate is moved by a hori
hydraulic cylinder, and a system of
and links. Position of gate when
is shown by heavy lines; when ope
dotted lines. As gate opens, a hing
is also raised to bridge gap between
chute and edge of skip. All opera
are controlled by hand levers on pla
above. An indicator shows whether
is open or closed, and if open pre
dumping ore into pocket (Fig 135).

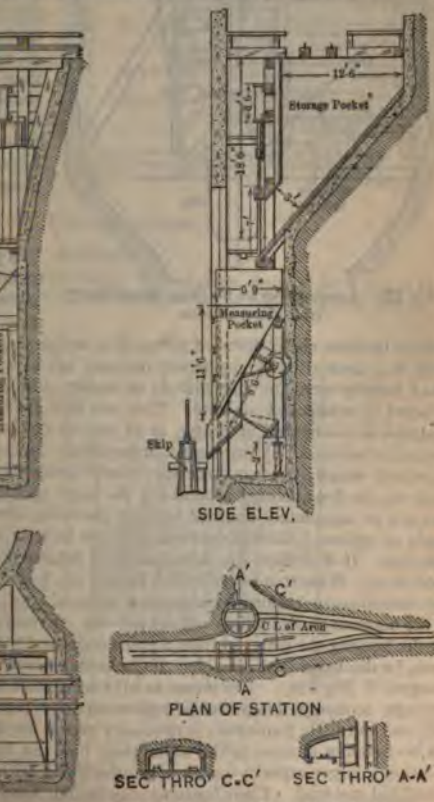
IV. Fig 136 shows mode of load
skip at Chapin mine, Mich. P
holding 5 tons (1 skip load), is of
plate, and slightly narrower than the
Over the front is a hood hinged at
and connected by a bar at bottom
pin *b* on a 3-ft hand wheel. Closed
tion is at *a*; open at *a'*. When ho
closed, crank pin *b* is below the ce
against stop *c*, and pressure of ore i
pocket can not open the hood. On re
ing the wheel, *b* rises above center
weight of ore forces hood open.
pocket is empty, weight of the hood makes it self-closing. The 5-ton skip is load
3 sec. Axle of the handwheel extends the width of the hood, with crank and conne
rod to the hood on each end.

Fig 132. Hancock No 2 Shaft Station (41)

Stresses in shaft pockets. Pressures on walls are computed as for su
ore bins (Art 29). But, because the service conditions are severe, and



and Loading Chutes, Hancock No 2 Shaft (41)



s and Storage Pockets, Negaunee No 3 Shaft (42)

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struction may not be so carefully carried out as for surface structures safety factor is advisable. If bin structure supports also the walls of

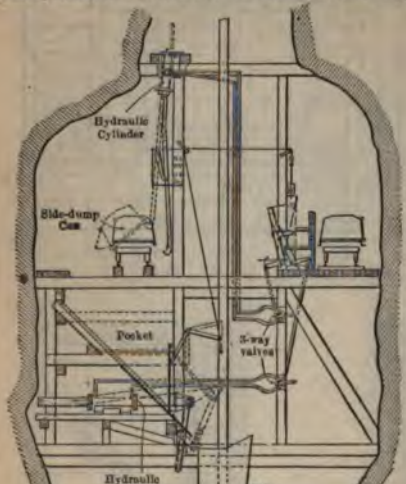


Fig 135. Loading Device for Skip, Grangerberg Iron Mine, Sweden

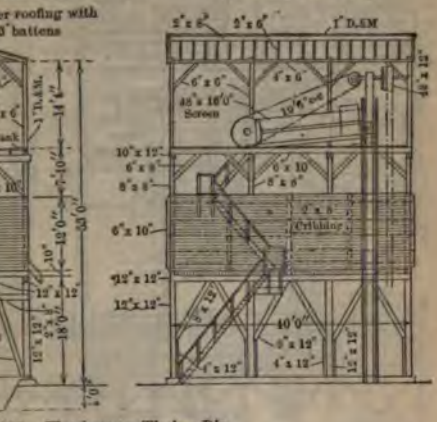
Fig 136. Loading Devic Chapin Mine

vation, this fact wou ally be the determini in design. For deep excavated in ore or gate or chute and it should be carefully since it may be sub unusual loads due to ing above and then fal some height.

29. Ore Bin

Types. For ore, stone and coal, bins signed: (*a*) of rectan tion, with flat bottom hopper bottom, either inverted pyramid or wedge-shaped; (*c*) of circul with flat, conical or hemispherical bottom; (*d*) as a suspension bunk FLAT BOTTOM BINS are built entirely of timber, steel frame with timber all steel, or reinforced concrete. They are best suited to conditions material is discharged at center, or at various points, of the bottom, veyers or into cars. Advantages: larger storage capacity for the same f occupied; simplicity of construction; the ore forms its own bottom and wear on the floor, except around the discharge spouts. Disadvantag can not be completely emptied without shoveling the material lying angle of repose. HOPPER-BOTTOM BINS are built of timber, steel, or concrete. If of timber, the sloping sides of hopper portion should be steel plate. If bottom is pyramidal, the bin can be discharged complete the vertical cross-section is triangular or wedge-shaped, there will always of ore between chutes which will not run out. Pyramidal bottoms are for discharging the material from a point underneath the bin; wedge-sh toms, for discharging at one side. If discharge on each side is desired, bo inverted V (Fig 138). CYLINDRICAL BINS are of steel or reinforced concr vantage: economy of material for a given capacity, because the walls are only and but little framework is necessary except for the substructure. are no bending stresses in the walls, these bins can be made much de securing a larger capacity on same ground space, with same or a less q structural material. Their bottoms may be flat, hemispherical or conic PENSION BUNKERS are of steel plate or reinforced concrete. The bi

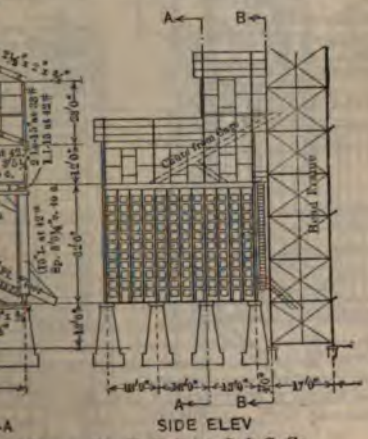
, which in turn are supported by columns. Weight assume form of the equilibrium polygon (Sec 36), es in plates are tension only. As form of the curve es of loading, the plates are slightly distorted; and einforcing, or steel framework supporting the rein- the concrete serving only to protect the steel.



137. Flat-bottom Timber Bin

Examples

m timber bin for crushed stone, designed for R R cars to g. Walls and partitions are of 2 by 8-in plank, laid flat



-A
oal Storage Bin, H. C. Frick C. & C. Co

ons are interlocked with side walls by making alternate walls are supported by tie rods and buck stays. Bottom

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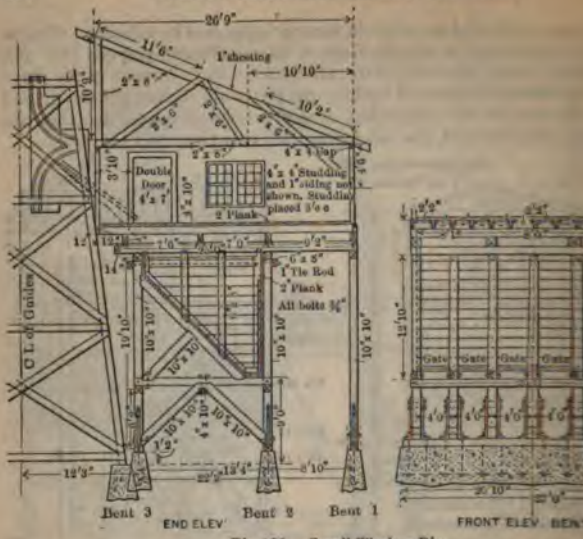


Fig 139. Small Timber Bin

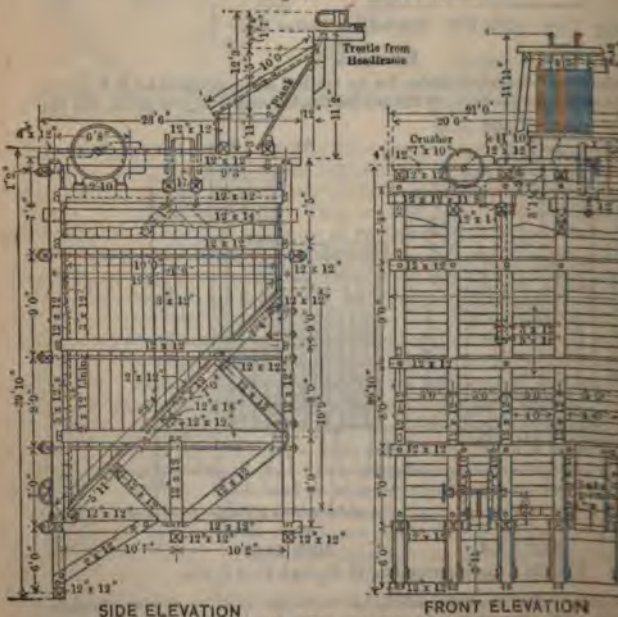
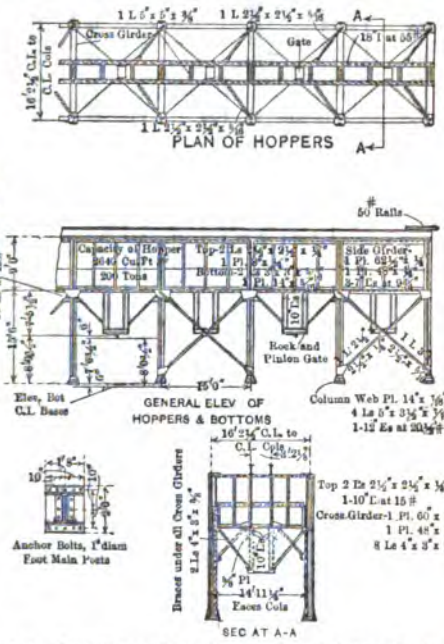


Fig 140. Bin with Grizzly and Crusher

dge. 47 000 ft B M of lumber were used, of which 35 500 ft ts supports, and 11 500 ft for superstructure, screen floor, 1 000 cu ft, or 550 tons of crushed stone; equivalent to apacity, or 3.2 ft B M per cu ft capacity.

for coal storage, at Phillips mine, H. C. Frick Coal and Coke The coal is discharged into the bins from self-dumping 0.25-in buckled plates, supported by 15-in, 42-lb I-beams, Inclined bottom is of 3/8-in plate, its framing consisting of -in web and flanges of 6 by 6 by 7/16-in angles. Girders are d tied together by 2 angles 8 by 8 by 0.75-in and 1 plate l 15-in, 42-lb I-beams at the top. Main side girders consist



141. Steel Hopper Bins, Cananea Consol Copper Co (43)

f onto a conveyer, through rack and pinion gates in bottom ed into 8 pockets, approximately 15 ft square; capacity of ns. Side plates are 0.25-in, stiffened with 7-in channels, lates are 3/8-in, stiffened with 10-in channels (43).

pee, Sherman & Co, Mineville, N Y (44), a cylindrical rein- r mill storage. Ore is magnetite, of 4-in size, delivered by ttom is carried on a concrete arch, which houses the feeder harge from bin. Capacity, 1 000 long tons. Outside diam, diam is stepped in, from 2 ft thick at bottom to 1 ft at top. tions, of 3/4-in matched lumber. The reinforcing is 1 1/8-in placed 1 ft apart for vertical reinforcing and 2 ft apart for

s are used in the rock house of the Quincy Copper Mining ock bin is 44 ft diam by 35 ft 5 in high. Lower 28 ft is

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5/16-in plate, upper 7 ft of No 10 sheet steel. Bin for receiving copper rock 14 ft diam, of 7/16-in plate. Poor rock bin is 13 ft diam by 12 ft 6 in high, of All bins discharge through side chutes into cars, the ore taking natural an and forming its own bottom.

VIII. Cement storage plant of Illinois Steel Co, Chicago, consists of 4 cyli forced-concrete bins constructed on the Monier system (43). Each bin is 2 53 ft 6 in high, the base 9 in above floor. They in a square and spaced the space between the occupied by 30-in circ The bins are carried o 10 in thick by 3 ft 5 in wide, reinforced by 4 to These piers rest on a 66 ft square by 3 ft thic by a 9-in mesh of 5/8-i Concrete girders cappi supporting bins are re steel rods and netting. of bin walls is 7 in thick 5 in; all reinforced by netting of No 9 wire, w mesh. Around the ne nately inside and outsi zontal rings of 3/4 to 1-i apart. Bottom of each with the point upward 18-in openings for disch around the circumferen base being 22 ft. Bot thick, reinforced. Conc dation and piers is 1 cem

Fig 142. Reinf Concrete Bins, Croton Iron Mines (45)

sand, 4 broken stone; in bin walls, 1 cement, 3.5 sand, no stone. Capacit space between bins, 25 000 bbl of cement.

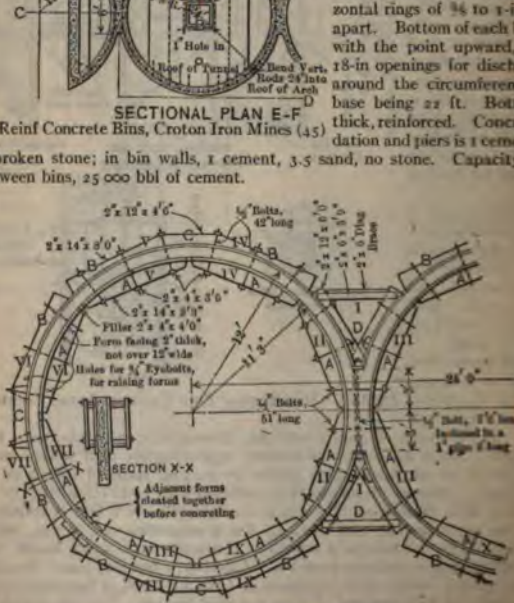
Filler 2" x 4" x 4'0"
Form facing 2" thick, not over 12" wide
Holes for 3/4" Eyebolts, for raising forms
1" Bolts, 51' long
SECTION X-X
Adjacent forms elected together before concreting

Fig 143. Forms for Bins in Fig 142

IX. A set of 4 cylindrical concrete bins, built 1914, at Croton iron mines, Bi (45), are arranged in line (Fig 142). The bins are 22 ft 6 in inside diam by 34

500 tons. Ore is withdrawn through hoppers onto belt nels under each bin. Bin walls are 9 in thick, reinforced 8 in apart, and horizontally by old 0.75-in cable, spaced n apart for 10 ft, and then 9 in apart to the top. Forms n 2-in chestnut. They were made by nailing face boards cut to proper radius. Inner and outer forms were held s, long enough to pass through 2 by 4-in vertical bolting bolts being greased and withdrawn, were used over and ld 25.5 cu yd of concrete, and was filled in 10 to 11.5 hr d fireman for the hoist. A 0.1-cu yd batch mixer was one end of line of bins to scaffolding above the form. ; walls, 1 : 2 : 4. Coarse aggregate of 2.5-in stone was ul tamping.

ning and Smelting Co, Ariz, has 8 cylindrical steel bins ix 17 ft 3 in deep; diameter of all, 15 ft. Bottoms are

Corr Steel
1 L 2" x 2" x 5/16"
2 L 2" x 2" x 5/16"
2 L 2" x 2" x 5/16"
13'0"
22 Corr Steel
8'1"
4 Pls. 12½" x ¼"
2 L 3" x 3" x 5/16"
2 L 3" x 3" x ¼"
1 L 2½" x 2½" x ¼"
4 Pls. 12½" x ¼"
2 L 3" x 3" x 5/16"
2 L 2½" x 2½" x ¼"
4 Pls. 12½" x ¼"
1 L 2½" x 2½" x ¼"
4 Pls. 12½" x ¼"
2 L 2½" x 2½" x ¼"
Pls. ½" x 12" x ¼"
2 L 2½" x 2½" x ¼"
10'4"
35'11"
18'01⅛"
5'1½"
4'0" abt.
12'1"
10'7¾"
6'4"
1 L 2½" x 2½" x ¼"
2 L 3" x 2½" x ¼"
1 Pl. 10" x 5/16"
1 Pl. 13"
than
loor

Ore Bins, Old Dominion Copper Mining Co (43)

le plates vary from 5/16 to ¼ in. Bins are connected and by 4 columns at 90° apart, riveted to the sides. Col- lb channel next to the bin, with 4 angles 3 by 3 by 5/16 in annel, plate and 2 angles extend about 40 in above the m, the other 2 to top of bin. Bins are filled from a belt and discharged at the bottom through swinging cutoff

ower house, N Y, has a suspension coal bunker with a . Length from center to center of end columns is 65 ft, ates are 0.25 in, lined with 2.5 in of concrete, reinforced with a 1-in granolithic coat. Angles 2.5 by 2 by 0.25-in, lates, are spaced 4 ft apart horizontally. Bunker is filled

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from a belt conveyer. Construction details of main members and plates are s
Fig 145 (43).

XII. Suspension bunkers for ore and limestone storage, Lackawanna Steel Co, N Y, are divided into bins 25 ft long by partitions of ⅜-in plate, supported by 10 I-beams placed vert 2.5 ft centers. Side in plate, unlined. T center is 16 ft; widt Bunkers are suspend plate girders, consi four 6 by 6 by 0.75-i and 1 plate 42 by These girders also ca for a traveling gant a gage of 26 ft whic the bins, and 2 standi tracks are carried on cr ers at panel points. to top of railroad trac Bins and tracks are su by posts spaced 25 ft made up of 4 Zs 6 by web plate 8 by ⅜-i 2 cover plates 14 b Each 25-ft bin is div through 4 bottom ga 2 ft by 2 ft 10 in.

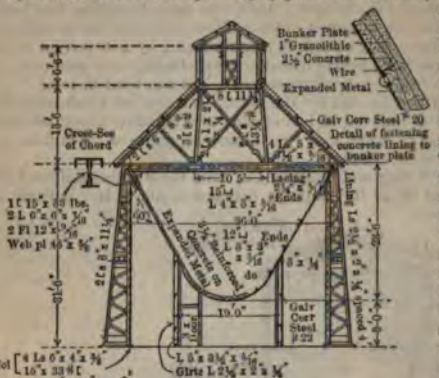


Fig 145. Coal Bunkers, Rapid Transit Subway, N Y (43)

loaded the stress in the plates is 17 000 lb per lineal ft. Openings are provided in th just above discharge spouts, so that the mater al can be loosened up if it should b

Cost of timber bins. Labor, nails, bolts, and iron work, exclusive plate lining, cost $20 to $25 per 1 000 bd ft, to which is added the timb

Cost of concrete bins. Four bins of the Croton iron mines (Exam cost, after crediting the future value of the form lumber:

For foundations, 221 cu yd at $3.81	$ 842
Structure above foundations, 380 cu yd at $9.76	3 720
Total	$4 562

This includes labor and material, lumber and reinforcement, but not the cost of tion, nor charges for superintendence and depreciation of plant.

A 400-ton reinforced-concrete coal pocket, at the Atlantic City (N J) water 30 ft diam, with conical bottom and pyramidal roof. Concrete was 1 cement, 2 5 gravel. Plain reinforcing bars were used. Excavation, 233 cu yd; concrete, 31 steel for reinforcing, 13 700 lb; steel beams, plates, etc, 3 250 lb. Contract price

A group of 4 reinforced concrete bins for sand storage, with a total capacity of yd, required 680 cu yd of concrete and 4 510 lb of steel reinforcing. Labor, 80 days for 11 carpenters and 14 laborers.

Cost of steel bins comprises: (*a*) cost of material; (*b*) cost of fabri (*c*) cost of erection; (*d*) cost of transportation. Cost varies with local tions, and largely with the design. As a guide for preliminary estimat following figures in ¢ per lb may be used:

(*a*) Material	1.8
(*b*) Fabrication, including drafting, mill details and shop labor, painting, and transportation from mill to shop	2.1
(*c*) Erection	1.0
Total cost, exclusive of freight from shop to point of erection	4 9¢

According to M. S. Ketchum (43) the costs of bins of different materials and d as follows: Wooden bins cost about half as much as steel or concrete. Ste

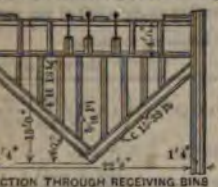
her types, costing 50 to 70% as much as rectangular bins. cheaper than rectangular. Reinforced-concrete bins cost steel bins of same type.

m receiving bins (Fig 146), at the smeltery of the n, Ariz, is given in Table 36 (46).

36. Cost of Steel, V-Bottom Bins

abor	Material	Total	Quantity	Total unit cost
23.11	$ 39.16	$2 342.27	1 428 cu yd	$1.64
35.51	2 247.70	3 483.21	612.3 cu yd	5.69
.....	.....	29 276.63	353.09 tons	82.92
01.15	1 984.93	2 886.08	30 gates	96.20
10.92	2 947.19	3 258.11	97.3 ft	33.49
55.19	2 498.03	2 853.22	117.3 ft	24.33
50.87	24.67	85.54	22 drops	3.89
.....	.....	$44 185.06	.....	.....

ctures: coarse-ore bin, 25 by 70 ft, divided into 4 pockets; divided into 6 pockets. Fig 146 shows a cross section of



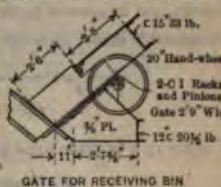


Fig 146. V-Bottom Bins, Ariz Copper Co (46)

gh the stream, and 16 for the concentrate bin cutting down- centers, has a 30-in belt, and includes traveling feeder, motor xcept steel frame for supporting the idlers. Conveyer No 2, lt, and includes the same items as No 1 conveyer.

38. Stresses in Bins

walls depend upon the weight per unit of volume, ure content of the bin filling, and the angle of friction alls.

the walls are flat, and the plane of rupture cuts the ts, the theory and formulas for retaining walls apply. heory, the plane of rupture forms with the bin wall a st," which exerts pressure on the wall. For a vertical r heaping of the material above the top, the plane of between the wall and the plane of repose of the filling, ust is normal to the wall. In this case, the resultant s making an angle with a normal to the wall equal to en the filling and the bin wall. Rankine's theory, on hat the direction of the resultant thrust on a vertical e top surface of the filling; also, that the angle between tant thrust and a normal to the wall is never greater the bin filling. In both methods of solution the point tant stress is assumed at one third the height of wall,

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Algebraic methods. Factors in the formulas for pressure

P = resultant pressure per ft of length of wall
N = total normal pressure per ft of length of wall
ϕ = angle of repose of bin filling
ϕ' = angle of friction of the filling on the wall
θ = angle between plane of wall and the horizontal measu
δ = angle of surcharge
ε = angle between direction of P and a normal to wall
λ = angle between P and the horizontal
h = vertical height of wall
w = wt of bin contents per cu ft

Rankine's formula, for a vertical wall without surcharge, is

$$P = \frac{1}{2} wh^2 \frac{1 - \sin\phi}{1 + \sin\phi}$$

and for a vertical wall with surcharge δ:

$$P = \frac{1}{2} wh^2 \cos\delta \frac{\cos\delta - \sqrt{\cos^2\delta - \cos^2\phi}}{\cos\delta + \sqrt{\cos^2\delta - \cos^2\phi}}$$

If $\delta = \phi$, then $$P = \frac{1}{2} wh^2 \cos\delta$$

In Eq 40, 41, and 42, the direction of P is assumed parallel to upp
contents, and its point of application at one-third of the vertical hei

Coulomb's theory gives the following formulas:

$$P = \frac{1}{2} wh^2 \frac{\sin^2(\theta - \phi)}{\sin^2\theta \sin(\theta + \varepsilon)\left(1 + \sqrt{\frac{\sin(\varepsilon + \phi)\sin(}{\sin(\theta + \varepsilon)\sin(}\right.}$$

If, in Eq 43, ε be made equal to zero, $P = N$ and

$$N = \frac{1}{2} wh^2 \frac{\sin^2(\theta - \phi)}{\sin^3\theta \left(1 + \sqrt{\frac{\sin\phi \cdot \sin(\phi - \delta)}{\sin\theta \cdot \sin(\theta - \delta)}}\right)^2}$$

For a vertical wall $\theta = 90°$, and Eq 44 reduces to

$$N = \frac{1}{2} wh^2 \frac{\cos^2\phi}{\left(1 + \sqrt{\frac{\sin\phi \cdot \sin(\phi - \delta)}{\cos\delta}}\right)^2}$$

For a level top surface $\delta = 0°$, and Eq 45 reduces to

$$N = \frac{1}{2} wh^2 \frac{1 - \sin\phi}{1 + \sin\phi}$$

which is identical with Rankine's formula (Eq 40) for a vertical wa
of filling material level with the top.

Cain's formulas assume that $\varepsilon = \phi'$, Eq 47 to 61 applying also
addition to previous nomenclature, N' = normal pressure on the wa

(*a*) Vertical wall, surface of filling level, $\varepsilon = \phi'$,

$$P = \frac{1}{2} wh^2 \frac{\cos^2\phi}{\cos\phi'\left(1 + \sqrt{\frac{\sin(\phi + \phi')\sin\phi}{\cos\phi'}}\right)^2}$$

$$N = P\cos\phi'$$

If $\phi = \phi'$, $$P = \frac{1}{2} wh^2 \frac{\cos\phi}{(1 + \sin\phi\sqrt{2})^2}$$

If $\phi' = 0$, $$N' = \frac{1}{2} wh^2 \tan^2\left(45° - \frac{\phi}{2}\right)$$

$= \delta$, $s = \phi'$,

$$\frac{\cos^2\phi}{\cos\phi'\left(1+\sqrt{\frac{\sin(\phi+\phi')\sin(\phi-\delta)}{\cos\phi'\cdot\cos\delta}}\right)^2} \tag{51}$$

$$\delta = \phi, \quad P = \frac{1}{2}wh^2\frac{\cos^2\phi}{\cos\phi} \tag{52}$$

$$N = P\cos\phi' = \frac{1}{2}wh^2\cos^2\phi \tag{53}$$

$$\phi = 0, \quad N = \frac{1}{2}wh^2\cos^2\phi \tag{54}$$

$< 90° + \phi'$, $\delta = 0$,

$$\frac{\sin^2(\theta-\phi)}{\sin(\phi'+\theta)\sin^2\theta\left(1+\sqrt{\frac{\sin(\phi+\phi')\sin\phi}{\sin(\phi'+\theta)\sin\theta}}\right)^2} \tag{55}$$

$$N = P\cos\phi' \tag{56}$$

$< 90° + \phi'$, surface surcharged,

$$\frac{\sin^2(\theta-\phi)}{\sin(\phi'+\theta)\sin^2\theta\left(1+\sqrt{\frac{\sin(\phi+\phi')\sin(\phi-\delta)}{\sin(\phi'+\theta)\sin(\theta-\delta)}}\right)^2} \tag{57}$$

$$N = P\cos\phi' \tag{58}$$

$> 90° + \phi'$, $\delta = 0$,

$$\frac{1}{2}wh^2\sqrt{\tan^2\theta+\tan^4\left(45°-\frac{\phi}{2}\right)} \tag{59}$$

$$N = P\cos\phi' \tag{60}$$

nponent of P parallel to wall, the thrust in plane of wall is

$$T = P\sin\phi' \tag{61}$$

eding formulas for shallow bins do not apply when a e of rupture of contents cuts the bin walls. Following Ketchum (43):

$$V = \frac{wR}{k\mu'}\left(1-e^{-\frac{k\mu' h}{R}}\right) \tag{62}$$

$$L = \frac{wR}{\mu'}\left(1-e^{-\frac{k\mu' h}{R}}\right) \tag{63}$$

us notation,

f filling, lb per sq ft
filling, lb per sq ft
+ circumference of bin in ft = "hydraulic radius" of bin
ny point, ft
ction of filling on filling
iction of filling on bin walls
on character of filling
2.71828

may be calculated from $k = (1-\sin\phi)\div(1+\sin\phi)$

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Constants applying to materials stored in bins are tabulated from sources in Table 37.

Table 37. Constants Applying to Materials Stored in Bins

Material	Wt, lb per cu ft	Angle of repose ϕ, degrees	Angle of friction ϕ'	
			Steel plate	Wood
Bituminous coal	47–56	35–40	18	35
Anthracite coal	52–56	27–30	16	25
Slaked coal	53	38–45	.....	.....
Coke	23–32	45	25	40
Ashes	40–45	40	31	40
Ore, soft iron	.....	35	.....	.....
Ore, various	100–140	38–42	15–20	38–40
Crushed shale	90	39	.....	.....
Portland cement	103	30–35	.....	.....
Sand	90–120	34	18	30

Suspension bunkers. The stresses are due to a load which varies from at the support to a maximum at the center, and the loading varies nearly ordinates to a straight line (Fig 147).

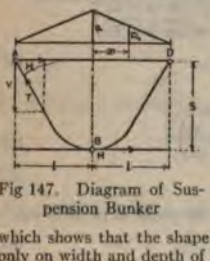


Fig 147. Diagram of Suspension Bunker

P = maximum load at center of bin, lb
l = one-half bin span, ft
S = depth of bin, ft
w = wt per cu ft of bin filling
T = maximum tension in plate per ft of length
C = capacity of bunker in cu ft per lineal ft of
B = lowest point of bin and origin of coordinat

The equation of the curve of the bunker is.

$$y = \frac{1}{2}\,\frac{S}{P}\left(3x^2 - \frac{x^3}{l}\right)$$

which shows that the shape of the curve is independent of maximum load and de only on width and depth of bin.

Capacity of a bunker level full is $C = \frac{3}{4}\,lS$

Maximum pressure P at center is calculated as follows:

$Pl = Cw$, $P = Cw \div l$, and from Eq 65 for a bunker level full, $P = \frac{3}{4}\,Sw$

Tension in the plates at supports A and D is

$$T = Cw\sqrt{\frac{1}{4} + \frac{P^2}{9\,S^2}}$$

Length of bin curve is not exactly determinate. If L be the length for one-half the

$$L = \int_0^l \sqrt{dx^2 + dy^2}$$

By substituting values of dx and dy from Eq 63, the length of one-half the curv

$$L = \frac{1}{2\,P}\int_0^l \sqrt{4\,P^2 + 9\,S^2\,(2\,xl - x^2)^2}\,dx$$

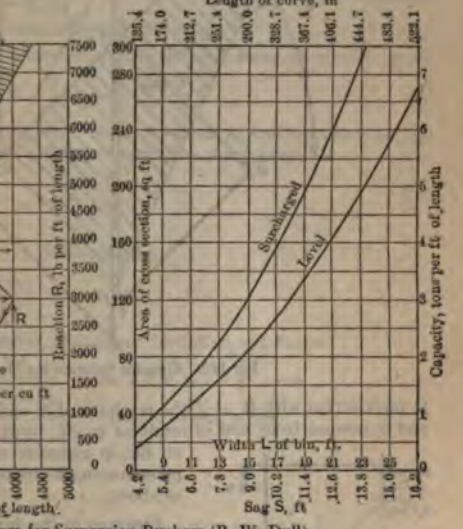
By using Simpson's rule for approximate integration, with 10 divisions, one-half length of curve becomes

$$L = \frac{1}{2\,P} \times \frac{l}{30}\,[y_0 + y_{10} + 4\,(y_1 + y_3 + y_5 + y_7 + y_9) + 2\,(y_2 + y_4 + y_6 + y_8)]$$

$$= \sqrt{4P + 9S^2(2xl - x^2)} \qquad (70)$$

= 0; y_1 = value of Eq 70 for $x = l \div 10$; y_2 = value of on. Ten divisions give sufficiently accurate results for

for calculating stresses, capacity, and length of curve of Fig 148. It is for coal weighing 50 lb per cu ft. For ltiply the stresses in the diagram, and the capacity per eight of material per cu ft to 50 lb. To use the diagram, of coal per ft of length is required, and that the bin is to am at 5 tons on the right and pass to the curve for sur- d the corresponding depth, 14.55 ft, and width, 19.25 ft, 72.2 in. Passing on to the left, the reaction R is 5 000 lb, o lb with the diagonal line is the compression B, 2 900 lb, s T in plate = 5 800 lb.



am for Suspension Bunkers (R. W. Dull)

allow bins a combination of algebraic and graphic solution. In bin $ABCD$ (Fig 149), with smooth e normal pressure on side BC is $N = \frac{1}{2} wh^2 \frac{1 - \sin \phi}{1 + \sin \phi}$.

N, unit pressure at $B = 0$, and the unit pressure tal normal pressure N_2 and unit pressures on side or the bottom DC, continue AB and DC to inter- riangle EBC acts through its center of gravity, and ressure on $EC = cb$, represented by the area of the mal pressure Co at C = (area ECo) ÷ ½ EC. Unit ound by construction, and the total normal pressure is represented by area of trapezoid $DCon$. It acts gravity of the trapezoid. The component of the g parallel to the plane of DC is de. If, in Fig 149,

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the bin be extended, with a horiz bottom, to the right, to such dist
the plane of rupture laid off from C to right of BC (instead of to left,
cuts the surface, the direction of N would be reversed. N would be

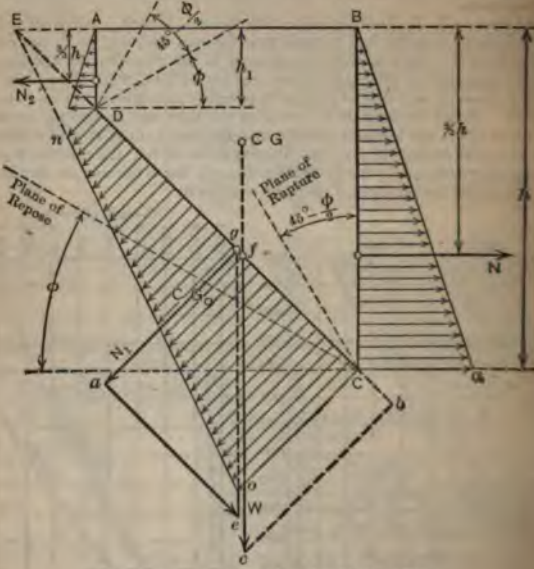

Fig 149. Graphic Solution for Shallow Bins

to intersection with W at f, the resultant of N and W laid off acting th
and a normal from end of resultant to EC extended represents norm
on EC, in a similar manner to ab. With
press known, the remainder of construction
as above.

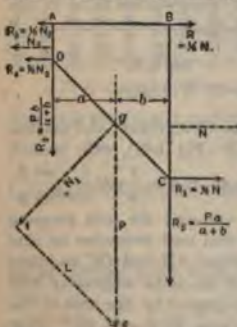

Fig 150. Stresses in Bin Framing

Stresses in the bin framing. Total
vertical stresses at the intersections are
Fig 150. These are carried by the frames
calculated in the ordinary manner.

Trautwine's formulas for the pressure on v
walls are:

(*a*) Surface horizontal, horizontal pressure
length of bin is $\frac{1}{2} wh^2 \tan^2\left(45 - \frac{\phi}{2}\right)$

For anthracite coal, total horizontal press
and pressure on lowest foot = 9.78 (z h

For bituminous coal, total horizontal press
and pressure on lowest foot = 6.32 (z h

(*b*) Surface sloping, $\delta = \phi$

Horizontal pressure $= wh^2 \sin^2\left(45 - \frac{\phi}{2}\right)$

al horizontal pressure = 14.22 h^2 and pressure on lowest foot

otal horizontal pressure = 10 h^2 and pressure on lowest foot

are usually calculated the same as for deep bins. For horiz
Eq 63 applies. Tensile stress in walls per ft of height at any
oriz press, lb per sq ft (Eq 63), and r = radius of bin. If
will be carried by bottom. If deep, Eq 62 applies. For
and equal at all points. If conical, tensile stress, at any
o an element of cone, is

$$H = \frac{\operatorname{cosec} \theta}{2\pi r_1} (\pi r_1^2 V + W_1) \qquad (73)$$

s with horiz; r_1 = radius of cone at point considered; W_1 =
this point; V = Eq 62, considering radius of bin r_1 instead

conical bottom is $H_1 = Lr_1$ (74)

3, using r_1 for r. In spherical bottoms, the tension parallel

$$H_2 = \frac{1}{2} Vr' + \frac{W_1 \operatorname{cosec} \theta}{2\pi r_1} \qquad (75)$$

in Eq 73, θ = angle of tangent to meridian with horiz, and
ension normal to a meridian at any point is

$$H_3 = \frac{1}{2} Vr' \qquad (76)$$

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SECTION 13

NAGE OF MINES

BY

RT VAN ARSDALE NORRIS
CONSULTING MINING ENGINEER

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D CONTROL OF MINE WATER

Sources of Mine Water

h outcrops is the chief source of mine water in coun-
ze rainfall, especially where mines are worked to the
drainage areas are above or tributary to extensive
ch cases, mines may be subject to sudden floods, de-
d usually requiring definite times to reach the work-
quantity and character of precipitation, and time and
inflow, should be kept for their value in giving warn-
ng under water-bearing surface wash, as in alluvial
to similar flooding, are subject to continuous perco-
, the amount dependent upon the character of the
mine workings to buried outcrop. In case of a coal
bed of fire-clay, the latter may be washed out over
isastrous inflow. Surface flood water may also find its
r open crop-falls,

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Surface water through cracked or broken measures is perh dangerous and troublesome than that from open outcrops. Where has been extensively worked by open or by shrinkage stopes unde rock cover, the network of cracks and crevices may admit large qu both surface and ground water, combining serious permanent flow wit cal flooding from rainfall.

Flooding may also come from caves extending into water-bearing wash or wa depositions, from mining under too thin a rock cover, or cutting into "pot buried valleys. In 1885, at Nanticoke, Penn, 26 men were buried in quicksa pot-hole over 100 feet deeper than the surrounding rock.

Water from permeable measures is always possible, and in formation massive sandstones may be expected. It may enter as seepage, or under pres from large areas, resulting in persistent inflows. Relief can be obtained only ing entire basins, or by general lowering of ground-water level. WATER FROM OR UNDERGROUND CHANNELS is sometimes encountered in shaft sinking or in mi coming in unexpectedly and in large volume. It may persist to the complet of territory tapped. WATER FROM UNSUSPECTED UNDERGROUND CAVITIES, eith caves or abandoned mine workings, is fortunately rare. Its serious aspect, quantity of water, is suddenness of inflow, which may endanger life and qu lower workings.

2. Prevention of Inflow

Surface. It is generally cheaper to keep water out of mines than to after it has entered. While it may be impossible to prevent the entr water the amount can usually be materially decreased by attention to inflow. In impervious soil, entrance of surface water can be minimize planned ditching around outcrop or fractured areas. In permeable so should be lined, or wooden or concrete flumes installed. If streams or channels cross outcrops or fractured areas they should be flumed, or di canals, designed to take maximum flood volumes, and located to minim of rupture from surface settlement often occurring during flood period

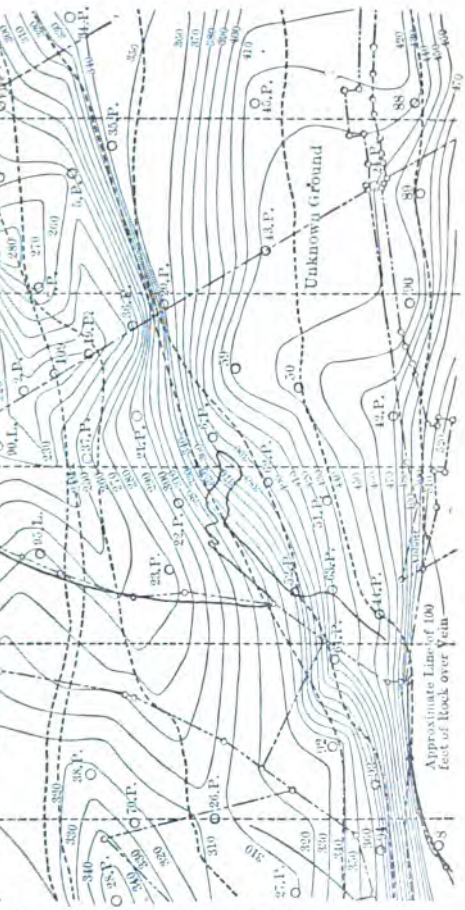
Drainage works are sometimes built through extensive districts by a comb interests, as in Hazleton region, Penn, where Black Creek has been diverted the coal measures for miles, relieving the drainage situation in a number o each of which formerly pumped practically the entire creek flow.

Besides diverting surface water, provision must be made for delivering th of mine pumps outside of drainage area of mine. This may cause contam streams. Some state courts have ruled that discharge of mine water into drainage channels is permissible, but that damages may be recovered for in into streams not reached by gravity in case the mine should fill. Thus draina discharging onto unpolluted watersheds may furnish ground for damages.

Entrance of flood water through mine openings other than crop-falls or cracke should be impossible. If there are openings below flood level, along creeks or flood plains, shut-off doors or temporary dams should be installed. In pla openings flood records should be obtained, and openings placed well above d Damage from flooding may be so serious as to warrant considerable expenditu it. In case of shafts, the cost of extending waterproof curbs above possible fl small, and the shaft spoil may be economically used for filling necessary to above high-water mark. This has been done at many shafts in the flood plain hanna River, Penn, recent shaft curbs being at least 5 ft above highest rec mark; besides which, provision is made for shutting openings promptly possible higher floods.

Underground. In mining under heavy water-bearing wash or q it is imperative to leave sufficient rock cover, and to keep workings l below buried outcrops to avoid danger. Preliminary exploration of face should be done by borings sufficiently close together for makin reliable contour map of the buried rock surface.

area of the buried valley of Susquehanna River, showing the urface revealed by borings. The wash varies from less than nd without borings the workings might easily have been ex- ep recesses in the rock. In such boring, especially in strati- oles must be continued into the rock, to determine its nature.



Map of Rock Surface of Buried Valley

be grouted to the rock surface, to prevent entrance of water t mine workings.

uired between workings and overlying wash de- acter of deposit, nature of overlying rock and method **al rule is possible.** In the Wyoming Anthracite Field,

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Penn, under the Susquehanna Valley and under the river itsel
left varies from 100 to 50 ft minimum; under deeper wash,
of thickness of wash. Under average conditions, with sands
overlying mine openings, and with ample pillars, a rock thick
ings equal to ½ of total cover is considered a reasonable minim
pitching measures, thickness of cover is determined by the dista
required to prevent inflow along bedding planes; best deter
test openings toward outcrop.

Douglas Bunting gives the following relations for 24 ft width of mi

$T = 1.5\sqrt{d} + 5$ (strong measures, rock surface fully explored).

$T = 1.5\sqrt{d} + 15$ (softer measures, liable to disintegration, rock surf

$T = 1.5\sqrt{d} + 40$ (strong measures, rock surface imperfectly explo

$T = 1.5\sqrt{d} + 50$ (softer measures, liable to disintegration, rock explored).

In these formulas: T = thickness of rock cover, d = depth of surf

Grouting and waterproof linings. Inflow from water- avoidable by waterproof linings of shafts and other opening under pressure into surrounding rocks.

In approaching known or possible OLD WORKINGS, and whe stone where cavities may be expected, drill holes, preferably b valves previously secured, should be kept well in advance o avoid disastrous inrushes of water, and to control flow from may be tapped by holes, letting the water out only in such such times as may be suitable for capacity of the pumps.

No means have yet been devised for materially diminishing the inf water, as encountered in porous formations of some western mines. ing plant is the sole safeguard. Grouting in advance of sinking ma bore holes surrounding proposed shaft location; or during sinking drill holes; or subsequently behind shaft lining of steel, brick, stone

In case of fissured or broken ground the flow may be reduced and cut off by cement grouting under pressure through bore holes, or by t process (23), consisting of injection through drill holes or into partl chopped straw, sawdust or other finely-divided material, followed by (first used at the Flat River, Mo, mines of Federal Lead Co, where it the flow from a network of fissures previously unconquerable).

Sometimes, when water pressure in cracks or fissures is so great th aration for grouting or mudding only results in spreading the water t areas of fractured or permeable rock, the pressure may be relieved holes up into the cracks, and plugging and grouting cracks, after w which have taken the flow temporarily are permanently closed.

Tapping underground reservoirs. When approaching of water under pressure, bore holes should be kept in advanc bored in such manner as to permit withdrawal of drill rods a when water is encountered. One hole (better 2 holes) should ahead of face, 10 to 20 ft in hard rock, 75 to 100 ft in soft r coal. Other holes, of about same depth, should be drilled (about 30° from center line) at intervals of 8 to 12 ft. If unexpectedly, insert a dry soft-wood plug of diameter smal freely; push to bottom of hole, and hold there by a rod o length, and a larger plug driven into neck of hole. In seve plug will have swollen tight, when outer plug can be remo outer 5 or 6 ft of hole. To a piece of heavy pipe screw a one end and a straight-way gate valve at other. Insert cou larged hole, and wedge with oakum and dry soft-wood wedge

ng, if pressure be great. Insert drill through gate
lug, withdraw drill, and regulate flow of water by
water body be known with certainty, the above
advance, and boring finished through a stuffing
f valve.

e waterproofing of a shaft be too costly, water rings
g, to lead water entering above them into sumps
vhence it may be pumped to surface, saving a mate-
lown the shaft, and power thus obtained utilized.
sumps are especially desirable for very deep shafts,
o or more lifts, the intermediate sumps then serving

sits opened by drift or tunnel, and later worked
pillar sufficient to retain permanently all surface
water level. Openings to dip are so arranged that
nd out through tunnel. A chain pillar ties up part
ver, is recoverable before final abandonment of the
umping throughout life of mine may pay for itself

Handling Water in Mines

eral, water should be intercepted at as high a point
to avoid unnecessary height of pumping lift. But
considerations as to the most suitable points for
g pumping plant, even at cost of increasing the
ater. The main collection point should be deter-
ment of a mine, and all openings driven from it
st 0.25% (0.5% being sometimes adopted, to make
al for both empty and loaded cars). Water ditches,
ide of track, in entries, drifts or tunnels, should be
in drainage levels are in broken or fissured ground,
with concrete; or wooden flumes, half-round terra
y be used, to prevent water from percolating into

lars should be maintained as long as possible under drain-
ere exhausted workings may be allowed to fill with water
as in small local basins in a flat deposit. Workings in
ded to the dip, if this would require installation of a num-
In such case, water may sometimes be drained off through
By similar means, water troubles may be avoided in sink-

ıse as adjuncts of main drainage systems, in con-
of a mine to another; for example, from a place
main sump, is
gher intervening
only when high-
ıan 34 ft above
Joints must be
presence of en-
limit of suction

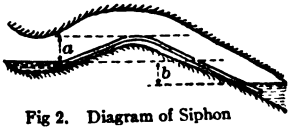

Fig 2. Diagram of Siphon

tisfactory when a (Fig 2) exceeds, say, 20 ft. The
ately less at altitudes above sea level. The follow-
e used for calculating siphons:

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$G = 1040\sqrt{\frac{d^5 h}{l}}$, in which: G = flow in gal per hr; d = diam pi

tive head, ft; l = total length, ft. In Fig 2, $b = h$.

The grade of pipe should be uniform, to avoid accumulations of ai To fill the siphon, for starting it, a short stand-pipe is placed at summ each end of main pipe.

Size of sump is chiefly dependent upon quantity of inflow. It enough to take care of inflow while pumps are stopped for ordin sump capacity of 8 hr maximum inflow, without submerging th ment, may be considered a minimum; capacities of 24 hr or more The possible results of deficient capacity are noted by W. F. Stau Tombstone mine, Ariz, was flooded, owing to a reduction of steam only 2.75 hr, involving large cost for recovery of the workings.

Proper location of sumps should be carefully considered in workings. Since haulage ways, with their drains, gravitate tow main sump is usually placed near foot of shaft or slope. W permit, a good sump is obtained at low cost by working out an ar to the dip, adjacent to shaft station, leaving ample chain p sump from lower workings.

Horizontal area of sump should be large. This gives large c material change in suction height of pump. Depth is of second Large sump area is also advantageous for condensing pumps, charge of heated water from condenser at a distance from su allowing it to cool before reaching pumps. In pitching, stratifi coal, where several veins are worked, the sump may sometim main workings and pumping plant in an overlying or underlyi being connected on main level only by a tunnel closed by a dar pipe from dam to pump is supplied with shut-off valves, thus, i mitting main workings to be flooded to considerable depths wit with operation of pumping plant.

Examples: At the Hazleton colliery, Lehigh Valley Coal Co, the Wharton vein, underlying main workings in the Mammoth. The Wh

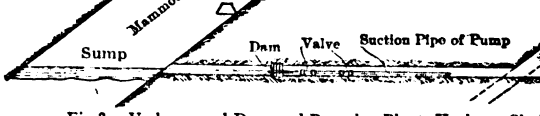


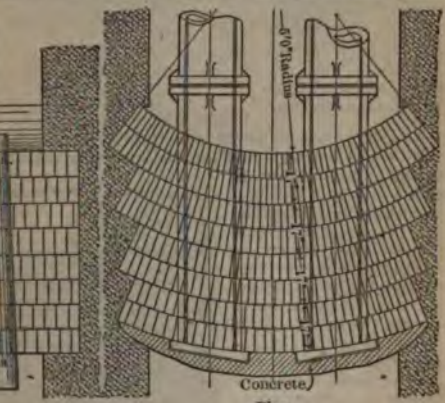
Fig 3. Underground Dam and Pumping Plant, Hazleton Shaf

rate slope, and for the lower 300 ft or more is not connected with mai by a single tunnel with its dam and pipe (Fig 3). In the Williamstow Branch Mining Co, Penn, a similar arrangement was installed duri pumps, located in a small seam, were cut off from main workings by d connecting tunnels, and the lower lifts, over 300 ft in vertical depth, w the water being readily pumped out after the strike. In both cases ar resulted from avoiding the flooding of large pumping plants during a when the power plant was operated only intermittently.

4. Underground Dams

These are often necessary to hold water under heavy heads. single- or better double-bulkheaded, with a clay core, may be Permanent dams should be backed by masonry, cal

water. They are sometimes of cut stone, but usually
lid abutments are essential, and all soft adjacent
, to insure against bodily movement of dam. Man-
st be provided, for draining off water, and to permit
For emergency use, outer portion of dam should
ich can be rapidly closed in time of need. During
laces, water-tight doors are erected behind the dam,
oderate heads, and arranged to be quickly closed,
rgency construction includes a hitch or masonry
which can be closed by steel I-beams, for checking
nent dam (Fig 4) can be built. In mining under



Plan

f Underground Dam, Hazleton Shaft Colliery

where a break in roof may cause inflows of sand and
es subject to periodical floods, such precautions are

s mine dams are subject to heavy strains and their strength
ulations are necessary. The following formula is for arch

$$t = \frac{pw\sqrt{4r - 1}}{4s}$$

r = shorter or external radius, in; w = width of opening
ter pressure, lb per sq in; s = safe compressive strength of
actor of safety, not less than 8.
W. S. Aldis (2), using same notation, but with factor of

$$\sqrt{1 - \frac{2p}{s}}\Big) \text{ for cylindrical (arch) dams.}$$

$$\sqrt{1 - \frac{1.5p}{s}}\Big) \text{ for spherical dams.}$$

e less than unity, or proposed material will be too weak.

lam, necessary where strata will not safely stand thrust of
plate supported on 4 sides, a condition requiring an intri-
plate tests have shown an increase in strength of fully one

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third over that of a beam supported at both ends. The New York Buildin 1903 permit for concrete floors a bending moment at the support of $wl \div 20$, f on 4 sides, in comparison with $wl \div 12$, for support at ends only (w = total formly distributed, and l = length span in ft). Hence, it is safe to calculate dam as a beam having its shortest length, loaded uniformly and supported at using the formula:

$$s = \frac{wl}{2\,bd^2} \quad \text{whence,} \quad d = \sqrt{\frac{wl}{2\,bs}}$$

Fig 5. Stepped Abutment, Concrete Dam

in which: s = unit stress in extreme fibers; b = lin d = depth, in; w = total load, lb; l = length, in. I the strength of plain concrete in tension should be tak over 50 lb per sq in, whence for a dam with minimu 7 ft, built to withstand 500 ft head,

$$w = \frac{62.428 \times 500 \times 7}{12} = 18\,208 \text{ lb per linea}$$

$$d = \sqrt{\frac{18\,208 \times 84}{2 \times 50}} = 125 \text{ in}$$

Result: a dam 10 ft 5 in thick.

A saving in material can be made by building a flat dam of reinforced concr

If the rock formation be of doubtful strength to stand direct thrust of either flat dam, a larger surface of abutment may be obtained by stepping the notches (Fig 5). In a large Schuylkill County, Penn, colliery, a special method was employed (Fig 6) to drown out a fire in upper levels without complete flooding. As the pillars were not considered safe to support the pressure which would be brought on the lower dams, these were relieved by flooding behind auxiliary dams, thus building up a series of counterbalanced pressures. Hence the coal in each pillar took no more pressure than that due to a sing any one dam.

Fig 6. Series of Dams, with Counterbalanced Pr

5. Drainage Tunnels

These are advantageous where location and topography permit. first cost is large, the elimination of pumping charges and freedom fro bility of flooding the mine, due either to accident to pumps or powe or to labor troubles, may warrant the investment. In most cases, h ditions are such that only the upper levels of a mine can thus be unwa gravity. When workings extend below tunnel level, pumps must be i But, drainage tunnels still have the advantage of saving cost of ra entire volume of water through a height equal to distance below w which tunnel intersects deposit. An estimate of length of tunnel fo comparison with a pumping plant must be made with careful consider cost of driving, maximum quantity of water possible for short peri height of lift for which the pumps and power plant must be designed.

Cost of drainage tunnels depends chiefly on character of rock, rate and cost of power and supplies. It ranges from about $5.50 to $16.0 yd; average say $10.00. Costs per linear ft of a number of western range from $13.50 to $27.25, averaging about $22.00. Cross-sectional

03

economy and convenience of driving than by water
ction, when timbered, is 8 by 8 ft, ranging from 7 by
being heavily timbered). Should the tunnel serve
ion, all of its cost is not chargeable to drainage.

ils are given of a number of drainage tunnels, itemized
ulated near end of section. Fig 12, 13, 14, 18, 19 and 21,
ons of mine tunnels, with position of drain and modes of
includes data for convenient reference here:

mples of Drainage Tunnels

State	Length, ft	Net section, ft		Approx av cost per ft
		Width	Height	
Ariz	11 892	8	8	$11.94
Mont	5 853	6	7	14.50
Penn	26 079	8	8	.....
Calif	5 055	10	9	.....
Colo	4 500	8	8	13.50
"	6 385	8	8	23 50
"	6 400	8	9	18 88
"	11 000	8	9	20.00
"	22 000	8	8	21.00
Penn	5 600	7	9	.....
Colo'	8 500	9	9	.....
Penn	21 650	8	12	.....
Colo	6 235	7 5	7	19.81
"	15 700	10	6.5	27.27
Utah	14 000	9.5	6.5	.....
Colo	2 600	7	7	23.38
Nev	22 000	14	10	.....

nnel for drainage only, the interest on first cost, plus
ortization, must be less than operating expense of
for amortization of pumps and power plant. The
nust be based on a conservative estimate of life of
a pumping plant, the final value of plant is its second-
less cost of removal.

OF WATER FROM MINES

6. Location of Pumps

generally be close to the sump, to avoid high suction
flat-dipping, thin deposits, where sump is excavated in
advisable to provide a channel from sump to a point
the pump, of such depth that sump may be entirely
The suction pipe dips into this channel. In case of a
s a rule, should be set from 20 to 30 ft above the bottom,
ntained within say 10 or 12 ft of the pump. The end of
well above bottom of sump, to allow room for settling
h must be cleaned out at intervals.

en pumping plant is extensive, the character of rock or
of pump room. With a firm roof, requiring little or no
be set side by side. With a weak roof, the span would
pumps must be placed in parallel chambers, or bette

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the form of a cross, with suction chamber in center. This gives
spans and permits easy supervision of plant.

Pump rooms should be placed as far as possible out of danger of disturb
mining operations. Special excavations in footwall rock are usually desirabl
plants. Width of room needs to be only sufficient to afford access to pumps
lubrication and replacement of parts. Length must be sufficient to permit
drawal of pistons or plungers with their rods, or removal and replacement
and barrels. Height must allow for convenient work with block and tackle o
working parts. Cramped pump rooms hinder repair work, and should be avoid
should be an ample supply of repair parts, and of tools, including jacks and ch
as repairs and adjustments are usually of an emergency nature. Steel lockers
are desirable, and covered cans for refuse and oily waste, all being kept scrupul

Pump rooms should be fireproof, with masonry, concrete, or steel
ports, to insure against falls of roof rock. Reinforced concrete arch

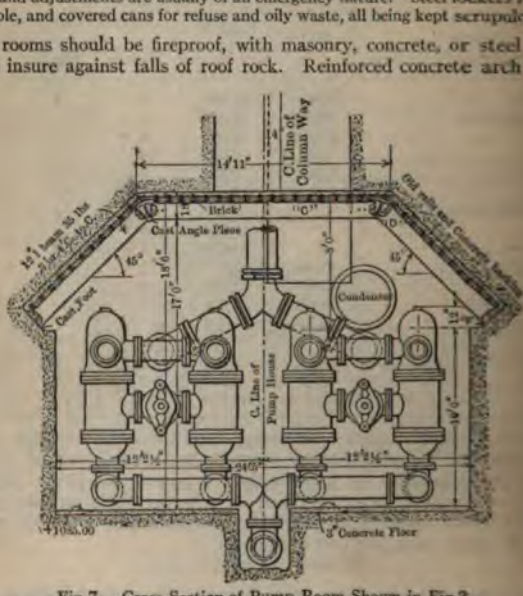

Fig 7. Cross Section of Pump Room Shown in Fig 3

sometimes used. Non-fireproof and badly kept pump rooms have
source of many disastrous mine fires. Electric lights should be employ
possible. Danger of fire in timbered pump rooms, common in the old
and in many western mining districts, may be greatly reduced by
paints or cement coatings applied to timbers.

Fig 7 shows plan and roof supports of a well-designed pump room at the
shaft of Lehigh Valley Coal Co, excavated in bottom rock of Wharton seam.
pound condensing high-duty pump. It has solid concrete floor and walls, ste
roof supports, lagged with steel rails, brick and concrete backing.

7. Classification of Mine Pumps

1. Cornish pumps, operated by { Beam engine / Bull engine / Horizontal or vertical, geared or direct-acting engine

eam pumps.. { Simple or single cylinder
Duplex { Single expansion
Compound

pumps..... { Steam-driven; flywheel, no gearing
Electric-driven; duplex, triplex, quintuplex, geared to motor

ups..... { Single-stage
Multi-stage

es and design of pumps are given in Sec 16 and 40.

8. Cornish Pumps

more large, single-acting plunger pumps, located in the long wooden or steel rods to steam engine on surface. engines were always used, but are now practically obsotype, a few of which are still in operation in the anthram cylinder is set at mouth of shaft, directly in line with inder is usually single-acting, the steam raising rod and weight on down stroke and are cushioned at end of stroke heavy rod, which is counterbalanced by one or more greased guides or is guided by rollers. Owing to the on, the speed is necessarily slow, partly compensated by Pumps with 24-in plungers and 10-ft stroke, with steam were not unusual; speed, 3 to 5 double strokes per min.

moderately economical of steam. A reported test (3) shows Assuming 10% friction, this shows a duty of 36,000,000 ft-lb team cylinder was 44 in diam by n-condensing; 21 5-in plunger, lift tor cards from this test. Working uld do considerably better. Cost large Cornish installation on the 3 per i h p per month; capacity 152 ft head. The question of econnts is largely one of type of driving gines, with their rather complicated omical, when working under proper as often been the case in the U S, y ordinary high-pressure engines eir advantage. Cornish pumps, though very costly to install. ely free from danger of flooding, as the pumping parts only are ell under water as out, provided valves remain unobstructed.

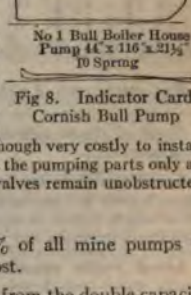

Fig 8. Indicator Cards, Cornish Bull Pump

9. Direct-acting Pumps

s probably comprise over 80% of all mine pumps in ges being simplicity and low cost.

Single and duplex. Aside from the double capacity of the duplex, there is little choice between these forms. Though desirable in point of compactness and moderate cost, their duty is low, because, having no flywheel, and their moving parts coming to rest at the end of each stroke, steam can not be used expansively. This

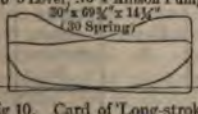

Fig 10. Card of Long-stroke Single-cylinder Pump

umption. Tests of 39 pumps (7) gave an average p per hr, ranging from 49.7 lb for a large, long-stroke

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single-cylinder pump, to 103.7 lb for a very short-stroke duplex pump, i condition.

Fig 9 is a typical indicator card from a short-stroke duplex pump, showing clea irregularity in stroke characteristic of the type. Total steam consumption und was 58.9 lb per i h p per hr. Fig 10 is from a 30 by 69.75-in single-cylinder pump, in plunger. The card shows starting effort on water column, and the curve of the from the small ports into the large cylinder. This pump, working against a 326-f showed total steam consumption of 49.7 lb per i h p per hr. Its economy was mat increased by exhausting into tail pipe, at a little below atmospheric pressure.

Compound direct-acting pumps. The objections to simple, direct- pumps can be largely overcome by compounding the steam cylinders, expa from first into second, and to a third and fourth if desired, in triple and ruple expansion pumps. While permitting expansive use of steam, the stant thrust essential to the design is retained, with greatly reduced consumption. In flywheel engines the only advantage of compounding reducing cylinder condensation, since, having a flywheel to equalize pow livery, any desirable ratio of expansion is attainable in a single cylinder. theoretical saving due to compounding may be shown as follows: Ass high-pressure cylinder 1 sq ft net area by 1 ft stroke, with an interme cylinder 4 sq ft area, and a low-pressure cylinder 16 sq ft area; then, initial steam pressure of 164 lb absolute, and neglecting cylinder condens clearance, and frictional losses, the results would be:

Cyl	Contents, cu ft	For'd press, lb abs	Back press, lb abs		Mean effec press, lb abs		Total press,	
			Cyl 2	Cyl 3	Cyl 2	Cyl 3	Cyl 2	Cy
1	1	164	38	38	126	126	18 144	18
2	4	38	0	8	38	30	21 888	17
3	16	8	.....	0	.....	8	.....	18
							40 032	53
Equivalent area of a single cylinder, of same stroke.....							1.7 sq ft	2.28

Ratio of steam consumption for uniform power: single, 1; compound, 0.53; expansion, 0.44.

In practice there are necessary losses: in cylinder condensation, in friction of through ports and passages, and in friction of additional pistons and other moving all of which reduce theoretical saving. Furthermore, for efficient multiple expa high steam pressures and vacuums are essential.

Actual steam consumption of compound and of multi-expansion direct-a pumps varies considerably. A plant of triple-expansion pumps at Bisbee, Ari showed 80 000 000 ft-lb duty per 1 000 lb steam. Recent tests of compound pu made with steam meter, gave following results under best load conditions:

Steam cylinders, in	Plunger, in	Stroke, in	Head, ft	Initial steam pressure, lb	Remarks	Duty per 1 000 lb steam ft-lb
23 and 40	14	48	590	80	Condensing	69 000
26 " 45	14	48	590	80	"	65 000
30 " 50	14	48	757	70	"	66 000
17 " 26	10	36	269	75	Non-conds'g	39 600

nps, respectively 24 and 38 by 10 by 36 in and 25½ and der 675 and 668-ft lifts, and condensing into tail pipe with ctively 46 000 000 and 47 000 000 ft-lb duty (Fig 11).

of direct-acting pumps is shown tests of a 30 and 50 by 14 by pump, tested by steam meter. ion is due to a combination of low speeds, changes in friction l variations of steam pressure, tling to attain slow speeds.

No 4 Level, No 2 Jeanesville Pump 25½ & 38 x 36 x 10 9½ Revs.–30 Spring Siphon Condenser.

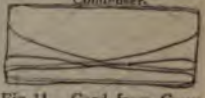
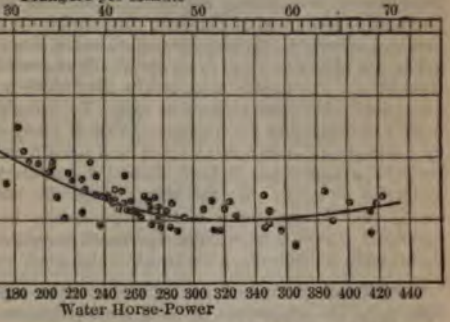


Fig 11. Card from Compound Pump

duty between similar pumps is steam pressure, and shows the e of designing pumps for much actually available. Many pumps are found by indicator initial pressure is half or even one third of the available. pressure found in 39 pumps (7) was 27.8 lb, boiler pressure minimum of 75 lb, during a 7-day test. This practice is



eed and Water h p, Based on 31 gal per Plunger. Example: Read to Bottom = 241 h p; then to Curve = 32.95 lb hp-hr; 241 × 32.95 = 8204.5 lb Steam per hr

le stoppage of pumps, in case of low steam pressure, but re cheaply provided for by ample reserve sump area, paid cessive expense for power. In wet mines, from 50 to 80% d for pumping, a saving in which is well worth extra expense plant, in order to permit installation of pumps properly

10. Flywheel Pumps

grade are common in waterworks and manufacturing are in mining. They range from quite small pumps st grade. Duties are from 100 000 000 to as high he best triple- and quadruple-expansion condensing mples may be cited two compound condensing fly- e plant (Art 9), 30 and 38-in steam, 6½-in plungers, lnst 1 000 ft head, which are said to have shown over The high-duty Knowles pumps, in Hazleton shaft of are reported to have given on test a duty of over pumps, of which there are a number in the West and

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Northwest, working under heavy heads, are among the best known of this of pump. At the Chapin mine, Iron Mountain, Mich, is a triplex differe Riedler, steam cylinder 22 and 36 by 30 in, differential plungers of 6¾ and 9 raising 2 200 gal per min, under 1 700 ft head.

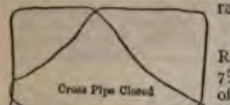

Fig 13. Card from Riedler Compound Pump

A comparison of indicator cards in Fig 13, from a Riedler cross-compound pumping engine, 13 and 20 7⅝ by 16-in cylinders, running non-condensing, with of Fig 10, from a larger direct-acting compound pump, at once the advantages attained by addition of a fly The disadvantages of the flywheel pump are: the gr weight and size of parts, higher cost, complication of anism, with resulting greater liability to accident or der ment, and the higher class of labor required to run succes high-duty pumping engines.

11. Electrically-driven Pumps

Plunger pumps have high efficiency, up to 85 to 90%, but, owing to disadvantage of requiring gearing to reduce the high speed of motor to necessarily slow speed of plungers, they are heavy, and expensive both in cost and in repairs. Where electric current is dear, their efficiency easily weighs the objections, and in all electrically operated plants their use shoul considered. The smaller sizes usually have single-acting plungers of the t type, dispensing with connecting rods. To maintain an even flow of wa from 3 to 6 plungers are common. A small pump of this type, 3¼-in plu by 5½-in stroke, driven at 33 r p m by a 100-h p motor, and delivering 25 per min against 1 000 ft head, showed a combined efficiency, for pump motor, of 78 to 84.4%; for pump alone the efficiency was from 86.5 90.9% (8). Large pumps of this type usually have double-acting, outs connected plungers, with large slow-speed motors and heavy gearing. T show excellent efficiency, quite equal to high-grade steam pumping engines.

Centrifugal pumps of the multi-stage type are coming largely into They have the advantages of compactness, lightness, relatively low first

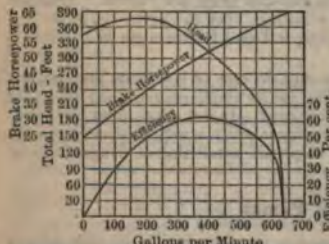

Fig 14. Curves from 5-inch Three-stage Centrifugal Pump at 1 788 r p m

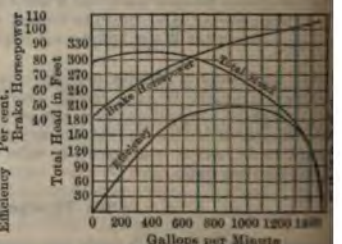

Fig 15. Curves from 8-inch Two-stag Centrifugal Pump, at 1 400 r p m

maintain a constant flow without shocks, and, owing to their high rotative sp are well adapted for direct connection to electric motors or steam turbi Their disadvantages are: relatively low efficiency and rather rapid wear, general unsuitability for gritty or very acid water. The efficiency of pu alone rarely exceeds 70% and falls rapidly as the pump departs in quantity head from its designed duty. Fig 14 and 15 (12) show the variation in effici

gals under different conditions. Fig 16 (13) exhibits lb steam, of a two-stage centrifugal, direct-connected rking against high ditions of variable to figure the com- mp at much over ler favorable con- r 70% combined d.

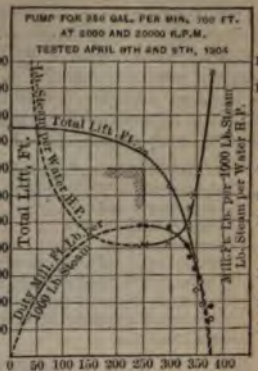
PUMP FOR 350 GAL. PER MIN. 700 FT.
AT 2000 AND 20000 R.P.M.
TESTED APRIL 9TH AND 9TH, 1904
Total Lift, Ft.
Duty Mill. Ft. Lbs. per 1000 Lbs. Steam
Mill. Ft. Lbs. per 1000 Lbs. Steam per Water H.P.
Gal per Min

Fig 16. Duty of Two-stage Centrifugal Pump

age, electric-driven ni, Mex, raising 600 020 ft total head, f pump and motor It is reported that o mine, Spain (11), to 450 ft, gave effi- %.

lts of tests of cen- in iron mine, Mich, d 16th levels, 3 000 l, by 6- and 3-stage respectively of 1 000 rks gave: av h p of e to water efficien- -stage, 73.9%; for obtained heads of 1 066 and 451 ft

lata on centrifugal pumps, see Sec 40; for electric-driven

ompressed-air Driven Pumps

mpressed air is less efficient than by steam or electric- warranted: (*a*) in gassy collieries, where electricity is too far from boiler plant for economical steam trans- ere heat from steam pipes is objectionable, or heat would injure roof rock; (*c*) where there is no electric ent required would not warrant an installation. It is d air in pumps built for steam, and in cylinders not the pressure employed. Under these conditions it is of indicated power at compressor is represented by th reheated air, the efficiency may rise to about 30%. that ordinary direct-acting pumps fail to make full e volume is too great for compressed air operation. le from larger flywheel pumps, designed for the service. trouble from freezing in exhaust ports may be mini- rs and traps in air lines close to the pumps, and by of water, taken under column-pipe pressure, into the e valves.

igh efficiency, many extensive air pumping plants are in use. or explosion, and absence of exhaust steam and of heat from counterbalance the cost of extra power required. With and a steam consumption of about 25 lb per h-p hr, a total approximates a duty of 16 000 000 ft-lb per 1 000 lb steam idering the usual losses involved in carrying steam under- umping duty of about 20 000 000 ft-lb.

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13. Air-lift Pumps (see also Sec 15)

This system of pumping, while convenient, has low efficiency, exceeding 30% total and under unfavorable conditions falling to A submergence of air-supply pipe of about 55% of pumping l give best general results. The Pohlé air lift, usually considered a pumping only, was installed in the Candelaria mines, Chihua for lowering water in an extensive area of permeable rock, but placed by compound steam pumps. A test of a Pohlé pump gave following results (18):

Net lift above average water level	
Submergence of delivery pipe (air admitted to delivery pipe 5 ft above lower end)	
Diameter delivery pipe	
Diameter air pipe	
Volume water delivered per min	
Volume free air used per min, at 107 lb gage	
Consumption free air per cu ft water	
Horse power consumed by compressor	
Total efficiency	

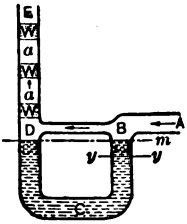


Fig 17. Diagrams of Starrett Air-lift Pump

Air-lift pumps have been applied for raisin in concentration of ores (20, 21), and for ha cyanide mills.

An ingenious air lift, not requiring subm in Fig 17 (22). Compressed air enters at *A*. U-pipe is raised by difference in pressure at due to throttling and friction, as air passes t diameter pipe *BD*. Thus the column *BC* is d and *CD* is correspondingly raised. The enteri cessive small volumes of water near *D* and e the ordinary air-lift pump. This pump (Starre through a considerable range of lifts by H. C to give from 25 to 34% total efficiency (inclu pressor), and from 36 to 50% for pump alon

14. Design of Mine Pumps (See also Sec

In general, piston pumps are the cheaper, but are unsuit about 300-ft lift is their limit in good practice. Plunger pump or outside-packed and connected. INSIDE-PACKED pumps, wit actuated by piston rod working between two water chambers pact; but it is somewhat difficult to keep packing tight, and t packing box for piston rod. This design is common for small for moderate-size pumps. OUTSIDE-CONNECTED AND PACKED plungers, entering opposite ends of water barrel, which is provi partition, plungers being connected by side rods. There are c boxes, readily accessible, plungers are easily removed, pump is in large pumps, weight of outer ends of plungers is carried type is generally used for large pumps, rarely for small.

Lining. For good water, pump barrels and plungers of ha satisfactory; for acid water, linings should be used, with chil resisting bronze plungers. Lead makes satisfactory linings, b apply and often wears rapidly and loosens. Wood lining, ca swelled with pure water, is best, but, because of its thickness castings and is consequently expensive. Various patented en in some cases, but have been little used. Cement lining is state with 75 gr free sulphuric acid per gal (24).

s are of 3 general types. Large single valves, of bronze or faced with leather r, are suitable for low lifts; with high lifts they are apt to wear rapidly from Multiple small valves, usually of bronze, are suitable for high lifts, but more trouble in maintenance. Mechanically operated valves are rarely ept in high-duty pumps, like the Riedler. They permit high plunger speed ice slip. Valves should have low lift, to prevent noise and breakage. Light n suction and relatively heavy springs on discharge valves reduce impact, nore rapid closing permit higher plunger speeds. Valve chambers should eparately from the water barrels, to permit easy replacement, and placed be readily accessible for repairs. Covers should have slotted bolt holes, ged bolts which are swung down instead of being withdrawn.

e. For common direct-acting mine pumps, 12-in stroke and upwards, speed of 100 ft per min is good practice; for long strokes, of 48 in plunger speed is about 125 ft; for mechanically-operated valves, 200 or even more, has been successfully attained. Ordinary pumps in y will run at from 200 to 300 ft plunger speed, but with large loss of , greatly increased slip due to valves not having time to close properly, notable shortening of stroke.

tions. For permanent pumps, solid foundations of stone, brick or concrete, orth while. They should be well built and true, of good materials, and only pump, usually the water end, should be bolted down, the other being left free for expansion. Timber foundations are suitable for small or temporary pumps.

ambers of ample size should be provided. For heavy lifts, air chambers should d on column pipe, as well as on the pump. They are best arranged by put- at bottom of column, connected to an extra length or two of pipe with closed ction air chambers are even more important, to take the blow from the water mp reverses its stroke. They should be beyond the valve chamber, the water nducted directly to air chamber elbow, past chamber inlets. Air chambers must be kept charged with air. This is done by some form mp, either a small pump actuated by the main chanism, or an automatic pump (10). Fig 18 automatic air charger. On the suction stroke, s vertical pipe through inlet check valve *A*, ced by the returning water on delivery stroke check valve *B* to the outlet. To operate, first ve connecting with pump, which must be in , then gradually close until check valves begin For large plants, a small compressor may led to charge the air chambers.

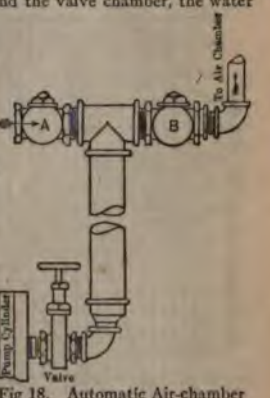


Fig 18. Automatic Air-chamber Charger

on pipe area should be not less than plunger area, the pipe as short and as possible, with no depressions which t as air traps, and with absolutely tight A strainer is always necessary; in its form, a wooden box bored with 1-in les, aggregating not less than 5 times the rea, or a frame of wood or iron covered rse-mesh wire cloth is satisfactory. There are special forms of strainer, in the pipe itself, but, as they are more apt to clog, they must be emovable for cleaning. Suctions of unusual length or high lift should led with a foot check valve, having an area of passage at least equal to he pipe; they are necessary to permit of filling suction pipe in starting . Elbows should never be used; bends where necessary should be of ius, preferably not less than 4 times pipe diameter. All bends should heavy, as they wear rapidly *with gritty or acid* water.

15. Pump Miscellany

Condensers are of 3 types: surface, barometric, and pu
little used in mining, because of its cost and of the large vo
quired. The barometric type, with its necessary height of abo
applicable where considerable quantities of water are run d
elevations to the sump (siphon condenser is a modification of
pump condenser is most used, a combination of barometric h
replacing the tail pipe. The pump is made relatively economi
into the condenser, or by using exhaust steam for jacket h
vacuum may be obtained by exhausting a pump directly into
the exhaust pipe must lie longitudinally in tail pipe, the diame
large enough to avoid undue friction from the obstruction so

In case of acid water, a serious objection to condensing pumps is i
water and its consequent greater chemical activity. Condensing wa
from a part of the sump distant from that heated by the discharge.
ment is largely dependent upon superficial area and shape of sum
water. With neutral or slightly acid water, the condenser may safel
suction, and the heated water be disposed of at once. This course is
deep sumps of small area, which would otherwise become overheate
water, and sumps of sufficient area for cooling, the condenser disch
end of sump and the supply close to pump suction. The economical
for a given case must be determined by experiment. Under favorab
26 in is attained, while with small sumps and very acid water as low
20 in may be most satisfactory.

Column pipe. The economical size of column pipe sho
from cost of pipe, cost of power, quantity of water, and frictio

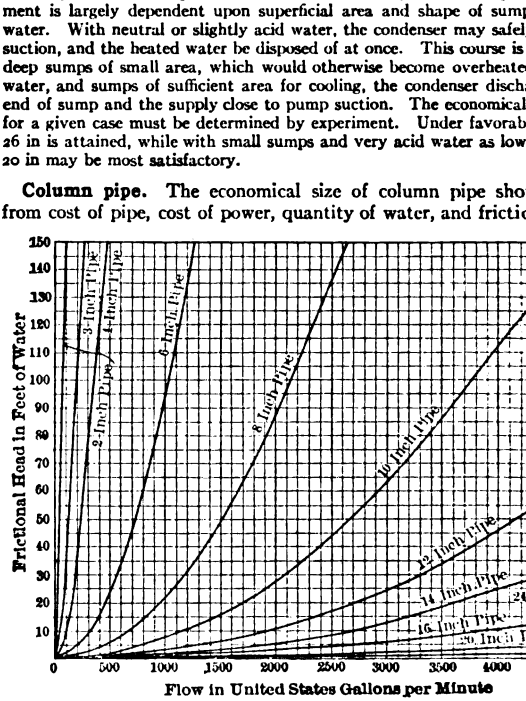


Fig 19. Friction Heads, New Cast-iron Pipe, 1 000 Ft Long (by D

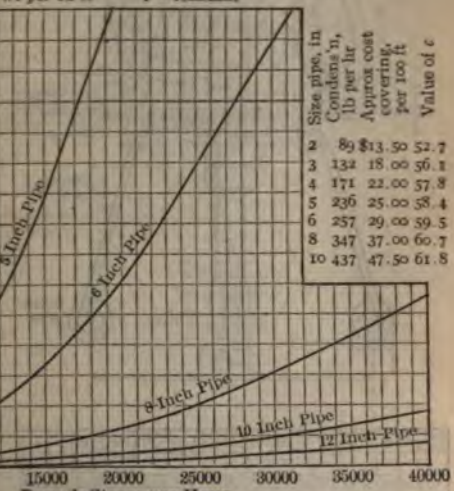
As the pipe may gradually become fouled or scaled, it is adv
diameter larger than that calculated. Column pipes are u
wrought iron; the latter is satisfactory for clean non-corrosi
for acid or gritty water.

ordinary bell-mouthed cast-iron pipe with calked lead joints . Cast-iron flanged pipe is in common use, the thickness generally in 10 or 12-ft lengths, with bolted flanged connections. Though male and female joints are good in case are troublesome when lengths have to be removed for ns to flanges, either with concentric V-grooves or slightly gaskets, as being more readily removable and quite as reli- ¼ to ½ in by 1 in, wrapped with tarred hemp makes a good e made by bevel rings treated in same way, without necessved pipe sections. EXPANSION JOINTS are unnecessary for but are essential for long lines. They are usually of the with gland down, the socket being secured to shaft bearers ere space permits, offsets with long-radius bends are preferapt to stick or leak. Elbows or tees should never be used ys use bends, of not less than 4 diameters radius, and ls.

ying column pipe for making pump repairs, a valve should d column, and a separate blow-off connection from the . To minimize friction, gate valves only should be used over 10-in diameter should have by-passes.

gram, Fig 20, is convenient for proportioning steam n probable use of steam per hr, permissible loss in

cu ft per min L = length pipe, ft d = diam pipe, in
wt per cu ft c = constant



Size pipe, in	Condens'n, lb per hr	Approx cost covering, per 100 ft	Value of c
2	89	$13.50	52.7
3	132	18.00	56.1
4	171	22.00	57.8
5	236	25.00	58.4
6	257	29.00	59.5
8	347	37.00	60.7
10	437	47.50	61.8

densation Losses in Covered Steam Pipes, per 1 000 Ft, for 100 Lb Pressure

er compared with cost of pipe, and find size of pipe ould be thoroughly covered with insulating material, . Approximate condensation losses under average overed pipe, and the saving due to covering, are ll steam lines should be fitted with efficient water lose to the pumps, and with gate valves both inside

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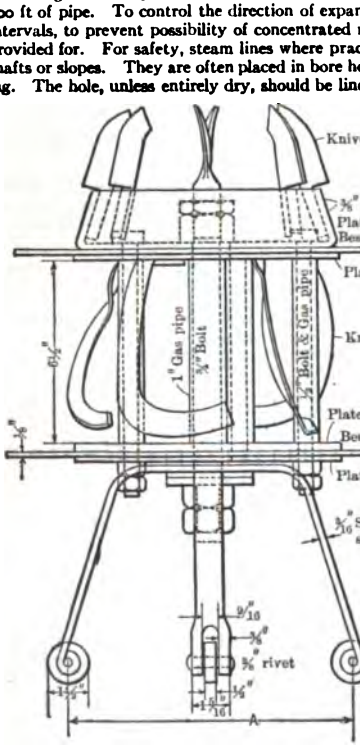
and outside to shut off steam in case of accident or rep
termediate valves decrease danger of accident.

Steam lines in tunnels or shafts are usually hung from timb
tected from heating which might cause charring or fire. Expa
they may be of the gland type, but a long-radius loop or bene
A safe figure for expansion of steam lines under ordinary m
100 ft of pipe. To control the direction of expansion, steam p
intervals, to prevent possibility of concentrated movement at
provided for. For safety, steam lines where practicable should
shafts or slopes. They are often placed in bore holes, thus avoi
ing. The hole, unless entirely dry, should be lined before insta



PIPE DIA.	A	B	C	D	E	F	G	H	J	K	L	KNIVES M	P	Q	R	S	T
10"	8"	7⅝"	6"	1 3/16"	5"	4½"	3½"	¾"	1"	11"	3"	As drawn	3½"	3½"	3½"	1¼"	11⅜"
12"	10"	9⅝"	8"	1 3/16"	10½"	6½"	4½"	⅞"	1"	1 3/16"	3"	As drawn	3⅝"	3⅝"	3⅝"	1¾"	13⅛"
14"	12"	11⅝"	10"	1 3/16"	8½"	8½"	6½"	⅞"	1 3/16"	1 3/16"	4"	As drawn	5"	5⅛"	5"	2"	16⅛"

Fig 21. Pipe Cleaner for Column Pipes

pipe and cuts loose the scale. This requires openings only at
economizes time and expense (25).

Neutralizing acid water. Acid mine water is some
pumping, thus avoiding damage to pumps and column pip
on actual conditions in each case is necessary to determine

character and quantity of water, type and cost of pump- or other alkali required. Milk of lime, added to sump at regular intervals, has been employed in Butte, Mont.

oles is a common practice in the anthracite region, the holes drilled from surface to mine workings. In weak h wrought pipe; in hard rock they are unlined, the rock practice permits any desired location of pumping plant, ape of mine openings, frequently reduces the lift, avoids d installing costly column pipe in shafts, and avoids possi- rkings from broken column pipe. Holes are from 8 to churn or shot drill. The objections to the practice are: les by caving or settlement of strata, and difficulty of re- n out. Bore holes in water-bearing, porous strata, with een also successfully used to lower water level in advance dalaria mine, Mex (26).

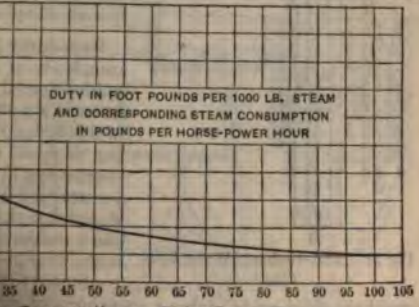
Duties and Cost of Pumping

Summary of Pump Duties

mp	Duty, millions ft-lb per 1 000 lb dry steam	
	Non-condensing	Condensing
.....	30–45	40–60
.....	20–35	30–50
.....	35–50	60–70
.....	.....	80–100
.....	.....	100–120
.....	65–100	100–180
r.....	16–20	.....
.....	20–45	.....
	10–30 (pump and compressor)	.....

cies of Electric-driven Pumps

	Pump only, per cent	Pump and motor, per cent
.....	85–92	78–84
.....	60–70	50–65


DUTY IN FOOT POUNDS PER 1000 LB. STEAM
AND CORRESPONDING STEAM CONSUMPTION
IN POUNDS PER HORSE-POWER HOUR
35 40 45 50 55 60 65 70 75 80 85 90 95 100 105

m Consumption per Horse Power Hour

Fig 22. Pump Duties

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The cost of pumping, being manifestly dependent upon ch cost of power, labor rate, height of lift, character of water, and q can not be given in figures of general application. At the colliery, Penn (28), the quantity of water pumped and the co were as follows:

	1901
Water pumped	567 113 616 gal
Average lift	1 141 ft
Total cost:	
Steam	$21 850.46
Labor and supplies	8 915.06
	$30 765.52
Cost per 1 000 gal, 1 000 ft vertical	0.0495
Cost steam per b h p per 24 hr	17.77

The plant was old, consisting chiefly of simple non-condensing pumps to 9 000 gal per min, depending upon the precipitation. It was afterw modern compound condensing pumping plant, at a cost of $120 000, pumping cost to under 3¢ per 1 000 gal, 1 000 ft vertical. At the L Co's Hazleton shaft, the operating cost of the high-duty compound is stated to be 1¼¢ per 1 000 gal, 560 ft vertical lift, or 2¼¢ per 1 000 g cost with the old Comstock Cornish pumping plant (5) is stated as $ for an average of 5 040 gal per min, pumped 1 074 ft vertical, or 19 1 000 ft.

These are probably about the limiting figures, and indicate roughly cost under different conditions.

17. Hoisting Water in Tanks

Probably the first water-hoisting plant was installed in 18 Fidler colliery, of the Pennsylvania Railroad Co, at Shamokin, at least 18 water-hoisting p installed in the Pennsylvani gion (30). This service re use of special shaft compartr with suitable head frames a gines. The tanks are always to 6 ft diam and 1 200 to 4 o There are 2 general types: en bottom-dumping.

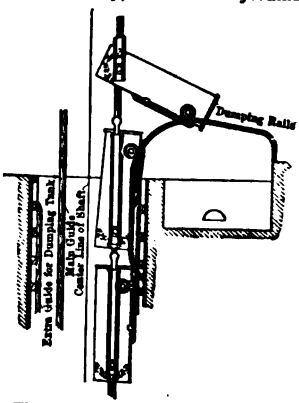
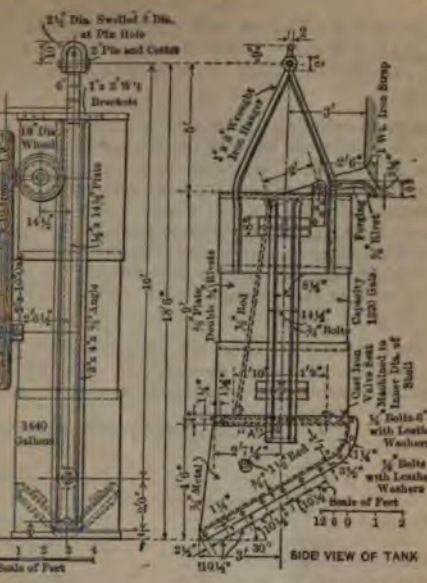


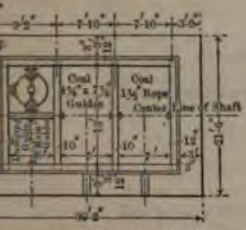
Fig 23. End-discharge Water-hoist Tanks

End-dump tanks (Fig 23 completely in about 2 secon quiring special movable chu water down the shaft. Th of this design are: necessity ing to avoid overwinding a overturning in the shaft, and construction and maintenan dumping tanks it is advisable the 2 regular side guides and angles (Fig 26). The side sl is adjusted to throw center of toward the third guide, giving a smoothness of motion well wort and trunnions are shown in Fig 24 and 25.

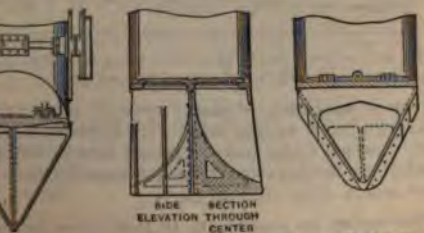


ist Tank,
olliery

Fig 25. Bottom-discharge Tank, William Penn Colliery



rangement of Guides for Water Hoisting



ge Casting,
Coal Co

Fig 28. Double Bottom-discharge, 5-ft Tank, Susquehanna Coal Co

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should be into a basin at shaft collar, having minimum depth o to avoid destructive action of falling water. The bottom-fil large, leather-faced clack valves, opening automatically as tank sump.

Bottom-dumping tanks (Fig 25) are similar in design to the chief difference being in the use of the discharge chute be (Fig 27 and 28). The trouble from impact, in striking the wa with flat bottom of tank and horizontal valves, is avoided in p struction shown in Fig 27, though a side thrust on enterin caused by the shape of discharge casting. This difficulty is re double-discharge bottom casting (Fig 28). On reaching the sur in both designs are opened automatically by a lever operat fixed guide at the landing (Fig 25). These tanks require abo discharge.

Cost of installation at two anthracite shafts (William Pen given below, the proper proportion of shaft sinking, headfram and boiler plant costs, being charged against the water hoist (3c

Costs of Installing Water-hoisting Plants

	William Penn	
Depth of shaft	953 ft	
Capacity of tanks (balanced)	1 440 gal	
Size of engines	32 by 42 in	
Size of drums	Straight 12 ft dia	Con
Rated capacity of hoist, 24 hr	2 100 000 gal (280 000 cu ft)	
Best record, 24 hr	2 291 040 gal (307 000 cu ft)	
Cost:		
Sinking and timbering	$20 673.81	
Headframe	4 224.13	
Water-hoist engines, foundations, and house	15 583.64	
Tanks and ropes	2 393.23	
Steam line	.....	
Boiler plant	3 726.12	
Cost charged to water hoisting, including proportion of shaft sinking and steam plant	$46 600.93	
Cost per 1000 gal daily capacity	$22 201.00	
Cost per 1 000 cu ft daily capacity	$10.57	
	$88.08	

The DUTY of the 1 500-ft Lytle shaft water hoist was 33 260 000 ft-l lb dry steam. Description of engines is given above.

Cost of operation of plants specified above varied from 1 000 gal, 1 000 ft vertical hoist. The higher figure is the avera shaft, 1 400-gal tanks, 918 501 200 gal hoisted during 3 years; unwatering a flooded 1 500-ft shaft, average lift 740.6 ft, 236 906 in one month, with 2 600-gal tanks.

Following data are given by Mr. D. Muir (33) on bailing 150 000 oo ing the 30-ft diam Rayas shaft, Guanajuato, Mex. Two hoists wo electric-driven, raising 575 gal per min, the other steam-driven, rais

des, each weighted by rocks weighing about 1 ton. The July, 1912, were (@ 50¢ U S per peso Mex):

	Costs:	
3 560 gal	Labor	$ 902.65
3 088 "	Material and supplies	501.54
2 648 "	Electricity and fuel	2 626.46
568 ft	Superintendent and electrician	112.50
	Work in stopes	101.64
		$4 244.79

o gal hoisted 1 000 ft.

useful for: unwatering flooded mines; regular drainage of as to require more than one pumping lift; and for mines ater, or where water is notably acid It is not warranted workings. In cost it compares favorably with pumping by conomical than multiple-expansion or high-duty flywheel electricity.

d water hoist was installed in 1905 by the Del, Lack, & ton shaft, Scranton, Pa (31, 32). The plant has been re-

rically
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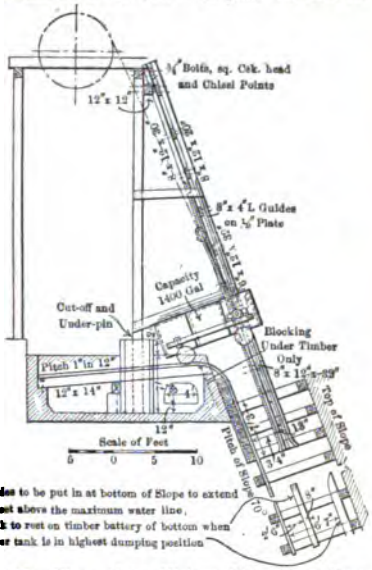
¾" Bolts, sq. Ctk. head and Chisel Points
12"x 12"
8"x 4"L Guides on ½" Plate
Capacity 1400 Gal
Cut-off and Under-pin
Blocking Under Timber Only
Pitch 1"in 12"
12"x 14"
Scale of Feet
0 5 10
Guides to be put in at bottom of Slope to extend 20 feet above the maximum water line.
Tank to rest on timber battery of bottom when upper tank is in highest dumping position

Fig 29. Water Hoist, Hickory Ridge Slope No 4, Union Coal Co

m, and danger of fire incident to their use; (*e*) almost of falls or squeezes in the mines; (*f*) above all, the

DISADVANTAGES are: (*a*) higher cost of installation; afts or shaft compartments, which can not ordinarily design of a plant; (*c*) greater steam consumption, as pumps.

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SECTION 14

E VENTILATION

BY

. ERNEST BRACKETT

MINING ENGINEER

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laborately developed in collieries, because of frequent occurust, but its importance for metal mines also is now clearly ot alone the health and safety of miners but also their

of the natural and artificial circulation of air currents in alysis of mine air, and the physiological effects of its imonly in so far as they affect ventilation. These subjects in Sec 23.

1. Mine Atmosphere

ides the normal constituents (Sec 23, Art 1), added of impurities: (*a*) Smoke and gas from blasting, the products being CO and CO_2. (*b*) Dust from mining und haulage roads. (*c*) Men, animals, and lights CO_2; ordinary lamp or candle flames are extinguished nt is reduced to about 17% by volume. (*d*) Emana- CO_2, CH_4, and H_2S; these may occur in metal mines, uantities; in collieries the first two are important, and oal and mixed in certain proportions with air be- in passing through wet mine workings absorbs watery ch saturation. (*f*) Minor sources of pollution are:

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gases from rotting timber, excretions of men and animals, mine fires which may produce CO (3, 6). For further detail acteristics of mine gases, see Sec 23, Art 2.

Mine gases or "damps." Firedamp is chiefly CH_4, bu portions of hydrogen, ethane, and higher hydro-carbons. Blac ual gas, left after partial or complete removal of O from the a usually consists of 85 to 99% N and 1 to 15% CO_2. Whitedar damp is a mixture of air, blackdamp, whitedamp, and CO_2; plosion of mixtures of firedamp, or coal dust, and air. In fres deeply colored with soot or smoke. Stonedamp is H_2S. (Fo see Sec 23, Art 2, 3.)

Table 1. Analysis of Mine Air, Rendered Dangerous by

Elemental gases	%	Air and mine "dam
Carbon dioxide (CO_2)	1.49	Atmospheric air
Carbon monoxide (CO)	0.60	Blackdamp
Oxygen (O_2)	18.25	Firedamp
Methane (CH_4)	1.25	Whitedamp
Hydrogen (H_2)	0.29	Hydrogen
Nitrogen (N_2)	78.12	
	100.00	

Interpretation of air analyses. (Method of analysis, Sec 2 names are given to various mixtures, as well as to elemental gases, t of expressing an analysis: (*a*) composition of the atmosphere is st of elemental gases and of those which result from chemical combinat is considered as composed of air, and the various damps and residu fied either as air or damp. In computing percentages under (*b*) fro a conventional method is adopted, viz: Percentage of normal air = 100) ÷ 20.93. CH_4 is reported as firedamp, and CO as whitedamp. less 0.03% is termed excess CO_2. 100% less the sum of percentage and all other constituents except total CO_2 and N, gives the excess excess N and excess CO_2 is termed blackdamp.

Safety-lamp test for methane (CH_4). CH_4 forms a trans called the "cap," over and surrounding the brighter part of must be free circulation of air about the flame. The cap is with the less luminous flames, and larger with the hotter flam against a black, non-reflecting background. Best test lamp alcohol, instead of vegetable or mineral oil. Davy lamp was f and is still sometimes used, though safer and more sensitive la had. With Davy lamp (flame lowered to 0.1 in high), 1.5% CH produce cap; at 2%, cap is about 0.1 in high above flame; at 4 3.7 in. Special test-lamps detect 0.25% CH_4. Lamp tests are scarcity of O or presence of CO_2. (For safety lamps and tests,

Natural temperature of a mine increases with depth belo but the relation between temp and depth is quite irregular. of strata at 20 to 30 ft below surface is constant, and about equ temp of the region. At greater depth, the temp increases 1° F of 40 to 100 ft. The rate varies even in the same region. Av in widely different places shows a rise in temp of 1° per 64 temp which may be expected is therefore shown by: $t = m$ where t = temp in deg F; m = mean annual temp, which in t is taken at 50° F; D = depth in ft below surface. (Examples are given in Sec 23, Art 4.)

Impurities in Mine Air

s (see Sec 23, Art 3). The danger arising from noxious gases, in blasting with all kinds of explo- Hence, if possible, blasting should be concentrated e gaseous products may be removed or sufficiently t comes on (20, 25). Generally 1 lb of explosive as.

imals. Oxygen is converted into CO_2 by breathing. air and of expired air breathed once are shown below. exhales about 7 cu ft per hr. y be consid- ced, or about ired. Thus, is converted all the O in nce, ordinary

	Fresh air	Expired air
O.....	20.90%	16.15%
N.....	79.06	79.90
CO_2.....	0.04	3.95
	100.00	100.00

uivalent to the introduction of 2.6 cu ft per hr of nosphere. As the men are at hard labor most of robably be doubled. A horse or mule requires from man. Expired air is always saturated with mois-

candle or small lamp produces about same effect . Miners' lamps require 14 to 17 cu ft air per min s A I M E, 19, 725).

(Sec 23, Art 3) are most important in collieries. CO_2, and N, the first predominating when much hiefly in the coal seam itself, under pressures rang- t 500 lb per sq in. The quantity evolved is more ily output and fresh face of coal than to the total as is undoubtedly given off from old pillars. Cu mined varies from about 15 to 8 000, or from 0.5 the coal. Averages for the whole coal surface ex- y from 0.2 to 5.0 cu ft per min per 1 000 sq ft. e production of gases may be taken at from 10 to ed, or from 14 to 112 cu ft per min per 1 000 sq ft t 50 to 100% may originate at the face. Within s of this kind may be made on ascertained data, tly unreliable. Moreover, large volumes of gas es liberated in the form of "blowers." In the y contain more gas than bituminous; in Europe very gassy. Quantity of gas probably increases under an impervious cover. Laboratory tests for m samples of coal are of little value for indicating al in place in the mine Reasons are: (a) Large ervoirs in the strata (U S Bureau of Mines, *Tech* pressure in pores of coal is rapidly set free, and a ain but a small part of the gases originally present.

per ton and per sq ft may be established when rate of ed coal to face is known. Assume advance to be 2 ft ace equal 1 ton. Then, each ton daily output requires l requires 0.5 sq ft of face. If all the gas came from the

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working face, the quantity based on output, as above stated, would be fr
ft per 24 hr per sq ft of face, or 0.7 to 4.15 cu ft per min per 1 000 sq ft ol
ing the old surfaces to be 100 times the area of new faces, the maxim
1 000 sq ft would be about 4.15 cu ft per min, over the whole mine.

Relation between firedamp and atmospheric pressure. observed that periods of low barometer, and of increasing outflow from the strata often coincide. The reason is that decreased allows the gas in old workings and in the strata to expand and en Periods of low barometer, especially during rapid falls, should carefully. Exhausted areas or "goaves" of a large mine, perhap with waste, may long continue to give off gas. CO_2 may be produ tion of carbonaceous matter. Goaves act as reservoirs of gas, wh tribute to the ventilating current in case of fall in barometric press (Sec 23, Art 3).

Diffusion of gases. Gases which do not combine chemically will contact with one another, without agitation, even when the heavier gas is bottom. This "diffusive power" assists ventilation and explains the r factory condition of the air in workings where the current is weak. Diffu is opposed by difference in sp gr of the gases, which tends to STRATIFY TH when at rest. Stratification is often pronounced, even within small hei is possible, on entering an old working, where the atmosphere has long b find an explosive mixture of firedamp above the head and a deadly mix damp about the feet, while the air at level of the head is comparatively pu able (Sec 23, Art 2).

3. Dilution of Impurities in Mine Air

Quantity of air required to render impurities harmless: (*a* mines this is based in general on number of men, animals, lights, of explosive used. (*b*) For collieries enough added air must be dilute the gases from the strata. Furthermore, in presence of co with possibility of sudden outbursts of gas from cavities, the dilutio kept far below danger point. Since quantity of gas given off dep on average area of fresh-coal surfaces, the air supply may well be to daily output or number of employes, with due allowance for ga ing from old workings.

Permissible percentage of impurity (Sec 23, Art 6). A mix and air is explosive when CH_4 amounts to from 5.5 to 12.8%. As li to 2% CH_4 may be dangerous, especially in presence of coal du may fall to 10% without causing injury (Haldane), but air is ge sidered unfit to breathe when lights are extinguished, which occur Deep breathing is caused by 3.5% of CO_2 (Thirkell). CO is noxiou sometimes fatal at 0.2%. In general, mine air should not contai 2% of all impurities, which, mixed with 100 volumes normal air, giv (Hygienic effects of mine gases are discussed in Sec 23, Art 5.)

Computation of quantity of air. A synthetic calculation summarize the quantities required to deal with each source of poll pirical estimates are based mainly on number of employes, or on Both methods are uncertain. Even if the requirements of all pos of pollution could be satisfied, an added quantity of air is still allow for leakage and incomplete distribution through the workings

Example of quantity of air required. A mine on an 8-ft seam of steam 15%, and producing but little gas, was slightly short of air at a volume of 2 min, but was fully supplied by 40 000 cu ft. Underground, 197 men and employed, producing 900 tons per 10 hr. Mine was worked by room and pilla

vorking rooms with 125 sq ft of face each, total 10 000 sq ft. main currents traversed 30 000 ft of headings, and swept is are added 320 000 sq ft, in the sides of the rooms, and ft of idle openings; total coal surface exposed, 1 280 000 sq ft ig 1 shows system of ventilation. It is assumed that roof ked-out portions of mine (goaves), covering 750 acres, were rom working places by brattices. (For data on quantities of ntation for quantity of air is as follows:

	= 17	cu ft gaseous pollution per min
	= 17	" " " " " "
	= 8	" " " " " "
1 000 × 14	= 140	" " " " " "
e + 1 000 × 0.1	= 128	" " " " " "
Total	= 310	" " " " " "

required to produce a 2% dilution of impurities, to allow the current, the air required is 310 × 100 = 31 000 cu ft ly with the 29 000 cu ft stated above. Had powder been 3 tons coal, or 30 lb per hr, it would have produced about s per min, requiring an added 500 cu ft of air.

a dangerous quantity of gas the estimate would be:

es, as before,		
17 + 17 + 8	= 42	cu ft gaseous pollution per min
× 112	= 1 120	" " " " " "
ce + 1 000 × 0.8	= 1 024	" " " " " "
Total	= 2 186	" " " " " "

e 218 600 cu ft per min, which seems ample under extreme d surfaces," included in first computation, 92 cu ft per min per inst 1 110 cu ft per min for the dangerously gaseous condition.

cu ft gas
000 sq ft
f outflow
It might
gas came
ut if out-
propor-
face, the
ning only
would be
reat area
rge. In
be more
r of men
with in-
llowance
e second
e rate of
he usual
here re-
is there-
result is

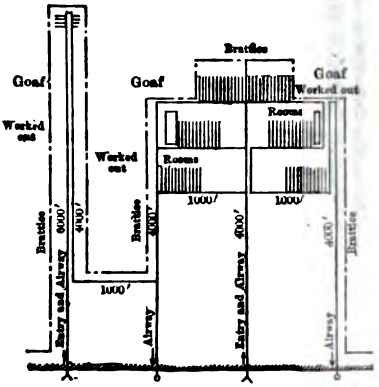

Fig 1. Diagram of System of Ventilation of a Room and Pillar Mine

egal requirements for quantity of air in circula- air per man, in cu ft per min are: Ordinary build- and gold mines, 20 (Truscott); collieries, minimum ng coal producing states in U S are from 100 to 200, 00 to 500 cu ft per min per mule; Prussian govern- eries, 70. For ordinary collieries allow 100 to 500 Headley); in fiery mines 500 to 600 cu ft per mi-

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plus 200 cu ft for each acre of goaf (Phillips); in French practice, for collieries allow between 100 and 200 cu ft per min per ton daily output. Except in extreme cases the Pennsylvania anthracite mines circulate to 1 300 cu ft per man per min, usually about 500, single fans sometim lating 350 000 cu ft per min. Nearly all bituminous mines in Penn from 100 to 1 000 cu ft, usually about 300. Between 100 and 600 cu ft per min will usually suffice for a colliery, but the extreme limits of requ are probably from 50 to 1 500. (For legal allowances for collieries, Dept Mines, Pa, 1910.)

4. Distribution of Ventilating Currents

Air currents. Mine atmosphere is in equilibrium when at same throughout the mine, and mouths of upcast and downcast shafts are at sam tion; or when the mine air has same density as outside atmosphere, rega difference in elevation of upcast and downcast. Circulation is caused b ence of density, or artifi pressure. Ventilating or MOTIVE HEAD thus p is absorbed by velocity rent and by friction surfaces of passages. duction of pressure by means see Art 10; by means, Art 11 to 18.)

Fig 2. Diagram of Ventilation by Single-entry System

Conditions. To proper ventilation, whe natural or artificial me openings to surface, eith or tunnels, are necessary opening becomes an in DOWNCAST; the other, CAST. A single shaft m both purposes, if divide tudinally into two c ments, in which the air flow in opposite direction for safety there should rate openings, as require In coal regions and els In metal mines having shaft, a second opening is often obtained by a connection with work an adjacent mine (16).

Position of ventilating machine. It is usually on the surface, away from dampness of the mine, and convenient to source of power. When electric available, ventilator is sometimes placed under ground. If air courses are hamp ing high motive head, an auxiliary or BOOSTER fan may be placed in the mine. fan may then be smaller, and leakage due to high pressure reduced. An electri propeller fan (Art 16) is compact and convenient for this purpose.

Layout of airways, in relation to haulage ways and working places of is dealt with in Sec 10. The 4 typical systems of ventilation illustr Fig 2 to 5 apply particularly to coal mining. (Scale of these cuts is dis to show clearly the principles.) SINGLE-ENTRY SYSTEM (Fig 2) involve mum expense for airways, and is adapted to relatively small mines. B

nerally prohibited by law because of the inadequate Chief objections to it are: (*a*) there is no air supply

Fig 3. Double Entry, Coursed System of Ventilation

; the fact that the main current sweeps the entire vor of this method of mining.

stem of Ventilation

Coursed and split ventilation. In a coursed system, some air leaks through the doors, due to imperfect construction and passage of mine traffic. Sometimes openings are purposely left around or through the doors to prevent entire current from entering cross-headings, where it would be superfluous. The mine drag or resistance to ventilation is thus reduced and a better distribution attained. This system requires doors on main haulage ways at each "district." A split system requires less motive head for the same quantity of air, since the effect of splitting is to increase the average area for the whole current, and decrease the length. Quantity of air in the splits is controlled by regulators (Art 6) placed at convenient points. The split system requires overcasts ng currents cross. It is necessary when the return night contain a dangerous proportion of firedamp,

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and is generally preferable for large mines, because the foul air from one d
not drawn through any other. Local conditions sometimes alter this r
some regions the coursed system is adopted, because the cost of making c
in hard roof rock is excessive, and the quantity of air and ventilating
required are small. T
state law provides that
than 65 men in the bit
regions, nor 75 men in
thracite region, shall wo
same air current or "s

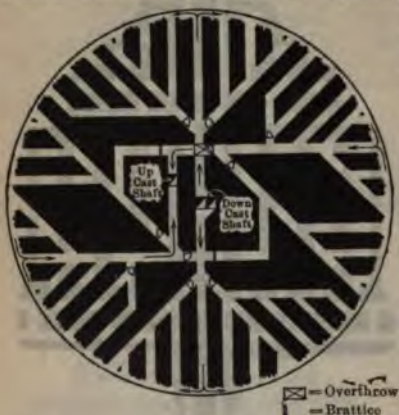

Fig 5. Ventilating System for Longwall Mining

Carrying air to face. In the soft-coal of the eastern U S, air are rarely forced by l pack walls, or other de pass into rooms or The current is carried adjacent haulage road, into the rooms by the tendency to flow throu channels towards a low sure, by diffusion, and turbances caused by t traffic. If these aids sufficient air, canvas are placed at interval the haulage way. In a very "fiery" coal mine it may be necessary to air to the face. In a room and pillar mine this can be done, as in Fig 6, by doors on all rooms of the group to be ventilated, except the first, and cu the last room with an overcast over the haulage way. A pack wall carrie middle of each room forces air to the face. When a seam makes insufficient waste to build a pack wall, wood or non-inflammable brattice cloth is a good substitute.

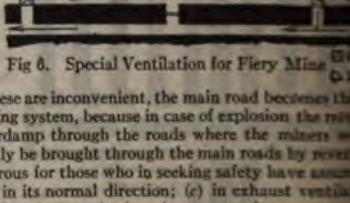
Fig 6. Special Ventilation for Fiery Mine

Exhaust vs blowing system (28). Advantages of exhaust system: (*a*) The foul return current passes through the upcast or outlet direct to the fan, pure air entering the mine by the main haulage ways; (*b*) with blowing fans, haulage ways can not be used as intakes unless doors are installed; and, as these are inconvenient, the main road becomes th This is objectionable with the blowing system, because in case of explosion the re ment of the current forces the afterdamp through the roads where the miners w In emergencies, fresh air may quickly be brought through the main roads by reve current, but this expedient is dangerous for those who in seeking safety have assu the current would be reëstablished in its normal direction; (*c*) in exhaust ventila negative press (depression) at the fan produces in the mine an aver barom pres below that of outside atmosphere; with blowing fans, slightly above. It has b that, with exhaust system, stoppage of the fan increases the mine press; thus exudation of gas. But, this rise of press is small, and its effect in checking outfl (which exists in the coal under considerable press) is probably negligible. It may, h prevent serious flow into working places of accumulations of gas in old workings.

(*a*) If return current contains gas, the fan may be injured by
). (*b*) If intake shaft or entry be wet, it may be obstructed
f ice. In such case, for mines containing little or no gas, the
, or, the fan casing so designed that the fan can be reversed,
winter, and as exhaust in summer (Art 14).

s beyond active currents. In absence of firedamp and
ooms 20 ft wide by 8 ft high can be driven 500 ft, with no
enings, without great inconvenience from bad air; or, an
iven say 250 ft beyond air connections. State mining
o working shall extend more than 60 to 100 ft ahead of
tting through the pillars to adjacent workings for air.

fiery mines. As firedamp is lighter than air, accumu-
occur on ascending side of a ventilating system, where
retard circulation. Therefore where practicable the
mine should ascend, and not be forced downward.

elocity of Ventilating Currents

nts. Moderate velocities are economical of power,
ction and consequent high ventilating pressure. If
tilation only, the velocity is determined chiefly by the
ed and economy in delivery; though it is sometimes
certain minimum velocity at the face, regardless of
to sweep out the gases. If the airway is also a haul-
e effect of high velocity on the health of the miners,
e considered.

on men. A velocity of 1 ft per sec is scarcely per-
he air is in pleasant movement; at 5 ft, and at usual
current is distinctly unhealthful for perspiring men;
reeze; at 50 ft, a high wind.

safety lamps (Sec 23). High velocities drive the
antle, and may ignite a surrounding gassy atmosphere.
nder this test. Davy lamp, having a full gauze un-
es unsafe in a velocity of 8 ft per sec. At 10 ft, the
he flame and at 20 ft, the Mueseler. Marsaut and
emain safe up to a velocity of 40 ft per sec.

naked lights. At 4 ft per sec, a candle flame flick-
ft per sec, an ordinary coal miner's lamp, with a 0.5-
e vegetable oil, can with difficulty be kept lighted on
is at work. But the same lamp, carefully trimmed
velocity of 12 ft per sec. For higher velocities a mix-
kerosene, or even pure kerosene, is used. At 20 ft
probably not be used. With miner's carbide lamps
17 ft per sec, but for ordinary working purposes it

t working face. In non-gaseous mines the utility
he face free of gas does not appear. Hence, in most
es the face only by diffusion and mixture due to move-
fiery mines a velocity of 2 ft per sec across the face

ent against a plane surface at right angles to direction of

$$p = 2 \times \frac{v^2}{2g} w = \frac{v^2}{400} \text{ (approx)}$$

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where p = press, lb per sq ft; v = velocity, ft per sec; g = 32.2
mospheric air, lb per cu ft (= 0.0807 at 32°; 0.07608 at 62° F). F
a partial vacuum is formed back of the plate, where press is as fa
is above on windward side. Smeaton states actual press as: p =
velocity of 10 ft per sec exerts a pressure of about 0.23 lb per sq
a press of about 5.80 lb (Art 7).

6. Brattices, Stoppings, Doors, and Overcasts (4,

Brattices or stoppings confine the air current to the requir
adjacent openings. If the mine produces enough waste m
throughs are often filled with small-size waste, and closed a
with a dry wall of large pieces. If this is not sufficiently a
be stopped with clay, or stones laid in clay. Where no wast
tices are often built of 1-in boards, nailed to light posts a
Brattice cloth of burlap, chemically treated to render it no
resistant to decay, is much used. Since brattices so constructe
remain airtight under slight movements of strata, which wou
the cracks of a wooden partition; an important feature in long
permanent service, brattice may be of brick, from 9 to 18
to their importance and the necessity of insuring against fai

Doors replace brattices in haulage ways to permit passa
They are usually of 1-in boards (sometimes heavier), and are
by gravity. Where possible, they open in a direction against
rent, so that the air press will tend to close them; otherwis
used. On main roads the doors are usually opened and clo
boy." Mechanical doors may be arranged to open automa
proach of a mine car, by means of a system of rods or chains
ing the door with a stop on the rail; motor-operated device

Curtains made of non-inflammable brattice cloth are us
doors, provided they offer sufficient resistance to the air
away with trappers and are not disarranged by movements

Regulators are brattices with an adjustable opening, usu
used to regulate quantity of air on branch air currents. F
(Art 4), a regulator is needed on every split except that wh
est motive press. (For their resistance, see Art 7; for their

Overcasts (overthrows or bridges) carry one air curre
They are usually made by blasting down part of the haul
building an airtight conduit in the space so formed. Over
wood, or in important places of brick or reinforced concr
is more convenient to pass the air course under the haulag
"underthrow"; these are generally objectionable, because
lect in them, and may completely obstruct the flow of air.

7. Velocity and Friction Head

Notation in this article (except as otherwise stated):

- a = area of air course or orifice, sq ft
- c = constant (Table 2)
- g = acceleration due to gravity, 32.2 ft per sec
- h = velocity head or press, measured by height of a colum
  produce that press, ft
- k = coefficient of friction (Table 2)
- p = air press, lb per sq ft = Wh
- Q = quantity of air, cu ft per min

, cu ft per sec
air way, ft (= area + perimeter)
exposed to friction of air current, sq ft
ent, ft per min
" " sec
f air, lb (= 0.075 ± = approx wt of mine air under ordinary

water gage = $p + 5.2$

ressure and velocity. Pressure head equals absolute of air, or $h'' = p + W$. Velocity head $h = v^2 + 2g$. At rent, unaffected by friction, the sum of these two heads $(v^2 + 2g)$; hence, every change in the velocity of air, to point, is accompanied by a corresponding change in ating apparatus is passed. This law governs the devel- y of air currents (entry head), flow through an orifice tution of velocity to press as illustrated by the adjutage , no current is free from friction against sides of chan- eddy currents; hence, the principle can never be per- 5).

ifice in a thin plate is less than its actual area by reason of rrent (vena contracta, Sec 38) on the low-press side. The fore $q = 0.65\, a \times \sqrt{2gh}$, where $h = (h' - h'')$, the difference atc, is the velocity head, and 0.65 the ratio between effective

Let air be flowing through a tube *ghef* of varying section n between two bodies of quiescent air, and the pressures to on being h' and h''. If h' be *ef* of tube be closed, press h' pened, the difference between v at *ef*, in direction of arrow, $h'')$. Also, $v \times$ area $ef = v' \times$ y at *cd*. Since *cd* is smaller v. If the density of air were absent, area *cd* could be con-

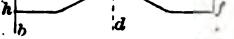

Fig 7. Adjutage of Venturi

hen the whole available press would be absorbed by velocity, But, in passing from *cd* to *ef*, part of the velocity is restored to ncrease of area is necessary to effect this restitution of head. ice eddies which convert velocity into heat instead of into press.

neasured with pitot tube or anemometer, or approxi- ed at which a lamp must be moved to keep flame ver- te at which powder smoke moves The anemometer ocity measurements should be made in a gallery which ndred feet on each side of the section measured. Sev- at different parts of the section, are averaged for mean ocity in a straight air course occurs at middle of sec- 1.2 to 1.4 times the mean velocity. Position of ob- not to obstruct or disturb the air current. If he must rea is deducted from cross-section in computing volume; ways prevent accuracy.

J-shaped glass tube about 0.75 in diam, open at both ends, ntally. Lower part of U is filled with water. Horizontal current. Press due to velocity is registered by difference in and measured on the scale. The velocity is

$$= \sqrt{2gh} = \sqrt{2g\frac{5.2\,w}{W}} = 18.3\sqrt{\frac{w}{W}}$$

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= quantity of air, cu ft per sec
= mean radius of air way, ft (= area ÷ perimeter)
= area of surface exposed to friction of air current, sq ft
= velocity of current, ft per min
= " " " " " sec
W = wt of 1 cu ft of air, lb (= 0.075 ± = approx wt of mine air under ordinary conditions)
= press, inches of water gage = $p \div 5.2$

tion between pressure and velocity. Pressure head equals absolute ivided by density of air, or $h'' = p \div W$. Velocity head $h = v^2 \div 2g$. At ts in a steady current, unaffected by friction, the sum of these two heads nstant, $h' = h'' + (v^2 \div 2g)$; hence, every change in the velocity of air, asses from point to point, is accompanied by a corresponding change in rovided no ventilating apparatus is passed. This law governs the devel- of initial velocity of air currents (entry head), flow through an orifice n plate, and restitution of velocity to press as illustrated by the adjutage turi. In practice, no current is free from friction against sides of chan- ernal friction, or eddy currents; hence, the principle can never be per- xtemplified (14, 16).

tive area of an orifice in a thin plate is less than its actual area by reason of traction of the current (vena contracta, Sec 38) on the low-press side. The discharged is therefore $q = 0.65\, a \times \sqrt{2gh}$, where $h = (h' - h'')$, the difference on each side of plate, is the velocity head, and 0.65 the ratio between effective al area of orifice.

age of Venturi. Let air be flowing through a tube *ghef* of varying section *ab* being a partition between two bodies of quiescent air, and the pressures to right of the partition being h' and h''. If h' be ter, and the mouth *ef* of tube be closed, press h' hin tube. If *ef* be opened, the difference between " produces velocity v at *ef*, in direction of arrow, g to $v = \sqrt{2g(h' - h'')}$. Also, $v \times$ area $ef = v' \times$ where v' = velocity at *cd*. Since *cd* is smaller v' is greater than v. If the density of air were and fluid friction absent, area *cd* could be con-

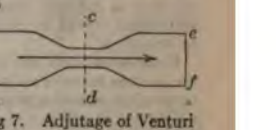

Fig 7. Adjutage of Venturi

ntil $v' = \sqrt{2gh'}$, when the whole available press would be absorbed by velocity, ffecting discharge. But, in passing from *cd* to *ef*, part of the velocity is restored to ead. A gradual increase of area is necessary to effect this restitution of head. anges in area produce eddies which convert velocity into heat instead of into press.

nt velocity is measured with pitot tube or anemometer, or approxi- y observing speed at which a lamp must be moved to keep flame ver- by observing rate at which powder smoke moves. The anemometer onvenient. Velocity measurements should be made in a gallery which t for several hundred feet on each side of the section measured. Sev- surements, taken at different parts of the section, are averaged for mean Maximum velocity in a straight air course occurs at middle of sec- is usually from 1.2 to 1.4 times the mean velocity. Position of ob- ould be such as not to obstruct or disturb the air current. If he must he current his area is deducted from cross-section in computing volume; disturbances always prevent accuracy.

be (Fig 8) is a U-shaped glass tube about 0.75 in diam, open at both ends, eg turned horizontally. Lower part of U is filled with water. Horizontal nted against air current. Press due to velocity is registered by difference in in the two legs, and measured on the scale. The velocity is

$$v = \sqrt{2gh} = \sqrt{2g\frac{5.2\,w}{W}} = 18.3\sqrt{\frac{w}{W}}$$

where w = difference in water level, in; W = wt of atmosphere. The end towards current has an annular metal cap with sharp edges, to avoid formation

Fig 8. Pitot Tube

Plane of the opening at other end must be parallel to c avoid velocity press. This end may be protected with a of cotton, or better by a thin diaphragm, covering but n the opening. Though these devices prevent serious esc the water columns, readings by the naked eye are inaccura use magnifying glass. By sloping the tube, a greater dis in direction of its axis is obtained, thus facilitating readin for static and velocity press may be placed in the air c connected by pipes with water columns set at a point n for reading. If the horizontal opening of the tube be tur from the direction of current, a partial vacuum is produ retically equal to the press produced by the velocity in position. The observed depression is about half that when the horizontal end current; attributable to eddy currents about the end of the tube.

Anemometer usually employed in mine measurements is a delicate wi oblique blades (Fig 9). Axis of wheel is geared to an indicator, which regist matically the lineal feet of flow of air current during period of observation cator can be disconnected from the wheel by a small lever, so that registration can be accurately started and stopped. Time to attain constant speed must be allowed before connecting the indicator. Velocity is computed by dividing difference in readings at beginning and end of observation by elapsed time.

Fig 9. Anemome

Errors may be caused by variation in friction of the wheel, or by slight bending of the vanes. A table showing error at different velocities is furnished by maker. Instrument should be returned to maker occasionally, to be tested.

Experiments with the wheel anemometer show that the relation between rotations of wheel and velocity of current is expressed approx by the straight-line formula, $v = an + b$, in which, v = velocity, n = number of revolutions in unit of time, and a and b are constants for the instrument.

Another type of anemometer has a vane 3 or 4 in square, pivoted in a frame held at right angles to air current, the velocity is indicated by the vertical angle vane is raised. A quadrant and index at side indicate velocity without comp An attached level enables instrument to be held vertical. This type is sensi convenient, but oscillation of the vane interferes with accurate reading.

Relation between pressure and friction. Friction head or press, r to overcome friction of the air against sides of airways, is expressed equation

$$p = ksV^2 \div a$$

where p is difference in absolute press between ends of air course. To tion head for a series of air courses of different sections is equivalent to t of individual friction heads. If areas of the different sections vary be they may be averaged.

Value of coefficient of friction k depends upon character and ro of sides of airways, upon their curvature, and upon size and form of cr tion. Effect of first two factors is measured by a numerical scale of ro denoted by c in Table 2. Effect of the sectional area is measured by co its mean radius R. Value of k assigned by Atkinson (0.000 000 021 most cases, too great, as shown by Table 2 (1, 10).

. Coefficients of Friction in Airways

5	1.0	2.0	3.0
	Values for $k \times 10^8$		
213	0.165	0.127	0.109
127	0.329	0.254	0.219
640	0.494	0.381	0.328
854	0.658	0.509	0.437
067	0.823	0.636	0.546
281	0.987	0.763	0.656
494	1.152	0.890	0.765
707	1.316	1.017	0.874
921	1.481	1.144	0.983
134	1.646	1.272	1.092
348	1.810	1.399	1.202
562	1.975	1.526	1.311
775	2.140	1.653	1.421

headings, $c \times 10^8$ varies from 0.20 to 0.60
bare rock walls, $c \times 10^8$ " " 0.60 " 1.60
dings, $c \times 10^8$ " " 1.00 " 2.60
whole mine, $c \times 10^8$ " " 1.00 " 1.60

rkings, the smaller values of c may be used; for rough, r values. Values 1 to 1.6 for mine average allow for airways (Art 8). The other values for c allow for no

The formula for friction is based on assumption that mine ft. Since k varies directly as the density, correction for a is found from $p' = pW + 0.0749$, where p' is press corre- W.

s in airways causes loss of head equal to height of air velocity head, multiplied by a coefficient determined

ient $= 0.9457 \sin^2(\frac{1}{2}i) + 2.047 \sin^4(\frac{1}{2}i)$

tween sides of bend. For a 90° bend, coeff is 0.98; for 45°

$= 0.98\,(v^2 + 2g)$; for 45°, $h = 0.18\,(v^2 + 2g)$

v = velocity of current. Thus, nearly all the velocity head is about 0.4 in two 45° bends.

resistance by increasing current velocity. Press lost gulator is that which is due to increased velocity in the velocity with which the air approaches the regulator.

ch = q + sectional area of heading, v' = velocity through orifice

$$p = \frac{0.075}{2g}(v'^2 - v^2)$$

v', found from above equation, may be used to compute size for "vena contracta," $a = q + 0.65\,v'$.

ng Pressure. (Notation same as in Art 7.)

ired to circulate an air current, is that which is neces- on and produce velocity. VELOCITY HEAD at inlet of es press necessary to develop the mean velocity due

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entry resistance (Art 7) If there be no subsequent contraction o
way, and no restitution of velocity to head, this original loss is t
FRICTION HEAD (Art 7) is measured by the decrease in press of ve
rent between mine
let. For an airwa
section (air at 0.07

$$p = \frac{1.5 \times 0.075}{2g}$$

For airway of irre

$$p = \frac{1.5 \times 0.075}{2g}$$

where v' is velocity
Σ the sum of resis
velocities V', in d
of the airways.
usually ranges fr
6.0 in water ga
head is so small, c
friction head, tha
ally be neglected,
high velocities occ
courses. Coef of
ance (1.5) is also c

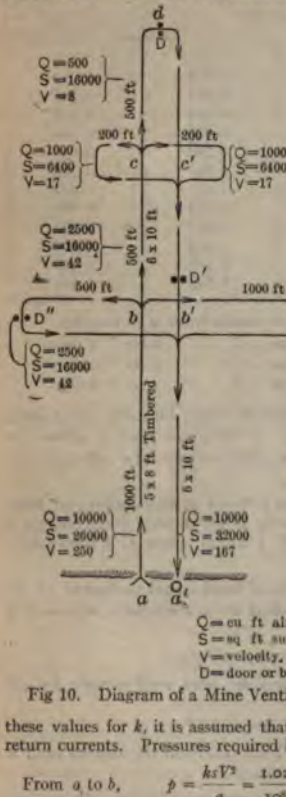

Fig 10. Diagram of a Mine Ventilating System

Example. In Fi
velocity of entry at
min, and press requ
it = (1.5 × 0.075 ×
32.2 × 60²) = 0.03 lb
a roughly-timbered
ing, $c \times 10^8$ is taken
R for first 1 000 ft is
(nearly), $k = 1.02$ +
maining airways, inc
at turns, $c \times 10^8$ m
1.3, and since R is
0.8 ÷ 10^8 (approx).

these values for k, it is assumed that there is practically no leakage betwe
return currents. Pressures required in different parts of mine are:

From a to b, $p = \frac{ksV^2}{a} = \frac{1.02}{10^8} \times \frac{26\,000 \times 250^2}{40} = 0.4150$ lb per

" b' to a' " $= \frac{0.8}{10^8} \times \frac{32\,000 \times 167^2}{60} = 0.1190$ " "

" b to c or c' " b' " $= \frac{0.8}{10^8} \times \frac{16\,000 \times 42^2}{60} = 0.0038$ " "

" c to c' via d " $= \frac{0.8}{10^8} \times \frac{2 \times 16\,000 \times 8^2}{60} = 0.0003$ " "

" c to c' via cross entry " $= \frac{0.8}{10^8} \times \frac{2 \times 6\,400 \times 17^2}{60} = 0.0005$ " "

" b to b' via left entry " $= \frac{0.8}{10^8} \times \frac{2 \times 16\,000 \times 42^2}{60} = 0.0076$ " "

" b to b' via right entry " $= \frac{0.8}{10^8} \times \frac{2 \times 32\,000 \times 83^2}{60} = 0.0588$ " "

equired between c and c' (0.0005 lb) exceeds press required in gulator D is provided (Art 6 and 7). Difference in press re- by inner part of mine is equal to sum of differences between nd b', or $0.0005 + 2 \times 0.0038 = 0.0081$ lb per sq ft. As this te the necessary quantity of air in first right entry, press be- nade equal to 0.0588; hence, resistance of main entry current nt by placing a regulator D' at some point on main return air ss entry. Entire press p required for the mine = 0.0588 + 0.6228 lb per sq ft, which divided by 5.2 gives 0.123 in water s press is lost by friction in the timbered heading, only 5% pro-

e in above example: Assume that of the 5 000 cu ft air enter- y 2 000 reaches last breakthrough. If the reduction in volume to distance traversed, the initial velocity is 83 ft per min; at and at end of heading, 33 ft per min. Average of squares of which is used for V^2 in friction formula. Hence, total friction e right heading, is

$$\times \frac{0.8}{10^8} \times \frac{32\,000 \times 3\,742}{60} = 0.0319 \text{ lb per sq ft,}$$

ure previously obtained. Reduction of press by leakage causes cies between computed and actual water gage. Another cause ugh the rooms. Some leakage is unavoidable, and often not number of men on any air course decreases as the face is ap- and percolation exist, a general value for k may be assumed at or airtight conditions, and the computation made as though n existed (1, 4, 8, 13, 14).

nches of a ventilating current not controlled by regu- oe quantity of air in current before splitting; Q' and Q'', n each of two splits, whence $Q = Q' + Q''$. Since same on both splits between points of separation and reunion d, which is always very small),

$$p = \frac{k's'\left(\frac{Q'}{a'}\right)^2}{a'} = \frac{k''s''\left(\frac{Q''}{a''}\right)^2}{a''}$$

r to one split, the seconds to the other. These equations nd Q''.

of 10 000 cu ft per min is split into 2 open airways: one, 100 , with section of 5 by 8 ft; the other, 200 ft long, slightly curved, of 6 by 10 ft. In first airway, c may be taken at 1.2, and as ^{8}. In the second, $c = 1.0$, $R = 1.9$, $k'' = 0.655 \div 10^8$. Also 600, $s'' = 6\,400$, and $Q = 10\,000$. Substituting in equation,

$$\left(\frac{Q'}{Q''}\right)^2 = \frac{k''s''(a')^3}{k's'(a'')^3} = \frac{0.655 \times 6\,400 \times 40^3}{0.875 \times 2\,600 \times 60^3}$$

$$\frac{Q'}{Q''} = \sqrt{0.545} = 0.74, \text{ and } \frac{Q}{Q''} = 1.74.$$

5 750, and $Q' = 4\,250$.

artificial and ventilating pressure. Centrifugal fans the press produced, measured from air at zero velocity ess of motionless air at the outlet. Where there is a veloc- arent press exceeds this "zero to zero" press by an amount y to produce the velocity.

ed between zero and zero velocity, h_o the absolute press of out- absolute press and velocity of air at intake, and h_1 and

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at the outlet. Let no unbalanced densities exist. Then, if ventilat
plied by a blowing fan, $h_a = h_2$, and $h_0 = h_1 - h_2 = h_1 - h_a$ = mine fric
an exhaust fan is used, $h_a = h_1 + \frac{v_1^2}{2g}$, and $h_0 = h_a - \left(h_2 + \frac{v_2^2}{2g}\right)$, whe

$$h_0 = (h_1 - h_2) + \frac{v_1^2 - v_2^2}{2g}$$

The value of $(h_1 - h_2)$ is identical in the two cases.

Total ventilating press is sum of the friction and entry heads, or (
For a blowing fan,

$$(h_1 - h_2) + \frac{v_1^2}{2g} = (h_1 - h_a) + \frac{v_1^2}{2g} = h_0 + \frac{v_1^2}{2g};$$

and for an exhaust fan,

$$(h_1 - h_2) + \frac{v_1^2}{2g} = (h_a - h_2) = h_0 + \frac{v_2^2}{2g}$$

The apparent press or water gage reading in first case is $(h_1 - h_a)$, in sec
If unbalanced densities exist, the value of h_0 must be increased or di
ing as they act against the direction of the current or with it (Art 10).

Equivalent orifice and orifice of passage. Ventilating
directly as the square of the velocity, or quantity of air deliv
the law followed by flow through an orifice in a thin plate, min
be expressed in terms of the area of an EQUIVALENT ORIFICE. T
OF PASSAGE is similarly used to express the resistance of the v
selves (10, 11). The formula for computation is

$$a = \frac{Q}{0.65 \times 60 \times \sqrt{2gh}} = (\text{approx}) \frac{0.0004\,Q}{\sqrt{w}}$$

where a is the required area of orifice, Q the quantity of air flo
loss in press in mine or ventilator. Unbalanced weights of air
nated (Art 10).

The equivalent orifice sometimes includes both friction and velocity
artificial ventilation is investigated the equivalent orifice computed fro
ance alone is more conveni
all the velocity loss is includ
passage, and the true resist
system is the sum of the los
orifice, considered as thoug
tween large bodies of motio

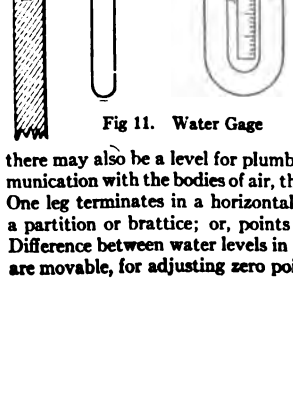

Fig 11. Water Gage

Example. A mine havin
of 0.75 in water gage passes
min. Here $Q = 50\,000$, $w =$
$0.75 = 3.9$ lb per sq ft; when
Substituting,

$$a = 50000 \div (0.65 \times 60\sqrt{2 \times 3}$$

Water gage (Fig 11) is
ing ventilating pressure.
shaped glass tube, partly
and somewhat resemblin
(Art 7). Attached to a lig
and between legs of the U i
there may also be a level for plumbing the gage. The open end
munication with the bodies of air, the difference in press of which i
One leg terminates in a horizontal brass thimble, which is inse
a partition or brattice; or, points of observation may be con
Difference between water levels in the two legs is read on the sc
are movable, for adjusting zero point to the water level in one

d against velocity press. In some gages the end of the the opening being set loosely in a small cavity in the ing a test the gage should be set so that the direction ndicular to axis of the thimble end of tube, which end n cotton to prevent oscillation of water level.

e is useful for obtaining a continuous record. It is like a ng of a circular paper chart, revolving once in 24 hr. Radial d night. An index pen describes a line varying in distance ng to air press.

ge for effect of difference in velocity of current at points of if accurate results are necessary. Correction for weight of age from points of measurement is required only when these s are at different elevations.

solute press at points tested, each represented by a branch gage,
absolute press (greater than A and A' by the press absorbed ocity of current (Art 7),
water in legs of gage, corresponding to A and A',
to weights of air in the connecting pipes, if any,
responding to current velocities at points tested.

in inches of water, G being the gage reading. Then:

$= p' - h'$, $w = A + t$ and $w' = A' + t'$

$A - A') + (t - t') = (p - p') - (h - h') + (t - t')$

ge are short and no leading pipes are necessary, $t - t' = 0$.
red are equal, $G = (w - w') = (A - A') = (p - p')$.
and the current velocity is expressed by $h = Wv^2 \div 334.9$,
y velocity, in of water; W = wt of air in motion, lb per cu ft
elocity, ft per sec. Let H be press of air column (in of water)
ference in level between points of measurement, and which has
. H is positive when current flows downward, negative when
otation refers to the point from which the current flows, then
friction and velocity is $(p - A') \pm H$; and the friction press is
$A') \pm H$. If the first point is in the quiescent air outside the
ess $= (A - A') \pm H$, and friction press must be computed by
f the mine air is not all of same density, the press H, required
ne current, must be computed by taking the algebraic sum of
ng with (+) or against (−) the current (Art 10).

g current = press × velocity × area of air course, $= paV$
city head) ksV^3 = ft lb of work per min. Horsepower of
$Q \div 33\,000$, p including both velocity and friction heads.

ating current in a mine passing 100 000 cu ft per min, at 2 in
$2 \times 5.2) \div 33\,000 = 31.6$ hp.

Mine Atmosphere (See also Sec 23, Art 1.)

depends upon its temperature and pressure. When the ight of a column of mercury, some ambiguity results, ercury varies with its temp and with the intensity of ng with latitude and elevation.

$$W = \frac{1.32527\,B}{459.2+t} = \frac{2.69819\,P}{459.2+t}$$

of pure dry air, lb; B = abs press, inches of mercury; P = = temp, deg F. The weight at Paris of a cu meter of dry air, ercury, is 1.293187 kg, or 0.08072 lb per cu ft. A mercurial 29.92 in high, and at 0° C, weighs 14.697 lb per sq in. This is ," taken as 14.7 for ordinary calculations.

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Weight of aqueous vapor. When aqueous vapor has same te which it is formed, it is "saturated"; at higher temp, it is "superh aqueous vapor is a perfect gas, and the relations between its pre are similar to those existing for other gases (Sec 39). Density c compared with air at same temp and press is 0.622. For every pr there is a corresponding weight, and a corresponding temp, which is If w is the wt of saturated vapor in lb per cu ft, and p its press in l Brownlee) $w = (p^{0.941}) + 330.36$.

Table 3. Pressure and Temperature of Saturated A

Press measured by height of column of mercury at 0° C

Temp C	Press, mm	Temp F	Press, in	Temp C	Press, mm
−32	0.32	−25.6	0.013	45	71.39
−20	0.93	− 4	0.037	50	91.98
−10	2.09	0	.....	55	117.47
− 5	3.11	14	0.088	60	148.70
0	4.60	23	0.122	65	186.94
5	6.53	32	0.181	70	233.09
10	9.17	40	0.248	75	288.51
15	12.70	50	0.362	80	354.64
20	17.39	60	0.517	85	433.04
25	23.55	70	0.733	90	525.45
30	31.55	80	1.024	95	633.77
35	41.82	90	1.410	100	760.00
40	54.91	100	1.917		

Note. The temps C in first column do not correspond with ten third column, except at boiling point (100° C = 212° F), and belo

Quantity of moisture. Saturated air contains all the aq absorb. When two or more gases or vapors are mixed, the p equals the sum of the pressures each would exert if occupyi alone. Thus, if a mixture of air and watery vapor together c exerts a press proportional to the ratio its volume bears to mixture. If H be the barom press and F the press of watery va of mercury), the porportion of vapor present is $F \div H$, and of

So long as the temp of the mixture remains above temp of conde vapor at press F, this proportion remains constant. Relative humic saturation, is the ratio of quantity of vapor present to that which absorbing at its actual temp t, in deg F. The ratio of the vapor mixture, in saturated air, is $f \div H$, where f is the press of saturat ing to its actual temp t, and the relative humidity is $(F \div H) \div (f$ dew point is the temp at which the vapor in any mixture begins to co when t corresponds to the temp of saturated vapor at press F, and relative humidity = 1 (for values of f see Table 3).

Weight of moist air. Weight of a unit volume is expressed by

$$W = \frac{1.293}{1 + at} \times \frac{H - 0.378\,pf}{760}$$

where W is wt in gm per liter, or kg per cu m of moist air; a is a c t, temp in deg C; H, height of barometer in mm of mercury; f, press vapor corresponding to temp t (Table 3); and p percentage of saturat by 100 (Deschanel). In English measures, if H and f be expressed and t in deg F, the wt W in lb per cu ft becomes

$$W = 0.08072 \times \frac{491}{459 + t} \times \frac{H - 0.378\,pf}{29.922} = \frac{1.3232}{459 + t} \times (H -$$

For dry air put p equal to zero. This formula assumes that densi is 0.622 when air is unity. Table of wt of dry and moist air is given

gases vary directly as the press and inversely as the abs cu ft in lb, at 32° F and 29.922 in barom, then wt per barom b, is

$$w = W \frac{491\,b}{(459 + t) \times 29.922}$$

Densities and Weights of Common Gases

	Chemical symbol	Density. Air = 1	At 760 mm merc, 0° C; kg per cu met	At 29.92 in merc, 32° F; lb per cu ft
.....	O + 4 N	1.00000	1.2932	0.08072
.....	O	1.1056	1.4298	0.0893
.....	H	0.06926	0.08957	0.005592
.....	N	0.97137	1.25615	0.07841
.....	CO	0.9569	1.2344	0.07706
.....	CO_2	1.52901	1.9774	0.1234
.....	CH_4	0.559	0.727	0.04539
.....	H_2S	1.177	1.522	0.09501
.....	NH_3	0.5967	0.7697	0.04805

nixtures can be computed when the proportions of the . The weight of moist air is found as above, and the ous vapor is regarded as a single element. This is not n the humidity is determined before the air is polluted can seldom be done, because mine conditions usually the error is negligible.

find wt of mine atmosphere at 50° F and 29 in barom, con- CO_2, and 1% CH_4. Weights per cu ft are as follows:

$$0.08072 \frac{491}{(459 + 50)} \times \frac{29 - 0.378 \times 0.75 \times 0.362}{29.922} = 0.07519$$

$$0.1234 \frac{491}{(459 + 50)} \times \frac{29}{29.922} = 0.11536$$

$$0.04539 \frac{491}{459 + 50} \times \frac{29}{29.922} = 0.04243$$

moist air	@ 0.07519	weighs	0.0736862 lb
CO_2	@ 0.11536	"	0.0011536 "
CH_4	@ 0.04243	"	0.0004243 "
of mixture		weighs	0.0752641 "

Direct readings are usually sufficient. For accurate work educe the apparent height of column to standard conditions. es observed height to height at 32° F. Length of scale also rection for capillarity of tube depends upon diameter, and een boiled in the tube. Correction for capacity is made when ustable to surface of mercury in cistern. Capillary and capac- es obviated by method of graduating scale. Index errors are osition of zero of scale. Correction for gravity reduces column andard latitude.

actuated by changes in form of a thin metallic vessel, partly ent is transmitted to an index, which shows the air press on a es from about 2 to 5 in diam, large sizes being the more accu- ompensated for temp. Scale is graduated to read altitude in corresponding with mercurial barometer. Aneroid should be

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tested occasionally by comparison with a mercurial barometer. N
ment is standardized for horizontal or vertical position. Before readi
gently tapped to dislodge the index, which may be retarded by frictio

Thermometer should be kept dry, and readings continued long en
the mercury or alcohol has become stationary. Instrument must n
surrounding objects, as lamps, persons, or walls of workings.

Wet and dry-bulb hygrometer, for determining humidity, cons
thermometers, mounted on a frame side by side. Bulb of one is cove
moist by a cotton wick leading from a small vessel of water placed ben
from the moistened bulb produces a depression of temp, so that this
lower than the dry bulb. Instrument is mounted so as to allow fre
around the wet bulb. Elevation of the vessel of water should be such a
attraction of the wick to keep the muslin moist but not dripping.
when the wick cannot be used, the bulb must be dipped in water
reading to freeze the film of water on bulb, which, covered with a
allowed to stand until temp becomes constant; then readings are take
no air current exists, the instrument should be fanned during obser
be the readings in deg F of the dry and wet bulbs respectively; f and
corresponding to saturation at those temps (Table 3), in of mercury;
ing, in; and F the tension of the vapor in the air at actual temp t (see
ture); then, for temps between freezing and boiling point,

$$F = \frac{h}{30}\left[f' - \frac{t - t'}{87}\right]$$

and for temps below freezing, $$F = \frac{h}{30}\left[f' - \frac{t - t'}{96}\right]$$

and percentage of saturation $= F \div f$.

Some hygrometers are accompanied by charts, carrying an index giv
ity and dew point on inspection. (See also Sec 23, Art 1.)

Example. If reading of dry bulb be 50° F and of wet bulb, 40°
for $t = 50$, $f = 0.362$, and for $t' = 40$, $f' = 0.248$.

If $h = 30$ in, $F = 0.248 - \frac{(50 - 40)}{87} = 0.133$ in, and saturation =

The dew point, or temp of saturated vapor corresponding to tension
polation in Table 3 to be 25° F, corresponding to 0.133 in mercury.

10. Natural Ventilation

Unequal densities of the air, due mainly to different temps,
ing constituents, create unbalanced pressures when the curr
of different elevation. Hence, if temp is higher in the mine
usually passes in at the lower opening and out at the upper.
mines, and some small collieries, are ventilated by natural circu

A downcast current tends to establish itself in a pump shaft, whic
and cooler than other shafts. Specific gravity of air is decreased b
creased by contact with the cool column pipe, and dripping water te
down mechanically. Feeble currents will form in single shafts, ever
partition to divide the currents; cool air generally flowing down near t
in a wet shaft) and warm mine air rising in the center. In an inclined
current often forms on the floor and an upcast near the roof.

Density pressure is computed by assuming the air curre
closed circuit; beginning at mouth of downcast, passing thr
top of upcast, then at an insensible velocity back to mouth
CDBAC, Fig 12). This circuit is divided by its highest and l
two legs, the difference in the weights of which tends to crea
the heavier to the lighter column. Where the current has
each split is part of a separate circuit which comprises also the

ircuits, having different pressures, except where they (as in the 2 circuits, *CEDAC* and *CDBAC*, Fig 13). er sq ft) is the difference in weight of two air columns, he circuit. Each and a height equal sity of each being rresponding leg at circuit the venti- the friction and 8). Several simul- from which the different parts of

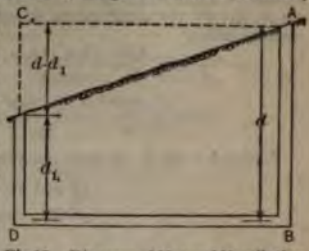


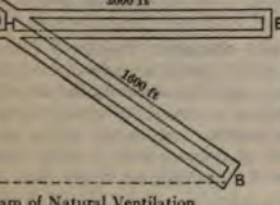
Fig 12. Diagram of Natural Ventilation

h 2 shafts of unequal oduced by difference and *CD*. If depths , t_1, and t_2, the respective temps in the shafts and outside air, (in of mercury) and air is pure and dry,

$$p = .13\,B\left(\frac{d_1}{459.2+t_1}+\frac{d-d_1}{459.2+t_2}-\frac{d}{459.2+t}\right)$$

equated to fric and veloc resistance. If p be positive, shaft .53 is the wt in lb of 459.2 cu ft of air at 0° F and 1 in barom.

Example 2. In Fig 13 let AB be a slope, and CDE a level, with connecting air courses. Cross-sec of all passages is 6 by 8 ft, and coeff of friction is $0.8 \div 10^8$. Average barom press between A and B is 30 in. Temp of outside air, 32° F, and its humidity 50%. Average temp and humidity of the slope from A to B, 55° F and 95%. Under these conditions the mine air will be lighter than outside air, and a current will flow in at C and out at A. Part of the air will pass into the level by first circuit, *CEDAC*, and part by the second circuit *CDBAC*. As *CE* and *DE* are level, the density of the air in them is of no effect. Influence of incoming air makes the average temp and humidity of slope *DB*, 50° F and 90%.



am of Natural Ventilation

weights of air under these conditions are (Art 9):

umidity	50%, barom 30 in,	0.081 lb per cu ft
"	90 " " "	0.078 " " " "
"	95 " " "	0.077 " " " "

r columns of the first circuit:

air @ 0.081 lb per cu ft	=	40.5 lb per sq ft
" @ 0.077 " " " "	=	38.5 " " " "
rence = motive press p'	=	2.0 " " " "

ircuit:

air @ 0.081 lb per cu ft	=	40.5 lb per sq ft
" @ 0.078 " " " "	=	62.4 " " " "
t of column	=	102.9 " " " "
air @ 0.077 lb per cu ft	=	100.1 " " " "
rence = motive press p''	=	2.8 " " " "

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If Q be total cu ft of air circulating per min, and q' and q'' l
second splits, respectively, the velocities are: $Q \div 48$, $q' \div 48$, an
rubbed by total current in CDA equals 56 000 sq ft; by air in fir
ft, and by air in second split, DBD, 89 600 sq ft. If both splits
lators, Art 6), the simultaneous equations to determine Q, q',
velocity head):

$$p' = \sum \frac{ksV^2}{a} = \frac{k \times 56\,000}{48} \times \left(\frac{Q}{48}\right)^2 + \frac{k \times 112\,000}{48} \times$$

$$p'' = \sum \frac{ksV^2}{a} = \frac{k \times 56\,000}{48} \times \left(\frac{Q}{48}\right)^2 + \frac{k \times 89\,600}{48} \times$$

Putting $k = 0.8 \div 10^8$, these equations reduce to:

$$5.6\,Q^2 + 11.2\,q''^2 = 2.76 \times 10^9$$

$$5.6\,Q^2 + 8.96\,q''^2 = 3.86 \times 10^9$$

$$Q = q' + q''$$

whence, $Q = 20\,100$, $q' = 6\,700$, and $q'' = 13\,400$ cu ft per min.

The friction pressures in different parts of mine (Fig 13) are:

From C to D (or D to A), $\frac{ksV^2}{a} = \frac{0.8}{10^8} \times \frac{28\,000 \times 17.6 \times 10^4}{48}$

" D to E (or E to D), " $= \frac{0.8}{10^8} \times \frac{56\,000 \times 1.95 \times 10^4}{48}$

" D to B (or B to D), " $= \frac{0.8}{10^8} \times \frac{44\,800 \times 7.8 \times 10^4}{48}$

The head produced by the different densities between A and l
ft. Of this, the air moving in CD and DA absorbs 1.64 lb, le
equivalent to water gage of 0.069 in, read on an instrument plac
crosscut at D. Also, this pressure measures the friction in level
per sq ft). The split in the slope is assisted by a difference of
amounting to 0.8 lb per sq ft. As the ventilating press at D = c
DBD is $0.36 + 0.8 = 1.16$ lb per sq ft, which equals the friction

Seasonal and daily changes. Weight of downcast
depends upon temp of the outside air as well as of the m
temp is fairly constant, and that of the outside air subject
variation, the strength of a natural air current fluctuates
its direction between summer and winter, and in warm
and day. At periods of reversal there is often tempor
when the temp curve of the outside air crosses that of the

Currents started artificially, when mine atmosphere i
change the temp in some parts of the mine. If this change
so as to produce flow in direction of the temporary cu
becomes permanent. Thus with 2 shafts of equal dep
winter is cooled by the incoming air and a current once sta
as the conditions remain unchanged.

Artificial ventilation is assisted by natural press, when
motion in same direction. If h_0 be the zero to zero pr
system (Art 8), n the density (natural) press, and v the
ventilator; then, total ventilating press $= h_0 + (v^2 \div 2g)$
sign when the artificial current opposes the natural curren

11. Furnace Ventilation

Field of use. When the natural difference in density
cast currents is insufficient to produce the required circu

ficial heat. This ancient mode of ventilation, applicable
ause of cheapness of fuel, was formerly in more general
t of the U S, the law forbids new installations, though
ometimes be operated with consent of district mine in-
ld not be used in gaseous mines. The added heat must
voiding possible reversal of direction of flow, as between
rt 10). To increase height of air column affected, a
t top of upcast shaft (33). For temporary use a FIRE-
m by 36 in high, built of wrought-iron bars, say 0.5 by
d near bottom of shaft.

types: open, outside-current, and dumb-drift. They
design, varying chiefly in their relation to the mine cir-
CE heats directly the entire return current, most of the
e; the remaining portion going through the grate bars
bustion. With OUTSIDE-CURRENT FURNACE walled air-
of the furnace, so that the return air passes freely over
ese airways assist in keeping the strata cool, besides
for the current (Fig 14, 15). DUMB-DRIFT FURNACE
rn current possibly carries explosive quantities of fire-
rnace is then supplied by a split from main intake, with
ort active combustion; generally about 23 lb (300 cu ft)
The hot furnace products pass through a flue long enough
ure cooling of the gases below ignition temp before join-
at the foot of the upcast. Main return current passes
ial airway, called the "dumb-drift." A similar arrange-
lit to feed the fire may be taken from the return current,
n installation of the furnace in the main return would
fic.

. A. In SHAFT MINES, the furnace should be close to
arge the hot air into the shaft as near bottom as possible,
ht of heated air column. Deepest available shaft should
st, for greatest security against seasonal reversals of
ace is best placed on the dip or lower side of shaft bot-
y rise to the shaft, and cost of excavation be minimized.
be used as an upcast. The water reduces temp, in fall-
nn of air, and, in evaporating moisture, part of the heat
more fuel is required. B. In DRIFT MINES the avail-
nn is usually small. Furnace is best placed at mouth
a shallow shaft, with or without a surface chimney, is
height of column. A safe plan is to build furnace on the
th return airway by an airtight brick or wooden drift.

cast current. For timbered upcasts, temp should not
ctice, it is usually less than 200°. Wood is slowly car-
nd coal takes fire at about 600°. If upcast shaft is also
should not exceed 100° F.

At a calorific power of 14 000 B t u, 1 lb of coal will
o cu ft) of air 1° F. Actual duty is from 500 000 to
er figure for dry shafts. A wet shaft increases fuel con-
ve.

e furnace, passing 120 000 cu ft of air per min. at 2-in water
air. Under lower press, at another colliery, consumption
t air.

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Grate area equals weight of coal burned divided by rate of combustion, w
depends upon the draft. For open and outside-current furnaces the draft may
the head requ
duce the velo
the furnace; v
the fire is take
intake, the dr
ference in pr
main intake
For open furn
water gage is
burn 9 lb semi
run of mine c
per hr. Tab
rate of comb
responding t
drafts in close
Ordinary fuel
tion of mine
from 7 to 14
hr per sq ft o

Table 5. Draft and Fuel Consumption of Furnaces
(Stirling Boiler Co)
Lb coal burned per hr per sq ft of grate

Draft, in water gage	Kind of fuel		
	Bituminous, run-of-mine	Semi-Bit, run-of-mine	Anthracite, pea
0.01	2.5	1.2	1.2
0.05	10.0	7.0	4.5
0.10	19.7	12.0	7.2
0.30	43.0	26.3	15.2
0.50	.....	35.7	21.5
1.00	.....	.....	32.0

Furnace resistance is due to: (*a*) added velocity of air in passin
constricted furnace area; (*b*) frictional resistance; (*c*) resistance of
furnace or flues; (*d*) added frictional resistance in upcast due to incr
ume of hot air. (*e*) Weight added to upcast column by
ucts of combustion. Effect of resistances (*a*) to (*d*) is co
shown in Art 7. Added wt (*e*) is computed from coal b

For open furnaces, the resistance is also conveniently det
an adaptation of Peclet's formula for chimneys (Rankine's "
gine"): $h = \frac{u^2}{2g}\left(1 + \frac{fl}{m}\right) - \frac{v^2}{2g}$, where h = head absorbed
and friction, ft of air column; u = aver velocity, ft per sec;
ft; m = aver "hydraulic depth" (area section ÷ perimeter), ft (

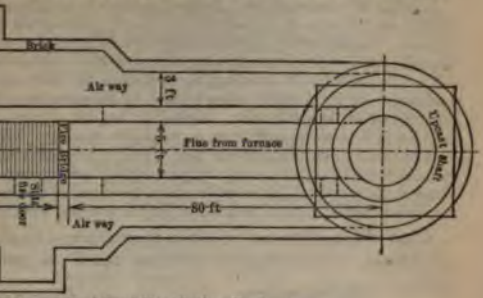
Fig 14. End-fired Ventilating Furnace

to total length of furnace, flue, upcast, and chimney, if any); f = coeff
v = velocity of approach to furnace, ft per sec; g = 32.2 ft per sec. Experime
side furnaces with 80-ft stacks, yielding 20 000 to 30 000 cu ft of air per min, s
be 0.0172 (for boiler stacks, f is usually taken as 0.012).

Design of furnace is based on press required to overcome mine an
resistance, and maximum adverse seasonal press. This press h (lb pe
the difference in wt between heated air column and cold air displaced.
temp is that which will produce h in the available height of column; t
being usually determined by local conditions. The greater the heigh
the temp, and the greater the useful effect per lb coal. For convenienc
and to avoid excessive friction and length of flame in the flue, it is reco
that the velocity of air through the furnace should not exceed 20 ft p

Mode of firing. Front or end firing is usual, in which case lengt
bars should not exceed 6, or at most 8 ft. For large furnaces with long

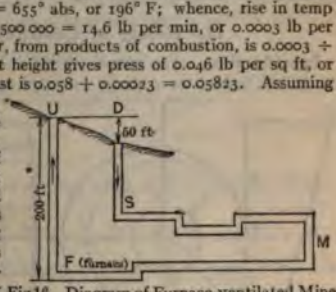
one or both sides, from which firing is done through oors on furnace and ash-pit are convenient in starting types of furnaces. With the open furnace they should fire, because they resist the air current. With side-



Side and End-fired Ventilating Furnace

for this purpose is less objectionable, but is generally -drift furnace, where only enough air to support the ne grate, doors are required.

ventilation. In Fig 16, U and D are upcast and downcast of 50 000 cu ft of air from D to U, via SMF, is 0.5 in water lled at F. The most adverse temp condition is assumed to °, mine temp 50° F, wt per cu ft of air in mine is 0.075 lb lverse press is therefore $50 \times 0.003 = 0.15$ lb per sq ft = ion, Art 7). Furnace resistance is tentatively assumed at nce, total press required is $0.50 + 0.12 + 0.03 = 0.65$ in ft. Original wt of upcast column is $200 \times 0.075 = 15$ lb ed to $15 - 3.38 = 11.62$ lb. Wt of the heated air = 0.075 cu ft, its volume being 1.29 times that of the cold mine air. $(459° + 50°) = 655°$ abs, or 196° F; whence, rise in temp $000 \times 146 \div 500\,000 = 14.6$ lb per min, or 0.0003 lb per nsity of hot air, from products of combustion, is $0.0003 \div$ through 200 ft height gives press of 0.046 lb per sq ft, or density of upcast is $0.058 + 0.00023 = 0.05823$. Assuming ss 50 000 ft of



gage for heated $5823 \div 0.075 =$ or cold air. If rnace be 500 ft s in and around ce resistance is $= 50\,000 \div 75$ elocity head is per min, veloc-

Fig 16. Diagram of Furnace-ventilated Mine

ce, 0.063 lb, or ing 4 000 sq ft ced by furnace walls, and $1 \div 10^3$ coeff of friction, the frictional d volume of hot air and increased veloc on old surfaces) is

$$\frac{0 \times \overline{666}^2}{75} = 0.236 \text{ lb per sq ft} = 0.045 \text{ in water gage.}$$

installation is therefore $0.009 + 0.06 + 0.012 + 0.045 =$ tentative assumption. To find rate of combustion, assum- oal: Current velocity of 666 ft per min produces 0.028 in

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water gage, at which draft, Table 5 gives 6 lb of coal per hr per sq
about 50% for open furnace conditions, say 10 lb coal per hr. Coal
is 14.6 lb per min = 876 lb per hr, whence grate area is 87.6 sq ft.
how this furnace may be constructed.

Details of construction. Body of furnace is usually a circular
brick from the skew backs, which are 1 to 2 ft above grate bars. Sid
to roof. Double arches, with air space between, are safer for furnaces
in which case, side air passages are also desirable. Spandrels are left
air, or closed by walls or filling, for roof support. For 7-ft span, fu
walls may be 12 in thick, when backed by air passages; otherwise,
fire protection, 2 to 3 ft thick. Grate bars are 2 to 3 ft above floor, l
1 in 12 to 1 in 6 towards front, to facilitate cleaning. Flame-bridg
should be 12 to 18 in above the bars. Flue to upcast is walled in, w
nace is built in a coal seam. Flames, sparks, and dangerous temp
beyond the flame-bridge in small plants, to 120 ft in the larger. Sin
not so extensive, in front of furnace, prevents danger from possible b

Cost of installation. General figures cannot be given, owi
conditions. Following unit costs are useful for preliminary
regions.

Rubble masonry	$8.00 per cu yd
Red brick "	0.40 per cu ft
Fire-brick "	0.60 " " "
Grate bars, in place	0.04 per lb = $2.4
Excavation, clay	0.50 per cu yd
Excavation, loose rock	1.00 " " "
Timber, in place	$25 to $50 per 1 00

An outside ventilating furnace, with 40 sq ft grate area, can be bu

Cost of operation. Coal mined near point of use costs 50
With labor @ $2.00 per day, firing should not exceed 50¢ pe
underground furnaces, fireman can often mine the coal from
With well-built furnace, repairs and depreciation are small.

Examples. In a shallow Penn bituminous mine, a furnace pass
per min cost $3.10 per day. In a deep English colliery, 2 furnaces,
per min, cost $10.44 per day.

12. Theory of Centrifugal Fans (11, 28, 29,

Introduction. Fan wheel consists of a number of vanes
to spiders keyed on a shaft, and inclosed more or less comp

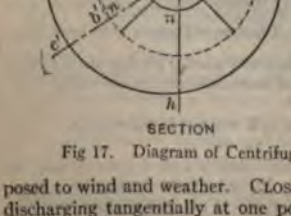

Fig 17. Diagram of Centrifugal Fan

The air ente
side casing, o
is caught up
receives a rot
being thrown
ery. Centri
veloped, whi
pressure fro
cumference.
FANS the ca
sides of the
air dischargi
atmosphere a
cumference.
fore, can op
exhauster, an

posed to wind and weather. CLOSED FANS are cased at both sid
discharging tangentially at one point only, through a flaring

hey can be operated as exhaust or pressure fans, by or chimney with the mine. Hence the difference ure fans (blowers) lies in design of the casing, the h. In the expanding chimney the discharge velocity is increased. In Fig 17 *bfghc* is the peripheral cas- e or both sides; *bcde* (and *ofp*) the chimney; *jl* and *km* *b'* the point of cutoff; *b'n* the cutoff clearance; *b'c'* rea represented by it is the ' is an adjustable shutter, ng the point of cutoff.

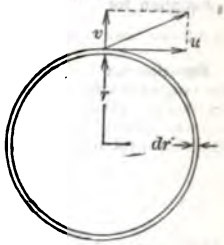

Fig 18

ing relations between veloc- work. Fig 18 represents an the fan, at a distance r from round its center at a tangen- adial velocity v. u depends d shape of vanes; v depends ed and the cylindrical area the axial width of fan at ra- of the ring, and δ the weight ss of the ring $m = 2\,\delta\pi r v\,dr$ developed by its rotation is f time dt, the radius is increased by $dr = v\,dt$; and the radial o changes dv and du. The forces producing these changes $m\,du \div dt$. Also, the radial deflective force controlling the $\frac{dv}{dt}\Big)$. Total work required is the tangential force applied l force applied through distance $v\,dt$, plus work of overcoming in time d or distance dr is $m\left(\frac{u^2}{r}v\,dt + u\,du\right) = \frac{mu}{r}d\,(ur)$.

he mass of air, being controlled by form and speed of fan, rk, a part of which appears in the form of pressure. Hence, tance dr is the total differential of both static and kinetic of u, v, and h. The real velocity of mass m is $\sqrt{u^2 + v^2}$. e mass is measured by the height h of an air column, the t of the moving air, then total energy of the mass is

$$e = \frac{m}{2}(u^2 + v^2) + gmh$$

$$de = mg\left(\frac{u\,du}{g} + \frac{v\,dv}{g} + dh\right) = \frac{mu}{r}d\,(ur)$$

$$dh = \frac{u^2}{r}dr - v\,dv$$

ies at inner end of vanes are h_1, r_1, u_1, and v_1, the integral

$$h_1 = \frac{1}{g}\int_{r_1}^{r}\frac{u^2}{r}dr - \frac{1}{g}\int_{v_1}^{v} v\,dv$$

press between inner and outer end of vanes, when r is ocities of the air.

nable from the vortex limited by r and r_1 is measured r and a press exceeding actual press at radius r by a head velocities at that radius, In practice this restitution is ortex surrounding the fan wheel and movement in the ex- restitution is never realized. If H be maximum possible city head at r − velocity head at r_1. As the velocity + 2 g and $(v_1^2 + u_1^2) \div 2\,g$, H becomes

$$\frac{v^2}{r}dr\Big) + \frac{u^2 - u_1^2}{2g} = \frac{1}{g}\int_{r_1}^{r}\left(\frac{u^2}{r}dr + u\,du\right)$$

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spokes. Useful hp, 75.7; absorbed by fric of air, 6.2 hp; mech fr
engine, 113.4 (see Example, Art 13). Press on 1 vane (Art 1
$8\ wg = (2 \times 2\pi \times 2.64 \times 2\,000 \times 0.075) \div (4.4 \times 32.2) \div 8 =$
Plate required to resist this press is too light to be durable, hence

Fig 23. Frame for Wooden Casing

of vane =
wt of plat
Total forc
× area, c
(number v
sultant).
(7.4 + 4.4
air = 75.
force = 1
+ (8 × 2
Each van
2 angles,
0.375 in;
2.11 sq in
Length of
7.4 − 2 =
= 2 × 5.4
vane = 10
= 178 + 3

$f' = 0.5 \times m \times (2\pi R)^2 \times (r + r_1) = 0.5 \times 5.5 \times (2 \times 3.142 \times 2.64)^2$
Tension = 8 900 ÷ (2 × 2.11) = 2 100 lb per sq in. Safe working
per sq in; but, to allow for vibration, bolt holes, and excess speed,

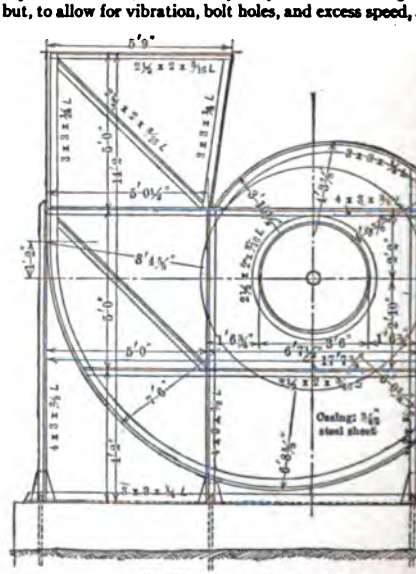

Fig 24. Frame for Steel Casing

as extreme fiber stress. Hence, 4 000 − 2 100 = 1 900 lb is ava
moment. Max permissible bending moment = 1 900 × (2 × 0.8
bending moment = 58 × (5.9 − 2) × 12 = 2 700 in-lb. Hence, ang
of shaft = 113.4 × 550 × 12 ÷ 2 × 3.142 × 2.64 = 45 000 in-lb

1 000 lb. If unsupported length of shaft is 10 ft, the combined 00 in-lb. Size must also be sufficient to prevent excessive de- about 6.25 in diam.

ons. Wooden casings are cheapest, steel or brick more or concrete pits for lower part of wheel enables engine t necessitates drainage. In gassy mines, fan should be loors placed so as to Direct-connected to belts or gearing, to breakdown. In 2 engines, one in are sometimes in- be controlled by n of the air current oad on fan. Cur- versible by placing conducted to inlet or as desired; Fig 25 nt. Coupled fans, air traverses both half the air, are

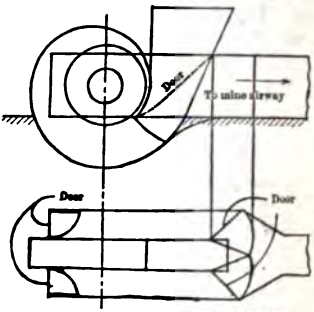

Fig 25. Setting for Reversible Fan

with local conditions, as well as with size, style, and ng figures are approximate. A small wooden-cased fan, h 8-ft diam wheel, can usually be installed complete with d low-speed engine, for $1 500 to $2 000. Same plant, 2 500 to $3 500. An 8-ft steel fan, built for moderately ev per min), will cost installed about $3 000, including e plant, with 16-ft fan, about $7 000. Total cost of 20 e from $7 500 to $10 000.

(J. W. Montgomery, Jeffrey Mfg Co) cost in 1915:

actory	$2 500
connections to mine	1 800
	1 600
	600
allation of engine, and connection)	500
	$7 000

lower Co, 1915) gives cost of fan wheel, shaft, bearings, 4¢ per cu ft of air passed per min for small sizes (to say 0.9¢ for large fans, of say 200 000 cu ft per min.

fans. Main items are fuel and labor. No figures of ven. In isolated steam plants, where fan is supplied by can attend both fan engine and boilers, up to capacity an is electric-driven, or located so that its steam supply plant, only occasional attention for oiling and inspection umption is 5 or 6 lb of coal per ihp per hr. The cost of 1.5 to 2.5¢ per kw-hr.

les of Types of Fans (27, 28, 29, 30, 32)

), the most widely used of all mine fans, is of the closed ding chimney. In its design are exhibited many features gal fan (Art 12 to 14). It has 6 to 10 vanes, plane side oat and point of cutoff, generally rectilinear backward-

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sloping vanes, and single or double inlet (usually single). I
of 35 ft or more. Peripheral casing varies greatly in design
into 2 classes: (*a*) spirals of great length (as in Fig 24), l
at or near point of cutoff; (*b*) short spirals (as in Fig 21)
above to 135° below point of cutoff. Many Guibal fan
100 000 to 300 000 cu ft air per min. Fig 22 shows a spe
fan. Thirty tests on a number of fans show peripheral spee
4 000 to 5 500 ft per min; water gage, from 1 to 3 in, a

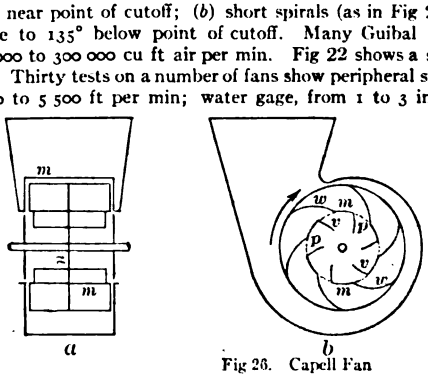

Fig 26. Capell Fan

Capell fan, also of closed type, is quite largely used, an
As originally built (Fig 26, *a* and *b*) the fan body is divide
a steel cylinder *m*, ex
side on the inlet circle
shaft by spiders. It
into 2 parts by a c
perpendicular to the
are 6 rectangular po
area of which is at le
area. There are 2 sets
inside of cylinder *m*, a
and 6 outside, mark
ports; all sloping ba
vanes are riveted to t

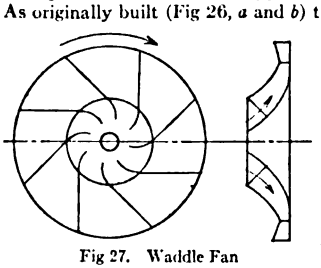
Fig 27. Waddle Fan

In the later form (Fig 26, *c*), cylinder *m* is omitted. Th
plane surfaces, with a slight backward slope. Their sides or
ward obliquely to plane of rotation, thus forming "scoops" (not shown in the cut), which assist in drawing the air into the fan. Outer vanes *y* begin at the ends of inner vanes, and are concave to direction of rotation, with nearly radial tips. It is believed that better manometric effic is obtained from concave vanes, but that formation of eddy currents behind the tips reduces the mechan effic. By means of curved plates *z*, on backs of main vanes, eddies are minimized, and the advan
retained. Peripheral speeds are usually higher than for th

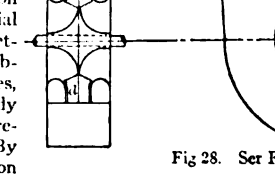
Fig 28. Ser

pen-running, and takes air on one side only. Vanes are whole structure rotating. There are 8 or 10 vanes, which

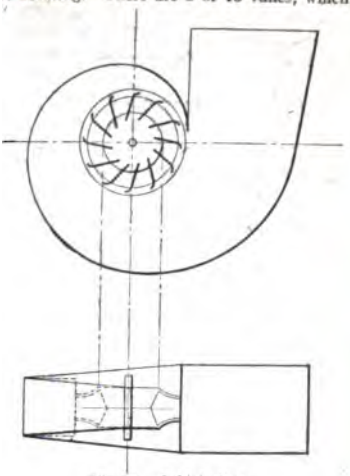

Fig 29. Schiele Fan

with vertex towards the incoming air. Casing is a large, phery than the tips of vanes. Tests show high mechanical

16. Propeller Fans

(Fig 30) has oblique vanes (either plane or warped marine propeller. Usually no attempt is made to by use of a chimney, as in centrifugal fans. Though his type of fan is not as efficient a pressure producer as the centrifugal, hence it is chiefly useful for low mine resistance. No complete analysis of its operation has been made, but approx press equations are obtainable either from the principle of deflecting surfaces or from that of the helicoidal screw.

k Fan

Theory of deflecting surfaces. In Fig 31, *a*, AA' is an oblique vane moving with pure translation at velocity u in direction DA, at right angles to air current, the velocity and direction of which are indicated by v. AB, the direction and velocity of the air with reference to vane, just before deflection, is found by combining v with a velocity equal and opposite to u. Direction after deflection (with reference to the vane) is the direction s such as will give a component normal to u, equal to v, or ddies or liquid friction are produced, the energy of the air he same at entry as it is on leaving, and $h_1 + AB^2 \div 2g =$ d h_2 are total heads before and after passing the vanes. ig 31, *b*), no eddies are produced, and $h = h_1 - h_2 = (u^2 + + v^2 + 2g$; where h_0 is the apparent press, corrected for

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velocity so as to indicate press between quiescent air at entry and
When terminal angle of the vane is 90°, $\sin \alpha = 1$, and $h_0 = (u^2 -$
on leaving the vanes entered a
at the proper angle, with redu
negligible quantity, the termin
would be restored to head and
$= u^2 \div g$ Value of u varies
most of the work is done near
air on leaving vanes has varie
according to the distance fro
of vanes, and further analys
If u = tangential velocity of t

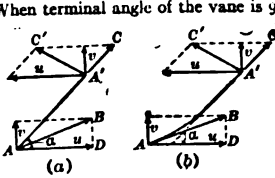

Fig 31. Diagram of Deflecting Surfaces

ft per sec (as in centrifugal fan), and k = manometric yield, the eff
air column, is $k = ku^2 \div g$; where k varies from 0.05 to 0.23, averag

Helicoidal screw theory. Assume that a fan with true helicoid
velocity on the air equal to the pitch multiplied by number of rev, an
duced is equal to that which is due to the difference between this ve
velocity of the air through the fan. Then, if P = pitch of vanes, ft, R
v = velocity of the air, ft per sec, the head produced is $H' = (PR$
head is absorbed by production of velocity v, by fan resistance h', an
Applying coefficient of reduction K (as for centrifugal fans):

$$H = \frac{K(PR - v)^2}{2g} = \frac{v^2}{2g} + h' + h_0,$$

where h_0 and h' are in ft of air column, and K varies approx from 0.5

Angle and number of vanes. Angle α between vanes an
is usually from 30° to 50°, and pitch $P = 1.8\,D$ to $3.8\,D$. B
upon relation between velocity of air through the fan and
Number of vanes appears to have little effect; from 5 to 12 v

Speed of rotation. Fans of 4 to 12 ft diam usually have
of 4 200 to 8 000 ft per min.

Size of fan. Sizes recommended by S. B. Stine, Osceola M
ent volumes, are given in Table 8.

Table 8. Propeller Fans

Diam, ft	Volume, cu ft per min	Diam, ft	Volur
4	20 000 to 30 000	8	75
5	30 000 to 45 000	10	95
6	45 000 to 60 000	12	100
7	55 000 to 75 000		

Efficiency. When useful work is computed by multiplyin
by water gage by the volume delivered, the mechanical effic i
20 and 40%. When the work of producing the velocity is i
work, effic may reach 70%.

Cost of propeller fans, including steam engine, is about $
and $1 200 for a 12-ft fan. With electric motor instead of s
of same size fans is about $350 and $1 400, respectively.

17. Volumetric Ventilators

Types. They are displacement machines, which isolate and remov
air at each revolution or cycle. There are 2 types: reciprocating,
Struve; rotating, including the Fabry, Lemielle, Root, Baker, and
chines are applicable for small volumes of air at relatively high pres
mine resistance is great, due to great length or small cross-section o

ontal, double-acting air pump. There are 2 square cylinders
wheels running on rails. Lower half of each cylinder head
contains inlet valves, the upper half, discharge valves,
thus resembling an air compressor. The machine is
driven from an engine by cranks and connecting rods.

FABRY ventilator (Fig 32) is an old machine (see Callon's "Lectures on Mining," Vol 2), not used in the U S. It has 2 wheels, on parallel shafts, revolving in opposite directions, and inclosed in a casing opening both above and below. Each wheel has 3 radial arms *a*, which fit closely to the casing and are of such length that in rotating they just clear the axis of the companion wheel. Mounted on arms *a* are cross blades *b*, on each end of which are short arc-shaped tips *c*, which make a close rolling joint with each other as the wheels rotate. A volume of air is thus inclosed, between the wheels and the outer

ator

olution. The Fabry machine is the prototype of the modern ROOT ich can be used as either exhaust or pressure ventilators (31).

, the volume delivered is independent of the press; but, as ckward through the joints of the machine also increases. Volu- by dividing the actual delivery by the theoretical delivery, or y between 50 and 90%. Let W = displacement, V = volume e lost by leakage. Then, if a be the equivalent orifice of the esenting the aggregate area of leaks,

$$+v, \text{ and } V+v=a+o; \text{ whence, } V=\frac{Wa}{a+o}$$

s of volumetric machines gave the following efficiencies: Lem-; Nixon, 45.9%; Struve, 57.8%. The reciprocating machines ing, 35%.

Emergency Ventilators

sionally used for temporary service; or in ntaining ventilation and preventing reversal eak-down, or repairing of fan. A group of one steam pipe, is placed either at top or en at mouth of shaft, the jets produce an t. Placed in bottom of shaft, the steam st column of air, and also acts mechanically, The Körting ejector has been used for these g jet, made by L. Schulte & Co, Phila, gives 9, with 45-lb steam press, working against er gage.

et is low; probably not over 15%, on basis reover, unlike furnace ventilation, the effi- depth of shaft, because of rapid condensa- and cools (26).

ts may also be used in emergen- is even lower than that of steam. pulsive force, and the cold due s temp of upcast column.

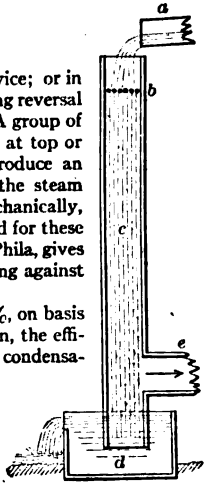

Fig 33. Hydraulic Box Ventilator

tor, or "trompe" (Fig 33), produces nical action of falling water. It is ating a small volume of air in a blind heading or stope. Water ron grid *b*, and down wooden pipe *c*, to cistern *d*. Air is entrained at *b*, and is carried down, escaping at *e*. With 10-ft fall, in a box red cu ft of air per min can be circulated against a small counter from two 1-in holes has produced about 8 000 cu ft of air per

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Table 9. Schulte Ventilating Jets

Size No	Diam steam pipe, in	Diam air passage, in		Cu ft of air per min	Size No	Diam steam pipe, in	D pa
		Inlet	Discharge				Inl
00	⅜	4	3	100	4	1	17
0	0.5	5	4	200	6	1.25	24
1	0.5	8	6	500	8	1.5	32
2	0.75	11	8	1 000	10	2	42
3	1	14	10	2 100			

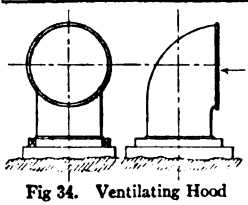

Fig 34. Ventilating Hood

Ventilating hoods (Fig 34) ventilate metal mines; also the as auxiliary means of increasing during periods of barometric dep winds are most likely to prevai hood will be placed at downcast turned to face wind. It is estim ing a velocity 10 meters per se ventilating current down air sh head of 16.5 mm (0.65 in) of wat

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SECTION 15

RESSED-AIR PLANT

BY

RICHARD T. DANA

CONSULTING ENGINEER

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theses refer to Bibliography at end of this section.

antages of compressed-air transmission of power for under- red with steam: (*a*) relatively small loss in transmission uced into the mine; (*c*) exhaust air aids ventilation; (*d*) ust steam underground is avoided; (*e*) power is more con- ent work, because there are no radiation nor condensation ined whether in use or not; (*f*) field of application is wider. ower, as in the compressed-air locomotive, and is useful for s after blasting. Compared with electricity the advantages narked. Each has its field, and both modes of power trans- mining. Chief points in favor of compressed air are: (*a*) dusty collieries; (*b*) superiority of pneumatic over electric- aids somewhat in ventilation.

1. Air Compression

ory of air compression is given in Sec 39.)

s. WET COMPRESSORS (practically obsolete in the air is cooled during compression by introducing wate-

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into the cylinders. There are 2 types: (*a*) hydraulic-plu bulk and also in form of spray; (*b*) those using spray or j wise similar to dry compressors. Though the air is thorc designed, slow-stroke wet compressors, injected water is mc able. DRY COMPRESSORS, in which the air is cooled by superseded the wet type. Compressors may be: STEAM-I air and steam cylinders are direct-connected; POWER-DRI connected, or gearing, or chains; WATER-DRIVEN, as by im

The compressors cited herein are not necessarily to be consider taken to illustrate standard types; no preference for any partic Table 1 gives close approximations to the actual hp, in 1, 2, or compared with the hp for isothermal and adiabatic compression

Application of different types. For 80 to 100 lb pr compressors are efficient. At 100 lb press, a 2-stage mach in power; at 80 lb, it may save enough to justify its use u For less than 80 lb press, a single-stage machine is gene 100 to 500 lb, a 2-stage compressor is required; from 500 over 1 000 lb, 4-stage. In figuring on the requirements c to specify at least 25% greater capacity than is absolutely for future expansion and prevent decreased efficiency due running. Simple straight-line compressors are furnished capacities up to 1 400 cu ft free air per min, duplex compres used for larger capacity. When power is furnished by ele belt drive, the duplex form is suitable for capacities above

Table 1. Horse-power Required for Compressing 1 Cu I at Sea Level to Different Gage Pressures (Ingers

(Initial temp of air taken as 60° F. Jacket-cooling no

Single-stage compression						
Gage press, lb	Abs press, lb	No of atmos	Isothermal		Ac	
			Mean effective press, lb	Hp	Mean effective press (theoret)	Mean e fect pre plus 15 frictio
20	34.7	2.36	12.62	0.055	14.30	16.44
25	39.7	2.70	14.68	0.064	16.94	19.47
30	44.7	3.04	16.30	0.071	19.32	22.21
35	49.7	3.38	17.90	0.078	21.50	24.72
40	54.7	3.72	19.28	0.084	23.53	27.05
45	59.7	4.06	20.65	0.090	25.40	29.21
50	64.7	4.40	21.80	0.095	27.23	31.31
55	69.7	4.74	22.95	0.100	28.90	33.23
60	74.7	5.08	23.90	0.104	30.53	35.10
65	79.7	5.42	24.80	0.108	32.10	36.91
70	84.7	5.76	25.70	0.112	33.57	38.59
75	89.7	6.10	26.62	0.116	35.00	40.25
80	94.7	6.44	27.52	0.120	36.36	41.80
85	99.7	6.78	28.21	0.123	37.63	43.27
90	104.7	7.12	28.93	0.126	38.89	44.71
95	109.7	7.46	29.60	0.129	40.11	46.12
100	114.7	7.80	30.30	0.132	41.28	47.46
110	124.7	8.48	31.42	0.137	43.56	50.09
120	134.7	9.16	32.60	0.142	45.69	52.53

Table 1.—(*Continued*)

Two-stage compression

	Inter-cooler press	Isothermal		Adiabatic				Saving over 1-stage comp. %
		Mean effec press	Hp	Mean effec press, theor	Mean effec press, + 15% fric	Hp theor	Hp plus 15% fric	
	20.6	25.70	0.112	29.31	33.71	0.128	0.147	12.3
	22.7	27.52	0.120	31.44	36.15	0.137	0.158	13.8
	24.5	28.93	0.126	33.37	38.36	0.145	0.167	14.2
	26.3	30.30	0.132	35.20	40.48	0.153	0.176	15.0
	28.1	31.42	0.137	36.82	42.34	0.161	0.185	15.2
	29.8	32.60	0.142	38.44	44.20	0.168	0.193	15.6
	31.5	33.75	0.147	39.86	45.83	0.174	0.200	16.3
	32.9	34.67	0.151	41.28	47.47	0.180	0.207	16.7
	34.5	35.59	0.155	42.60	48.99	0.186	0.214	16.9
	36.1	36.30	0.158	43.82	50.39	0.191	0.219	18.4
	37.3	37.20	0.162	44.93	51.66	0.196	0.225	19.0
	38.8	38 10	0.166	46.05	52.95	0.201	0.231	19.3
	40.1	38.80	0.169	47.16	54.22	0.206	0.236	19.5
	41.4	39.50	0.172	48.18	55.39	0.210	0.241	20.1
	47.6	42.70	0.186	52.84	60.76	0.230	0.264	.....
	53.4	45.30	0.197	56.70	65.20	0.247	0.283	.....
	58.5	47.30	0.206	60.15	69.16	0.262	0.301	.....
	63.3	49.20	0.214	63.19	72.65	0.276	0.317	.....
	67.8	51.20	0.223	65.93	75.81	0.287	0.329	.....
	72.1	52.70	0.229	68.46	78.72	0.298	0.342	.....

Three-stage compression

Intercooler press		Isothermal		Adiabatic				Saving over 2-stage comp. %
Stage 1	Stage 2	Mean effec press	Hp	Mean effec press, theor	Mean effec press +15% fric	Hp theor	Hp plus 15% fric	
14.4	42.9	30.30	0.132	33.30	38.30	0.145	0.167	5.23
18.2	59.0	35.59	0.155	40.30	46.50	0.175	0.202	5.92
21.2	73.0	39.50	0.172	45.20	52.00	0.196	0.226	6.67
23.8	86.1	42.70	0.186	49.20	56.60	0.214	0.246	6.96
26.1	98.7	45.30	0.197	52.70	60.70	0.229	0.264	7.28
28.2	110.5	47.30	0.206	55.45	63.80	0.242	0.277	7.64
30.0	121.0	49.20	0.214	58.25	66.90	0.253	0.292	8.33
31.8	132.3	51.20	0.223	60.40	69.40	0.263	0.302	8.36
33.4	142.4	52.70	0.229	62.30	71.70	0.273	0.314	8.38
35.0	153.1	53.75	0.234	65.00	74.75	0.283	0.326	8.80
36.3	162.3	54.85	0.239	66.85	76.90	0.291	0.334	8.86
37.6	171.5	56.00	0.244	67.90	78.15	0.296	0.340	9.02
38.9	180.8	57.15	0.249	69.40	79.85	0.303	0.348	9.18
40.1	189.8	58.10	0.253	70.75	81.40	0.309	0.355	....
41.4	199.5	59.00	0.257	72.45	83.25	0.315	0.362	....
42.5	207.8	60.20	0.262	73.75	84.90	0.321	0.369	....
43.4	214.6	60.80	0.265	74.80	86.00	0.326	0.375	....
44.6	224.5	61.72	0.269	76.10	87.50	0.331	0.381	....
45.7	233.3	62.40	0.272	77.20	88.80	0.336	0.383	....

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Table 2. Piston Displacements, Double-acting Cyl, C
(Chicago Pneumatic Tool Co)

Diam, in	Stroke, in							
	12	14	16	18	20	22	24	30
8	0.698	0.814	0.930	1.04	1.16	1.27	1.39	1.74
9	0.883	1.030	1.177	1.32	1.47	1.61	1.76	2.20
10	1.090	1.272	1.454	1.63	1.81	2.00	2.18	2.72
11	1.319	1.539	1.759	1.97	2.19	2.41	2.63	3.29
12	1.571	1.832	2.094	2.35	2.61	2.87	3.14	3.92
13	1.843	2.150	2.457	2.76	3.07	3.37	3.68	4.60
14	2.137	2.494	2.850	3.20	3.56	3.91	4.27	5.34
15	2.454	2.863	3.272	3.68	4.09	4.49	4.90	6.13
16	2.792	3.257	3.723	4.18	4.65	5.12	5.58	6.98
17	3.152	3.677	4.203	4.72	5.25	5.77	6.30	7.88
18	3.534	4.123	4.712	5.30	5.89	6.47	7.06	8.83
19	3.937	4.593	5.250	5.90	6.56	7.21	7.87	9.84
20	4.363	5.090	5.817	6.54	7.27	7.99	8.72	10.90
21	....	5.612	6.413	7.21	8.01	8.81	9.62	12.02
22	....	6.159	7.037	7.91	8.79	9.67	10.55	13.19
23	....	....	7.693	8.65	9.61	10.57	11.54	14.42
24	....	....	8.377	9.42	10.47	11.52	12.56	15.70
25	....	....	9.090	10.22	11.36	12.49	13.63	17.04
26	....	....	9.830	11.06	12.29	13.51	14.74	18.43
27	....	....	10.602	11.92	13.25	14.57	15.90	19.88
28	....	....	11.402	12.82	14.25	15.67	17.10	21.38
29	....	....	....	13.75	15.28	16.81	18.34	22.93
30	....	....	....	14.72	16.36	17.99	19.63	24.54
31	....	....	....	15.72	17.47	19.21	20.96	26.20
32	....	....	....	16.75	18.61	20 47	22.34	27.92
33	....	....	....	....	....	21 77	23.75	29.69
34	....	....	....	....	....	23.11	25.22	31.52
35	....	....	....	....	....	....	26.72	33.40
36	....	....	....	....	....	....	28.27	35.34
37	....	....	....	....	....	....	....	37.33
38	....	....	....	....	....	....	....	39.37
39	....	....	....	....	....	....	....	41 47
40	....	....	....	....	....	....	....	43.63

In calculating capacity of stage compressors, the dimensi
cyl are used, Capacity in free air (air at atmos press) per mi
of cyl displacement per rev by number of rev per min, l
For effect of leakage of valves, see Art 10.

Piston clearance. Towards end of stroke in a dry comp
press exceeds that in the discharge pipe, the compressed ai
begins to pass through the delivery valves. Due to clearar
tity of hot compressed air remains in the cyl at end of stroke
this clearance air expands and partly fills the cyl behind th
can enter through inlet valves until cyl press falls below at
fore, since it is never possible to take a full cyl of fresh a
must be made a minimum, say about 1/16 in, or ranging fr
entire cyl volume on short-stroke, to less than 1% on long
(2, 3, 4). In wet compressors, the clearance spaces are filled
not produce this effect.

Losses due to clearance, piston and piston-rod displace
leaky valves, reduce the volumetric capacity to about 70 to
displacement, according to type and physical condition of c

tage Steam-driven Compressors (3)

sors are not suitable for high pressures, because: the ld not be efficiently dealt with by the water jackets of e produces severe stresses; they are more difficult to oling is by water-jacketing, sometimes with ante- . They are classified, according to arrangement of ign, into: STRAIGHT-LINE and DUPLEX.

have steam and air cyl set tandem. They have a flywheel iven by outside connecting-rods from a cross-head between mple or compound, the latter giving better steam economy. ned, requires small space and inexpensive foundations, and lling an important field, the steam economy is lower than in are not self-regulating at much less than 40% of full load. acities up to 1 700 or 1 800 cu ft of free air per min.

furnish examples of standard makes, types, and sizes of of engineers who may not have catalogues at hand.

n-driven Compressors, Class N-SS (Chic Pneum Tool Co)

Maximum capac			Minimum capac			Max air press	Floor space	
Displ, cu ft per min	R p m	Hp	Displ, cu ft per min	R p m	Hp		Length ft in	Width ft in
51	260	11	43	220	9	100	6 5	2 2
80	260	13	67	220	11	70	6 5	2 2
116	250	24	100	215	21	100	9 2	3 1
182	250	30	156	215	25	70	9 2	3 1
262	250	31	225	215	26	40	9 2	3 1
218	240	45	191	210	40	100	10 5	3 7
314	240	58	275	210	50	80	10 5	3 7
427	240	65	374	210	57	60	10 5	3 7
559	240	66	489	210	58	40	10 5	3 7
361	230	75	322	205	67	100	11 10½	3 9¼
492	230	90	438	205	80	80	11 10½	3 9¼
642	230	99	572	205	87	60	11 10½	3 9¼
813	230	96	725	205	86	40	11 10½	3 9¼
549	220	114	499	200	103	100	13 6	4 7
716	220	131	651	200	118	80	13 6	4 7
907	220	139	825	200	126	60	13 6	4 7
1 120	220	132	1 018	200	120	40	13 6	4 7

ave plain slide valves; 14-in stroke, Meyer adjustable cutoff.

t-line Compressor, Compound Steam, Simple Air End

L-p steam cyl, in	Steam pipe, in	Rev per min	Displacement, cu ft	Proper boiler capac	
				Cond'g	Non-cond'g
22	3	125	909	85	110
22	3	125	1 100	100	130
26	4	110	1 380	120	160
26	4	110	1 622	145	190
30	4	90	1 924	170	225
30	4	90	2 209	195	260
34	5	90	2 513	225	295
34	5	90	2 837	250	330
36	5	80	3 393	300	395
38	6	80	3 780	330	440
40	6	80	3 780	330	440

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Duplex compressors usually have 2 pairs of tandem steam with a common flywheel shaft. For same capacity they occu than the straight-line, are better adapted to varying load, and h of speed. In general they are more economical, and with comp cylinders may be run on say 20% less steam than simple com

Table 5. Ingersoll-Rand Duplex Steam-driven Imperial XPV
Air press, 15 to 100 lb; steam press, 80 to 120 lb

Cylinders, in			Rev per min	Piston displacement, cu ft per min	Air press, lb	Indicated hp, steam cyl
Diam, steam	Diam, air	Stroke				
10	11	14	200	608	80–100	86– 97
10	12	14	200	725	60– 75	91–105
10	13	14	200	952	45– 55	108–123
10	14	14	200	990	35– 40	103–113
10	16	14	200	1 295	25– 30	113–128
10	18	14	200	1 640	15– 20	105–131
12	13	16	180	876	80–100	125–141
12	14	16	180	1 016	65– 75	137–149
12	15	16	180	1 166	45– 60	131–157
12	17	16	180	1 504	30– 40	139–170
12	19	16	180	1 880	20– 25	141–168
12	21	16	180	2 300	15– 20	146–181
14	15	16	180	1 164	80–100	164–186
14	16	16	180	1 326	55– 75	159–193
14	18	16	180	1 686	35– 50	164–208
14	21	16	180	2 296	25– 30	198–225
14	24	16	180	3 002	15– 20	188–234
15	15	20	160	1 294	75–100	181–213
15	17	20	160	1 666	50– 70	194–238
15	19	20	160	2 086	35– 45	211–249
15	22	20	160	2 300	25– 30	241–274
15	25	20	160	3 620	15– 20	227–281

Compressors in Table 5 (typical duplex design) are either single or or compound steam cyl proportioned for elevations up to 10 000 ft the same in all. Piston valves, adapted for either saturated or supe used on steam end; Corliss inlet and direct-lift poppet discharge valv

3. Two-stage Steam-driven Compressor

Two-stage compressors. In these, compression is in 2 step (*a*) by dividing work between 2 or more cyl, there is a more unifo stresses in the machine; (*b*) less heat is generated in each cyl tha press is produced in 1 cyl, and cooling effect of the larger combine of water jackets is greater; (*c*) added cooling is obtained from a 12) between the cyl. Thus, at sea level, and compressing adia gage, a 2-stage compressor delivering 1 256 cu ft free air per m 164 hp, as compared with 184 hp for single-stage (Table 1); sa

Saving of fuel by stage compression is partly offset by greater cost and repairs of the more complicated machines. The point at which economy balances the added capital and maintenance expense corresp per sq in delivery press. Because of the friction losses involved in usi compression in a poorly designed plant may cost more than compressio

Two-stage steam-driven compressors may be straight-line or dup compound steam ends. They are the most important class for mini ver single-stage machines for large plants and high press.

and cooling, the cylinders should be proportioned to divide the n them; that is, there is a "best intercooler pressure" (Sec 39).

ompression: (*a*) Lower aver temp, lower aver press and greater eased safety and ease of lubrication. When high final temp, part increasing wear on piston and cylinder. Sometimes the combi- reaches proportions of an explosive mixture, and, if temp passes s likely. These accidents are rare, even in single-stage work; in ith proper intercooling, they should be impossible. (*c*) Greater al press in the low-press cyl is much lower than in single-stage en expanded to atmos press, occupies less space. Hence, the in- s begins earlier in the stroke. (*d*) Air delivered by a stage com- single cylinder. At constant press the power of air to hold water mp, and in passing through intercooler much of original moisture Hence, there is less trouble from condensation in delivery pipe.

traight-line Compressor, Class WB-2; Simple Steam, Two-stage Air Cyl

oke	Piston displacement, cu ft free air		Rev per min	Steam press to produce 90 lb air press	I h p in steam cyl, at 90 lb air press	Over-all dimensions		
	Per rev	Per min				L'gth, ft in	Width, ft in	H'ght, ft in
4	1.81	290	160	60	50	14 1	4 4	4 0
4	2.48	396	160	80	68	14 6	4 6	4 0
6	3.72	558	150	80	95	16 2	5 0	5 5
6	4.70	706	150	80	120	16 2	5 2	6 0
0	7.27	909	125	80	146	18 11	5 10	6 6
4	10 54	1 160	110	80	190	21 9	6 5	7 4
4	12.54	1 380	110	70	222	21 9	6 6	7 0
0	18.43	1 843	100	70	296	25 6	7 1	7 8
0	24.50	2 450	100	70	395	26 0	7 1	8 3

ficiencies of Sullivan Class WB-2 Compressors
Air Press, 100 lb. Tests by Sullivan Mach'y Co)

	290	396	558	706	909	1 380	1 840	2 450
	246	340	492	621	820	1 250	1 675	2 240
	47	65	92	115	148	224	300	398
	52	71	101	127	163	245	326	430
	85	86	88.2	88.3	90	90.5	91	91.5
	90	90.5	91	91.3	91	91.5	92	92.5

WB-2 compressors, at 100 lb steam press, non-condensing.
p-hr at ¾ cutoff 16-in stroke, 32 lb per i h p-hr at ⅜ cutoff
" " ⅜ " exceeding } 37 " " " " " " ¾ "
" " ¾ " 16-in stroke } 31 " " " " " " ⅜ "

s of Sullivan WC Compressors, at 100 lb Air Press
ions, see Table 9. Tests of Sullivan Mach'y Co)

900	2 000	2 450	3 150	Vol effic, %....	90	90.5	91	91.5	92
360	1 820	2 240	2 900	Mech " " ...	91	91.5	92	92.5	93
244	326	398	514	Steam used, lb per i h p-hr	18.5 / 23.5	17.5 / 22.5	16.5 / 21.5	15.5 / 20.5	14.5* / 19.5†
267	355	430	552	Aver cost installation	$400	$550	$750	$850	$1 000

acuum, 26 in. † Steam press, 125 lb non-cond.
at other pressures, subtract 0.5 lb steam for each 25 lb in
1 5% to steam consumption; for ½ speed, 15%; ¼ sr

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Table 9. Sullivan Corliss Class WC Compressors, Tandem, Steam, Two-stage Air Cyl

*Displ, cu ft free air per min	Steam press, lb	Condensing or non-cond	Cyl diam, in				Stroke, in	I h p in st'm cyl for 100 lb air press	Rev per min	Boile hp
			Steam		Air					
			H-p	L-p	H-p	L-p				
909	100	Cond	12	24	12·5	20	20	160	125	10
	125	Cond	12	22	12.5	20	20	160	125	9
		Non-c	14	22	12.5	20	20	160	125	12
	150	Cond	12	22	12.5	20	20	160	125	9
		Non-c	12	22	12.5	20	20	160	125	11
1 330	100	Cond	16	28	14·5	24	24	240	110	14
	125	Cond	16	28	14.5	24	24	240	110	14
		Non-c	16	26	14.5	24	24	240	110	17
	150	Cond	14	28	14.5	24	24	240	110	13
		Non-c	14	26	14.5	24	24	240	110	16
1 843	100	Cond	18	30	16.5	26	30	320	100	19
	125	Cond	18	30	16.5	26	30	320	100	18
		Non-c	18	28	16.5	26	30	320	100	23
	150	Cond	16	30	16.5	26	30	320	100	17
		Non-c	16	28	16.5	26	30	320	100	22
2 450	100	Cond	20	34	18	30	30	425	100	24
	125	Cond	20	34	18	30	30	425	100	24
		Non-c	20	32	18	30	30	425	100	30
	150	Cond	18	34	18	30	30	425	100	23
		Non-c	18	32	18	30	30	425	100	29

* No allowance for necessary losses, due to space occupied by piston ro

Table 10. Laidlaw-Dunn-Gordon Cross-compound, 2-stage Corli

(Steam press, 125 lb; air press, 90–100 lb)

Cyl diam, in		Stroke, in	Rev per min	Displacement, cu ft per min	Floor space, ft and in		Cyl diam, in		Stroke, in	Rev per min	Displacement
Steam	Air				Length	Width	Steam	Air			
Non-condensing							Condensing				
13 & 22	20 & 12	24	125	1 090	23 0	11 0	12 & 24	20 & 12	24	125	1
14 " 24	22 " 14	24	125	1 320	23 0	11 6	13 " 25	22 " 14	24	125	1
15 " 25	24 " 15	24	125	1 570	23 0	12 0	14 " 28	24 " 15	24	125	1
17 " 28	26 " 16	24	125	1 850	23 6	12 6	15 " 30	26 " 16	24	125	1
18 " 30	28 " 17	24	125	2 140	23 9	13 0	16 " 32	28 " 17	24	125	2
14 " 24	22 " 14	30	115	1 520	26 6	11 6	13 " 25	22 " 14	30	115	1
15 " 25	24 " 15	30	115	1 810	26 6	12 0	14 " 28	24 " 15	30	115	1
17 " 28	26 " 16	30	115	2 120	26 6	12 6	15 " 30	26 " 16	30	115	2
18 " 30	28 " 17	30	115	2 460	26 6	13 0	16 " 32	28 " 17	30	115	2
20 " 32	30 " 18	30	115	2 825	26 6	13 3	17 " 34	30 " 18	30	115	2
21 " 34	32 " 20	30	115	3 220	26 6	13 6	18 " 36	32 " 20	30	115	3
18 " 30	28 " 17	36	105	2 700	28 9	13 3	16 " 32	28 " 17	36	105	2
20 " 32	30 " 18	36	105	3 100	28 9	13 3	17 " 34	30 " 18	36	105	3
21 " 34	32 " 20	36	105	3 525	30 9	15 0	18 " 36	32 " 20	36	105	3
23 " 39	35 " 22	36	105	4 450	32 0	16 0	20 " 42	36 " 22	36	105	4
21 " 34	32 " 20	42	95	3 720	35 0	15 0	18 " 36	32 " 20	42	95	3
23 " 39	36 " 22	42	95	4 700	35 0	16 0	20 " 42	36 " 22	42	95	4
24 " 41	38 " 23	42	95	5 240	35 0	17 0	22 " 44	38 " 23	42	95	5
23 " 39	35 " 22	48	90	5 100	38 0	16 0	20 " 42	36 " 22	48	90	5
24 " 41	38 " 23	48	90	5 675	38 0	17 0	22 " 44	38 " 23	48	90	5

Note. — Cond'ns of service, as altitude, steam and air press, vary greatly; ber of steam and air cyl comb'ns. Tables give gen outline of sizes meeting

ree- and Four-stage Compressors

essors are used chiefly as locomotive chargers, readily to 1 000 lb. Because of the large range of compression, necessary between cylinders, and ante- and after-coolers mpressor capacity required depends on pipe-line press, perated, capacity of locomotive tanks, air press carried, arging periods.

Norwalk Three-stage Locomotive Charger

varied according to work desired, height above sea level, and

power	Steam pipe, in	Exhaust pipe, in	No	Steam cyl, in	Stroke, in	Rev per min	Horse-power	Steam pipe, in	Exhaust pipe, in	Compressors No 8 & 12 can be equipped with Corliss steam cylinders
	2	2½	7	20	24	100	135	5	6	
	2½	3	8	22	24	100	165	5	6	
	3	4	9	24	30	85	210	6	8	
	3	4	10	28	30	85	285	8	10	
	4	5	11	30	36	80	435	8	10	
	4	5	12	36	36	80	550	10	12	

nd Straight-line Three-stage Compressors, Class AA-13, 1 000 lb Press

Rev per min	Piston speed, ft per min	Displac't, cu ft free air per min	I h p in steam cyl at air press given, & at sea level		Approx shipping wt, lb		Floor space, ft and in	
			800 lb	1000 lb	Domestic	Export	Length	Width
160	373	201	70	75	17 900	20 000	18 8	4 0
150	400	313	108	116	21 300	23 700	20 3	4 6
140	420	375	130	140	26 800	30 000	22 0	4 9
130	455	518	180	193	34 500	38 450	25 0	5 4
120	480	661	229	246	43 700	46 800	27 6	5 8
110	495	863	300	321	51 500	56 675	30 0	6 1

nd Four-stage, Duplex Steam-driven Compressor, Class O-24, 1 000 lb Press

Stroke, in	Rev per min	Piston speed, ft per min	Displac't, cu ft free air per min	Hp in steam cyl		Approx shipping wt, lb		Floor space, ft and in	
				800 lb	1 000 lb	Domestic	Export	Length	Width
16	160	426	302	102	109	30 380	33 630	21 6	8 0
18	145	435	444	147	156	47 630	51 970	24 6	9 0
21	125	438	609	201	214	66 400	71 800	28 0	10 0
24	110	440	816	270	286	74 300	80 800	30 0	10 0
27	100	450	1 036	342	364	87 451	95 000	32 6	12 6
30	90	450	1 260	424	452	127 000	136 800	37 0	14 9
36	75	450	1 505	506	539	156 000	167 400	41 0	15 6

npressors are of Class OC–24. In all of them there are 4 single-st and second stages, forming 1 tandem unit behind 1 ste— rth-stage air cylinders, another tandem unit on the other

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If C = comp cap in cu ft free air per min and L = locomotive free air; N = number of charges required in given time, T; then if $N = 3$, $L = 5\,200$ (corresponding to 100 cu ft of air at 750 lb g $C = (3 \times 5\,200) \div 60 = 260$ cu ft free air per min. When the lo at approx equal intervals throughout the day, one application o otherwise, max and min rates of consumption must be determined.

Compressors in Table 11 are standard Norwalk locomotive cha 3-stage compressor taking air at atmos press is desired. Steam en valves and speed governor. In operation, air is admitted to lar center of machine, for first stage of compression. Water jacket a stract most of the heat, delivering air to second cyl at or near t From second cyl, air is led to second intercooler, and thence into t pression is completed. All cylinders are water-jacketed.

5. Turbo-compressors (8–11)

Turbo-compressors are essentially rotating impellers, ings having on their inner surfaces stationary vanes. In p reverse of the water or steam turbine, the fluid being acte instead of actuating them. They have from 1 to 30 stages separate impeller with discharge vanes and casing, and eac a few pounds only. ADVANTAGES: simplicity, economy of s freedom from vibration (due to the steady, non-pulsating angular velocity, large overload capacity, high effic under lower initial cost and lower cost of maintenance, attendan dom from danger of explosion in air receiver or pipe lines, oil in the interior of the cylinders. The only actual conta bearings. Floor space required is 25 to 33% less than tha machine of equal capacity, and weight is 50 to 60% that o steam-driven compressor. Steam-driven turbos are re constant press. They have a speed governor, the action mented by a fluid-operated piston, with one side connecte

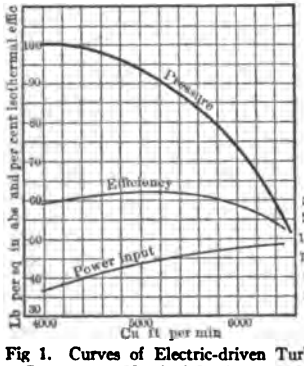


Fig 1. Curves of Electric-driven Turbo-Compressor; Nominal Rating, 5 500 cu ft Free Air per min; Discharge Press, 84 lb abs; Speed, 2 980 rev per min (Ingersoll-Rand Co)

Sizes are 3 500 air per min, with 1 Ingersoll-Rand tur pressing to twice at cooled through dia jackets.

Tests of 6 turbo-c many for So Africa, ga max isothermal effic, cu ft free air per mi Rated capacity, 21 2 at 68° F to 118 lb per cu ft per min were con machine was operate chronous motors, at Machines above 6 00 should develop 70 to 8 3 000 and 6 000 cu ft, full load, and below 5

Tests of 2 Rateau t first, built in 1909 for brücken, is driven a a 1 000-hp, mixed-p

Compressing 264 900 cu ft free air per hr to 99.6 lb abs, the consu (taken at 15.6 lb abs and discharged at 1.14 lb) was 7.98 lb per 10 was used 5.05 lb live steam at 85 lb abs). The second, for a We

, and has 6 wheels in the low-press group, 7 in the high. Be- the impeller wheels are water-cooled partitions. Air is dis-

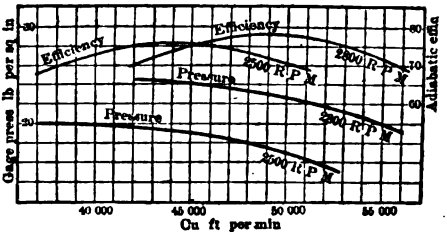
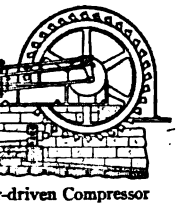


Fig 2. Pressure, Volume, and Efficiency Curves of a Turbo-blower, at 55 000 cu ft per min and 20 to 30 lb press (Ingersoll-Rand Co)

e center, thus balancing end thrust. These blowers are similar pressor, the speed of the impellers being controlled by vary- of a Venturi tube in the intake.

. Water-driven Compressors

e impulse wheel (as the Pelton, Risdon, or Knight) is best t or belt drive for compressors. Their efficiency ranges from about 70 to 85%. Peripheral speed should be 40 to 50% of the velocity of the jet of water.



-driven Compressor

Examples. At the Morning mine, Idaho, high and low-press cylinders are placed tandem on each side of a set of 3 Pelton wheels, one 33-ft and two 12-ft diam, mounted on the crank shaft. The cyl are 33.5 and 18 in diam by 42-in stroke; piston speed, 560 ft. Air is compressed in the low-press cyl to ess, to 90 lb. Inter- and after-coolers are placed in the tail race. l, a 30-ft Pelton wheel drives a 300-hp 2-stage compressor. There nd 18.5 by 30 in, set at 30° to the horizontal. The wheel makes l of 775 ft, the water being delivered through a 1.75-in nozzle (3).

4. Norwalk Water-driven Compressors

roke	Rev per min	Capac, cu ft	Hp	Pipes, in		
				Air	Water	
12	240	260	46	3	0.75	Note. — Diam of water wheel varies according to local water-power conditions.
12	240	376	67	3.5	"	
12	240	513	92	4	"	
16	200	570	102	4.5	1	
16	200	744	134	4.5	1	
16	200	942	170	5	1	
20	170	1 236	222	5	1.25	
24	150	1 309	235	6	"	
24	150	1 584	285	6	"	
24	150	1 885	340	6	"	
24	150	2 212	398	8	"	
24	150	2 566	450	8	"	

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Table 15. Sullivan Duplex, Two-stage, Water-driven Compress

Cyl dimensions, in			Displac't capac, cu ft free air		Rev per min	Terminal press, lb	Hp	F
Diam		Stroke	Per rev	Per min				Len
Low press	High press							
20	12	16	5.78	1 215	210	100	244	13
21	12.5	16	6.38	1 340	210	100	266	13
22	13	16	7.0	1 470	210	100	289	13
23	14	18	8.6	1 720	200	100	337	15
24	14.5	18	9.38	1 875	200	100	364	15
25	15	18	10.17	2 035	200	100	392	15
26	15.5	18	11.0	2 200	200	100	420	15
26	15.5	20	12.21	2 320	190	100	442	16
27	16	20	13.20	2 500	190	100	473	16
28	16.5	20	14.20	2 695	190	100	505	16
29	17.5	20	15.20	2 890	190	100	538	16

Note. — Cyl proportions in Table 15 are for sea level. For altitudes a diam of low-press cyl is proportionately larger than that of the high-press

7. Power-driven Compressors

Types. These compressors are driven by belt, chain or gearin electric, gas, or other motor. The air ends are similar, in arrang and details, to those of the direct-connected steam-driven type. electric power is low, as in some Western mining districts, this pressor may be more economical than one operated by steam, and power is available, may often be used as auxiliary to main compresse

Relative merits of power drives. Belt-drive is commonest; efficienc Wear of belting must be considered. Silent-chain drive has about 95 % ef

Table 16. Straight-line Power-driven Ingersoll-Rogler Compressors, Class ER-1

Cylinder, in		Rev per min	Piston displ't, cu ft per min	Air press, lb gage	Brake hp at motor, including belt loss	Belt wheel	
Diam	Stroke					Diam, in	Face, in
6	6	275	52	80–125	8 –10	36	5.5
7	6	275	72	50–100	9.5–12	36	"
8	6	275	94	25– 50	9.5–12	36	"
9	6	275	121	10– 25	10 –13	36	"
12	6	275	215	10–	15 –20	36	
8	8	250	113	80–125	17 –22	45	8.5
9	8	250	145	60–100	19.5–24	45	"
10	8	250	179	25– 60	18 –25	45	"
12	8	250	258	15– 25	20.5–25	45	"
14	8	250	354	10– 15	21 –25	45	"
10	10	235	210	80–125	33 –38	58	10.5
12	10	235	304	50–100	38 –50	58	"
14	10	235	415	20– 50	32 –50	58	"
17	10	235	615	10– 20	27 –48	58	"
12	12	220	340	80–125	54 –61	72	14.5
14	12	220	464	45–100	53 –73	72	"
17	12	220	683	30– 45	55 –73	72	"
20	12	220	955	15–30	35 –70	72	"

Class ER-1, str-line, power-driven compressor, provided with Rogler air va very short belt-drive. Class FR-1 is a simple, steam-driven compressor, hav end as the ER-1, but with piston steam valves, which permit higher speeds t

alk Belt-driven Tandem Two-stage Compressors

	Capac, cu ft per min	Belt wheel				Power, hp		Pipes, in	
		Diam, ft	Wt, lb	Face, in	Belt	Shaft	Electric motor	Air	Water
)	200	5.5	1 058	12	Single	46	35	3	0.75
)	376	5.5	1 541	16		67	55	3.5	0.75
)	513	5 5	2 116	22		92	75	4	0.75
)	570	6.75	3 300	24		102	80	4.5	1
)	744	6.75	3 650	24		134	110	4.5	1
)	942	6.75	4 600	24		170	140	5	1
)	1 236	8	6 882	26	Double	222	175	5	1.25
)	1 309	9	8 695	28		235	188	6	1.25
)	1 584	9	10 545	34		285	225	6	1.25
)	1 885	9	12 580	40		340	275	6	1.25

Sullivan Class WJ Two-stage Compressor

plac't, cu ft Per min	Rev per min	Hp at 80 lb air	Hp at 100 lb air	L'gth, ft in	Width, ft in	Pulley Diam, in	Pulley Face, in
248	190	43	48	8 4	6 4	54	10.5
338	190	58	65	8 6	7 3	54	12.5
537	165	87	97	10 9	7 11	70	15
680	165	110	123	10 9	8 1	70	15
873	150	142	157	12 4	9 3	84	24
1 056	150	170	190	12 4	9 3	84	28
1 188	150	191	214	14 1	9 3	84	28
1 414	150	227	254	14 1	9 3	108	28
1 845	150	297	332	15 6	11 0	108	37

VI Single-stage Duplex Air Cylinders

.	.	Term press, lb	Hp				
496	190	35	54	8 4	6 6	54	10.5
676	190	25	59	8 6	7 6	54	12.5
823	165	50	112	10 9	8 0	70	15
1 074	165	35	116	10 9	8 2	70	15
1 360	65	25	118	10 9	8 4	70	15
1 515	165	20	111	10 9	8 6	70	15
2 376	150	40	275	14 1	9 3	108	28

lass WJ-3, Belt-driven and WN-3, Direct-connected, Angle Compound Compressors

isplac't capac, cu ft free air r rev	Per min	Rev per min	Hp at belt for 100 lb press	Floor space, ft and in Length	Width	Pulley Diam, in	Pulley Face, in
.78	445	250	80	8 10	4 0	70	10
.79	628	225	113	9 11.5	5 11.5	76	15
.12	886	215	160	11 2.5	7 6.5	80	19
.09	1 094	215	196	11 4.5	7 10.5	80	23

essor 16 and 9.75 in by 12-in stroke, in Table 19: at 225 r p m, 75 r p m, 88%; 125 r p m, 86%. Mech effic, 92%. Com- e-stage hp) + air i h p = 93% (tests by Sullivan Mach'y Co)

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Table 20. Ingersoll-Rand Power-driven Straight-line, 3-Class NE-3

Cylinders, in				Rev per min	Piston displ, cu ft per min	Piston displ, cu ft per rev	Brake hp at belt wheel	Bel
Low press	Inter med	High press	Stroke					Dian ft i
For 1 030–1 200 lb Terminal Press								
5 1/2	3 1/2	1 3/4	6	220	27	0.123	10	2 4
6 3/4	4 1/4	2	7	210	46	0.220	17	3 6
8	5	2 3/8	8	200	71	0.354	26	5 0
9	5 3/4	2 3/4	10	190	105	0.552	39	4 0
10	6 1/2	3	12	180	146	0.811	54	5 0
For 1 750 lb Terminal Press								
5 1/8	3 1/8	1 3/8	6	220	24	0.111	10	2
6 1/4	3 7/8	1 5/8	7	210	40	0.193	17	3 6
7 1/2	4 1/2	1 7/8	8	200	64	0.323	27	5 0
8 1/8	5 1/8	2 1/8	10	190	94	0.498	41	4 0
9 1/4	5 5/8	2 3/8	12	180	131	0.73	58	5 0
For 2 500 lb Terminal Press								
4 7/8	3	1 1/4	6	220	22	0.101	9	2
5 7/8	3 1/2	1 3/8	7	210	37	0.175	16	3 0
7	4 1/8	1 5/8	8	200	57	0.285	26	5 0
7 7/8	4 3/4	1 7/8	10	190	85	0.445	40	4 0
8 3/4	5 1/4	2	12	180	119	0.663	57	5 0

Internal-combustion engines using gasolene are widely small compressors, especially for portable or truck-mou 21, 22).

Table 21. Sullivan Portable Compressors, Cl

Power of engine	15 hp
Displacement capacity, cu ft per min	95
Terminal air press, lb	90
Diam of gasolene engine cyl, in	7 5/8
Stroke of gasolene engine cyl, in	9
Diam of compressor cyl, in	8
Stroke of compressor cyl, in	10
Rev of engine per min	400
Rev of compressor per min	165
Diam of flywheels, in	40
Pulley on engine flywheel, in	diam 18, face 8
Main air receiver, in	diam 16, length 6
Gasolene tank, in	diam 16, length 2
Capacity of gasolene tank, gal	20
Auxiliary starting receiver, in	diam 12, length 3
Front truck wheels, in	diam 36, face 5
Rear truck wheels, in	diam 44, face 5
Track gage	4 ft 8 in
Axles, center to center	8 ft 0 in
Length over all, with pole	22 ft 0 in
Length over all, without pole	11 ft 6 in
Width over all	6 ft 0 in
Height over all	6 ft 4 in
Total net wt, including pole and tools	6 100 lb

driven Compressor, Class HSG, Truck-mounted
(Chicago Pneumatic Tool Co)

Stroke, in	Max hp	Air press, lb	Fly-wheels		Receiver		Wheel base, ft and in	Approx wt complete with truck and receiver, lb
			Diam, in	Wt both wh'ls, lb	Diam, in	Length, ft		
8	14	100	42	800	30	9	9 2	6 315
10	21	100	48	1 350	34	11	10 8	7 735
12	30	100	52	1 500	36	12	12 0	9 600

s ER-1, mine-car type compressor is designed for
, for local supply of compressed air without necessity

Rogler, Class ER-1, Mine-car Type Compressor
rtable machine, steam, gasolene and motor-driven)

iston olace't, u ft r min	Air press, lb	Brake hp at motor, including gear loss	Approx wt, lb	Dimensions, ft and in	Height over rails, with 10-in wheels, ft and in
78	90–100	12–12.5	4 000	7 0 × 4 0	3 2
102	50– 60	12–13	4 100	7 0 × 4 0	3 2
145	90–100	23–24	6 000	8 3 × 4 8	3 10
179	50– 60	22.5–24.5	6 200	8 3 × 4 8	3 10
304	90–100	51–53	9 000	9 4 × 5 8	4 4
415	50– 60	52–56	9 800	9 4 × 5 8	4 4

s. It is of dirt
nstruction, com-
or d-c motor and
or truck. Chief
ation, automatic
e-gear drive, and

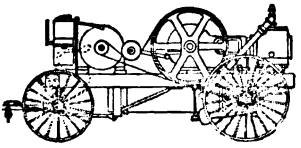

Fig 4. Sullivan Portable Electric-driven Compressor

ven portable com-
rk, as trench exca-
y are capable of running several hammer drills. Cost, $400
y.

Hydraulic Air Compression

s embrace 2 types of plant: (*a*) those driven by water wheels;
ction of falling water. For compressors under (*a*) see Art 6.

t action of falling water. The Taylor plant is the
ylor Hydraulic Air Compressor Co, Montreal, Canada.
water be allowed to fall in a vertical pipe, any air
the water on beginning its downward motion is com-
water. Then, if the direction of flow be suddenly
he velocity lessened, the air will be liberated, and may
etainer or separating chamber.

is one of the largest installations. The Montreal River, at
t 54 ft in 0.25 mile. At head of the rapids are 2 steel-lined
20 by 26-ft tunnel, 1 000 ft long, with a collecting chamber,
d shaft, 298 ft deep by 22 ft diam, at the foot of the rapids.
gh an intake head, comprising 66 pipes tapering from 14 to 5
he steel lining or compressing pipe of each shaft is 16 ft diam
ad is set; it then rapidly reduces to 8.5 ft diam, running

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Table 29. Comparison of Compressor Valves (O. P. Hoo

Valves	Liability to leakage	Power losses			
		Loss by leakage	Other losses	All losses	
				Inlet	Discharge
Poppet	20%	1.32%	6.88	2.76	8.86
Corliss	26.4%	0.91%	5.63	5.35	7.45
Riedler	37.5%	0.88%	7.29	8.31	7.78
Piston inlet	16.6%	0.93	1.98	2.73	.....

with the plotted adiabatic. When this curve fell below the adiabatic, it wa
both ends, as it would be were there a cooling effect, and evidence of le
valves was found in the expansion line of the clearance air.

11. Governors, Regulators, and Unloaders (3)

Classification: (*a*) Modified centrifugal or flyball governors, or other devices, for combining speed control with REGULATION OF AIR PRESSURE; (*b*) air-cyl UNLOADERS, which automatically control the operation of belt-

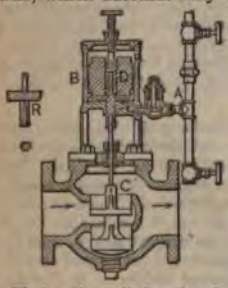


Fig 5. Norwalk Speed and Pressure Regulator

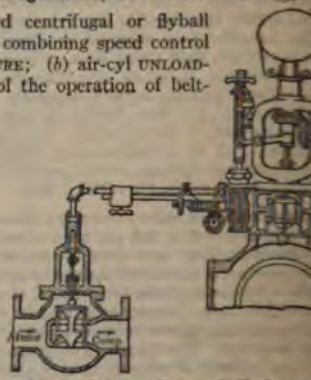


Fig 6. Chicago Pneumatic Tool Co's and Unloader

driven compressors, or are used in conjunction with a governor for st
compressors.

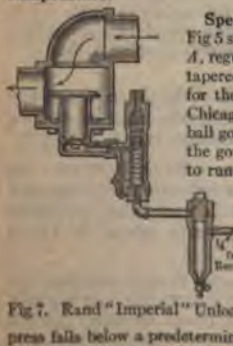


Fig 7. Rand "Imperial" Unloader

Speed and pressure regulators for steam-driven
Fig 5 shows the Norwalk governor. Air entering cylind
A, regulates steam throttle *C*. The rise of piston *D* i
tapered recess *R* in the piston rod, furnishing a grad
for the air, and preventing complete throttling. Fig
Chicago Pneumatic Tool Co's governor and unloader
ball governor is belt-driven from the crank shaft, and
the governor spindle, regulated by springs and a hand
to run the compressor at the desired speed. A small
and piston are connected with the air receiv
and needle-valve, with adjustable weight.
press becomes excessive the small piston ris
closes the throttle. Complete closure and
the compressor is prevented by a stop and
The Sullivan governor is similar in general
walk skip valve is used as an inlet valve
press cyl in stage compressors. When th
press falls below a predetermined point, the skip valves remain open, causing
to reject its supply of air, throwing it back into the intercooler.

ers for belt or motor-driven compressors. Nordberg governor, s the inlet valve before the stroke is completed, combining le delivery. Rand "Imperial" unloader (Fig 7) has a plunger e pipe. Its position is regulated by a small spring-controlled ith the air receiver (3).

Cooling Devices and Receivers

ease compressor efficiency by reducing the temp of the ed. This also decreases danger of explosion (Art 14) ter compression.

Initial Air Temp on Efficiency and Capacity of Air Compressors (Ingersoll-Rand Co)

re	Initial temp		Relative capac	Initial Temp		Relative capac
ic	° F	abs Deg	and effic	° F	abs Deg	and effic
	40	501	1.040	110	571	0.912
	50	511	1.020	120	581	0.896
	60	521	1.000	130	591	0.880
	70	531	0.980	140	601	0.866
	80	541	0.961	150	611	0.852
	90	551	0.944	160	621	0.838
	100	561	0.928	.....	.....	.....

capacity and efficiency assumed at 60° F.

oling; (*b*) cooling during compression; (*c*) inter-cooling; OOLING is by leading the air to the compressor from the cool- r by use of ante-coolers (similar to after-coolers). COOLING direct contact between the air and water (as in wet compressors,

y use of INTER- ompres- ssed air proper s largely n. Fig a shell through l baffle- around

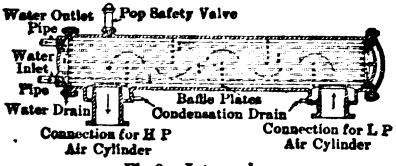
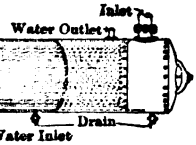


Fig 8. Intercooler

AFTER-COOLER cools the air, and therefore deposits moisture, elivery pipe. Fig 9 shows a Sullivan after-cooler.



r Compressor, Capacity t Free Air per Min

Air receivers are plain steel shells, which cool and reduce the velocity of the air before it passes into the main, causing deposition of moisture where it can be conveniently drained off; they equalize flow of air; eliminate pulsating effect of the piston strokes, thus minimizing friction losses; and serve in some degree as res- best results, the receiver must be close to the compressor. the other end of the air main, and near the operating eous.

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Receiver capacity required depends upon rate of air consumption, a drop. The higher the press, the smaller the receiver proper for a giv Receivers and after-coolers may be either vert or horiz.

Table 31. Dimensions, Weights, Sullivan Air Recei

	Diam. in	L'gth, ft	Size of outlet & inlet, in	Capac, cu ft	W'g lb
Vertical	30	6	3	30	7
	36	6	3	42	9
	36	8	4	57	1 2
	42	8	4	77	1 4
	42	10	5	96	1 6
	48	10	6	126	2 0
	48	12	6	150	2 3
	54	12	6	191	3 1
Horizontal	30	6	3	30	8
	36	6	3	42	1 0
	36	8	4	57	1 3
	42	8	4	77	1 5
	42	10	5	96	1 7
	48	10	6	126	2 0
	48	12	6	150	2 3
	54	12	6	191	3 1
	54	16	7	255	3 5
	60	14	7	275	3 9
	60	16	7	314	4 3
	66	16	7	376	6 3

13. Care of Compressors

General. So locate the compressor that parts are readily accessible. Foundations must be level in both directions and, in power-driven machines, the motor and compressor must be accurately alined.

Air valves. Examine at least once a month, to see that there is no cutting, and that the springs, if any, are in good order.

Cutting is caused by ineffective lubrication or grit entering inlet. If the latter, remove the cause of dust, or change position of intake. Keep spare valves on hand for prompt renewal, before wear or defect becomes serious. Leaky discharge valves greatly reduce volumetric effic.

Air lines. Watch closely for leaks, which are costly. A waste enough air to run a 2.25-in drill.

Lubrication must be sufficient, but not excessive.

An air cyl requires less oil than steam cyl of same size. Use best a valves, having a flash point not lower than 500° F. Never use ker deposits in the exhaust valves and ports, as it has a low flash point cause explosion (Art 14).

14. Explosions in Compressors and Receive

Causes: excessively high internal press, due less to the air to that produced by ignition, in compressor, piping, or receive mixture of air and gas from lubricant. The underlying causes ignorance, improper use or kind of lubricant, or faulty design

Lubricants in general use are: commercial cylinder oil, and a n water, each having its proper function. Soap and water has i qualities; if used alone a much greater quantity is necessary than

In a compressor lubricated almost exclusively by soap and water, was found, which readily ignited at 400° F. A very small quantity of of 400° F had also been used, which indicates that the use of soap ventive of explosions. Nevertheless it will clean the cylinder and v ting down, and its use is recommended. All oils give off comb heated. The lowest temp at which this begins is the FLASH POINT, being the BURNING POINT. As ordinary lubricating oils flash at ab below the usual working temp of compressors), special high-flash cyl

Temp due to compression depends upon initial temp, t and the effic of the cooling devices,

gle-stage compressor is found by $T' = T\left(\frac{P'}{P}\right)^{\frac{n-1}{n}} = 0.29$,

ial and final temp, P and P' = abs initial and final press,
mula, for adiabatic compression, is not absolutely correct,
ery little loss of heat by radiation. Near sea level, at
nd at 80 lb gage press the final temp is

$)^{0.29} = 917°$ F, absolute, or 458° F thermometric.

temp may be materially higher (3).
om imperfect combustion of the oil, and carried with the
ay cause danger to the miners.

high temp:

e adapted to the conditions; cyl proportions for sea level
des. (*b*) Intake pipe should be of wood or other insulat-
e taken from as cool a place as possible OUTSIDE of the
5° F may increase effic by 1%. (*c*) Unloader should be
e heating when in operation. (*d*) Largest possible area of
and plenty of the coldest water obtainable used. (*e*) For
it inter-coolers. (*f*) After-coolers increase efficiency, and
ating water be reused, provide ample water coolers. (*h*)
op and outlets about 1 ft above the bottom, to insure clean-
r should have blow-off cocks at the bottom, and man-hole
(*j*) Place a recording thermometer between high-press cyl
atic blowout valve, to act if temp rises above a safe point,
ould be free from dust; washed if necessary. (*m*) While
e into the compressor to cut carbon deposit.

Manufacturers of Compressors

Westminster	General Electric Co, Schenectady, N Y
	Ingersoll-Rand Co, New York
ee, Wis	International Steam Pump Co, New York
ra, Ill	(Laidlaw-Dunn-Gordon and Clayton)
rd, Pa	McGowan Co, J H, Cincinnati, Ohio
p Co, New York	McKiernan-Terry Drill Co, New York
, Chicago, Ill	National Brake & Electric Co, Milwaukee,
hia, Pa	[Conn
o, Indianapolis,	Wis
[N J	Norwalk Iron Works Co, South Norwalk,
, East Trenton,	Sullivan Machinery Co, Chicago, Ill
icago, Ill	Waterworks Equipment Co, New York
	Westinghouse Air Brake Co, Pittsburg, Pa

hen inquiring about air compressors. When writing
nation, give as complete data as possible regarding service
al conditions. Following points should be covered:

air is to be used. (*b*) Volume of free air required, cu ft per min.
(*d*) Altitude, if over 1 000 ft above sea-level. (*e*) Number, size,
operated by compressed air. (*f*) If air is to be used for pumps,
head. (*g*) If for raising water by the air lift, state flow per min
well, and height of delivery above aver height of water in the well.
r air will be constant or intermittent. (*i*) Whether the compres-
am or power. (*j*) If steam-driven, state steam press, kind and
e preferred, and whether condensing or non-condensing. (*k*) If
e power, and whether direct connection, belt, or gearing is pre-
give hp at belt, and if space is limited, state max distance allow-
ers. (*m*) If electric-driven, give particulars as to current and
ver is to be used, give hp available, or head or fall of water in ft;
. (*o*) State transport facilities; if machine must be sectionalized,
e for a single package. (*p*) State style of compressor preferred; if
r surface or underground service, and kind and source of power.

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16. Transmission of Compressed Air in

Pipe lines. Wrought iron and steel pipe is lap or bu weld is the stronger, it is used for the larger sizes.

Pipes up to 3 in are usually butt-weld, though lap-weld pipe as sr Pipe and fittings should be galvanized inside, as the scale from machines using the air. EXTRA-HEAVY PIPE for high press may be riveted pipe is useful for large sizes. Rolled sheets, with punched ing, are convenient for transport in remote regions.

Sizes and weights of standard, extra heavy and double e fittings, are given in Sec 41.

Joints. W-I pipe lengths are connected by sleeve couplings, which the pipe ends are expanded or threaded. Sleeve couplings, all except very large sizes, should b red lead, especially where leaks m ground. Gaskets are used for flan near the receiver, brown paper e JOINTS are necessary on long lines, b avoided as they are likely to leak.

Cost of laying gas mains of 4, 6 an sleeve-joint, 6 and 8-in air pipe, is gi Data," pp 1 802, 1 804.

Table 32. Cost per Foot of Air Lines on the Rand (39)

Nominal diam, in	Surface	Underground, cents
1	$0.18	$0.24
1.25	0.24	0.32
1.5	0.30	0.40
2	0.43	0.58
2.5	0.56	0.76
3	0.71	0.94
3.5	0.86	1.16
4	1.02	1.36
4.5	1.18	1.56
5	1.34	1.80
6	1.70	2.26
7	2.04	2.74
8	2.42	3.26
9	2.82	3.74
10	3.30	4.28
12	4.02	5.36

Transmission losses in com Sec 39). The heat of compressi the first few hundred ft of air m nomically be retained by non-c Before using air expansively, it (Art 20).

Transmission line hints. Loss unsound pipe often exceed all oth Pipes should be inspected regularly power. Pipe of too small diam red causing high velocity and undue fric ity in mains should not exceed 20 to branch pipes, it may be 40 or 50 ft. causes excessive friction loss. Each length should be cleaned of fo coupling. Lead forced into the pipe at couplings makes obstructive should be protected, to avoid freezing of the moisture and consequ elbows, and other fittings cause friction and should be avoided where

Table 33. Friction of Globe Valves, Tees, an

Reduction of press by GLOBE VALVES is the same as that caus of straight pipe, as follows:

Added length = (114 × diam of pipe) + [1 + (3.6 ÷						
Diam of pipe, in.....	1	1.5	2	2.5	3	3.5
Added length, ft.....	2	4	7	10	13	16
Diam of pipe, in.....	7	8	9	12	15	18
Added length, ft.....	44	53	70	88	115	143

Reduction of press by ELBOWS and TEES is equal to two-thirds o valves. Following are the added lengths of straight pipe equivale

Diam of pipe, in.....	1	1.5	2	2.5	3	3.5
Added length, ft.....	2	3	5	7	9	11
Diam of pipe, in.....	7	8	10	12	15	18
Added length, ft.....	30	35	47	59	77	96

These additional lengths of pipe for globe valves, elbows, and tees case to length of straight pipe. Thus a 6-in pipe, 500 ft long, with 1 gl 3 tees, would be equivalent to a straight pipe 500 + 36 + (2 × 24) +

pressed Air Transmission (Ingersoll-Rand Co)

Diam of pipe, in										
2½	3	3½	4	4½	5	6	7	8	9	10
Loss of press, in lb, by friction in 1 000 ft of pipe										
0.39	0.14	0.06	0.03	0.02	0.01	.....	.....	.....	.....	.....
0.64	0.24	0.11	0.05	0.03	0.01	.....	.....	.....	.....	.....
0 91	0.34	0.15	0.07	0.04	0.02	0.01	.....	.....	.....	.....
1.25	0.47	0.21	0.10	0.06	0.03	0.01	.....	.....	.....	.....
1.63	0.61	0.27	0.13	0.07	0.04	0.01	.....	.....	.....	.....
2.53	0.96	0 43	0.21	0.12	0.07	0.02	0.01	.....	.....	.....
3.67	1.38	0.62	0.30	0.17	0.09	0.03	0.01	.....	.....	.....
4.99	1.88	0 84	0.41	0.23	0.13	0.05	0.02	0.01	.....	.....
6 53	2.45	1.11	0.54	0.30	0.17	0.06	0.03	0.01	.....	.....
8.25	3.13	1.40	0.69	0.38	0.22	0.08	0.04	0.01	.....	.....
0.81	3.83	1.73	0.85	0.47	0.27	0.10	0.05	0.02	0.01	.....
....	5 61	2.46	1.22	0 68	0.39	0 15	0.06	0.03	0.02	0.01
....	7.46	3.37	1.66	0.92	0.53	0.20	0.09	0.04	0.02	0.01
....	9.86	4.42	2.18	1.19	0.69	0.27	0.12	0.06	0.03	0.02
....	.....	5 61	2.77	1.54	0.88	0.34	0.15	0.08	0.04	0.02
....	.....	6.64	3.29	1.82	1.04	0.40	0.18	0.09	0.05	0.03
....	.....	15.41	7 62	4.24	2.43	0.95	0.43	0.22	0.12	0.06
....	.....	.....	13.62	7.58	4.32	1.69	0.77	0.39	0.21	0.12
....	.....	.....	.....	11.79	6.88	2.64	1.19	0.60	0.33	0.19
....	.....	.....	.....	.....	9.72	3.79	1.73	0.87	0.48	0.28
....	.....	.....	.....	.....	13.25	5.27	2.35	1.19	0.65	0.37
0.34	0.12	0.05	0.02	0.01	.....	.....	.....	.....	.....	.....
0 52	0.19	0.08	0.04	0.02	0.01	.....	.....	.....	.....	.....
0.76	0.29	0.13	0.06	0.03	0.02	0.01	.....	.....	.....	.....
1 03	0.39	0.17	0.09	0.04	0.03	0.01	.....	.....	.....	.....
1.36	0.51	0.23	0.12	0.06	0.04	0.02	0.01	.....	.....	.....
2.06	0.77	0.35	0.17	0.09	0.05	0.02	0.01	.....	.....	.....
3.04	1.14	0.51	0.25	0.14	0.08	0.03	0.01	.....	.....	.....
4.11	1.54	0.69	0.34	0.19	0.11	0.04	0.02	0.01	.....	.....
5.40	2.06	0.92	0.45	0.25	0.15	0.05	0.03	0.01	.....	.....
6.85	2.57	1.16	0.57	0.32	0.18	0.07	0.03	0.02	0.01	.....
8.21	3.08	1.39	0.68	0.38	0.22	0.08	0.04	0.02	0.01	.....
9.21	4.58	2.14	1.03	0.57	0.33	0.12	0.05	0.03	0.02	0.01
....	6.19	2.79	1.38	0.77	0.44	0.17	0.07	0.04	0.02	0.01
....	8.13	3.67	1.81	1.00	0.57	0.22	0.10	0.05	0.03	0.02
....	10.23	4.64	2.29	1.27	0.76	0.28	0.13	0.06	0.04	0.02
....	12.39	5.00	2.76	1.23	0.85	0.34	0.16	0.08	0.04	0.03
....	.....	12.81	6.68	3.51	2.03	0.78	0.36	0.18	0.09	0.05
....	.....	.....	11.35	6.61	3.62	1.41	0.67	0.33	0.18	0.10
....	.....	.....	.....	9.50	5.51	2.14	0.97	0.49	0.27	0.16
....	.....	.....	.....	14.04	8.11	3.16	1.44	0.76	0.39	0.23
....	.....	.....	.....	.....	10.95	4.26	1 93	0.98	0.53	0.31
0.28	0.10	0.05	0.02	0.01	.....	.....	.....	.....	.....	.....
0 43	0.16	0.07	0.01	0 02	0.01	.....	.....	.....	.....	.....
0.63	0.23	0.11	0 05	0.03	0.02	0.01	.....	.....	.....	.....
0.85	0 32	0.14	0.07	0.04	0 02	0.01	.....	.....	.....	.....
1.11	0.42	0.19	0.09	0.05	0.03	0.01	.....	.....	.....	.....
1.73	0.65	0.29	0.15	0.08	0.05	0.02	0.01	.....	.....	.....
2.51	0.94	0.42	0.21	0.12	0.07	0.03	0.01	.....	.....	.....
3.39	1.27	0.58	0.28	0.16	0.09	0.04	0.02	0.01	.....	.....
4 44	1.67	0.75	0.37	0.21	0.12	0.05	0.02	0.01	.....	.....
5.61	2.11	0.95	0.47	0.26	0.15	0.06	0 03	0.01	0.01	.....
6.95	2.61	1.18	0.58	0.32	0.18	0.07	0 03	0.02	0.01	.....
0.00	3.76	1.69	0.84	0.46	0.27	0.10	0.05	0.02	0.01	0.01
3.60	5.11	2.31	1.14	0.63	0.36	0.14	0.06	0.03	0.02	0.01
7.80	6.68	3.01	1.49	0.83	0.47	0.18	0 08	0.04	0.02	0.01
....	8.45	3.81	1.88	1.04	0.60	0.23	0.11	0.05	0.03	0.02
....	10.42	4.71	2.32	1.29	0.74	0.29	0.13	0.07	0.04	0.02
....	23.48	10.59	5.23	2.90	1.65	0.64	0.29	0.15	0.08	0.05
....	.....	18.81	9.30	5.15	2.94	1.15	0.52	0.26	0.15	0.08
....	.....	29.40	14.52	8.05	4.60	1.80	0.82	0.41	0.23	0.13
....	.....	.....	20.90	11.59	6 63	2.59	1.18	0.60	0.33	0.19
....	.....	.....	28.51	15.78	9 01	3.53	1.61	0.81	0.45	0.26

es, friction loss is proportional to length; thus, for 500 ft, t 4 times the above, etc.

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Table 35. Theoretical Volume of Free Air, in Cu Ft, that straight pipes, 100 ft long, without loss of press (Chicago

Gage press	Nominal diam of pipe, in							
	1	1½	2	2½	3	3½	4	5
60	42	131	261	425	753	1 105	1 535	2 747
70	44	140	278	453	802	1 176	1 634	2 925
80	47	148	294	479	847	1 244	1 728	3 091
90	49	155	309	504	892	1 307	1 817	3 253
100	52	163	324	527	934	1 370	1 904	3 407
110	54	170	338	550	974	1 429	1 985	3 552
125	57	179	357	581	1 028	1 510	2 097	3 754
150	62	195	388	631	1 117	1 641	2 280	4 080
175	66	209	416	678	1 201	1 762	2 448	4 382
200	71	223	443	721	1 277	1 874	2 603	4 659
250	78	247	491	800	1 416	2 079	2 889	5 170
300	86	270	536	873	1 546	2 269	3 152	5 642
400	98	310	615	1 002	1 775	2 606	3 619	6 477
500	109	345	685	1 116	1 976	2 902	4 031	7 215

D'Arcy's formula for transmission of air in pipes. A gr
of determining the economic size are given in *Trans* A I M E,

17. Air Compression at Altitudes

Effect of altitude. Donsity of the atmos is less at
and the compressor output is diminished because: (*a*
reduced on account of the smaller wt of air taken into
(*b*) due to lower initial press, the heat of compression

Volume of air delivered decreases with barom press,

The relation between press and vol of air delivered by comp
higher altitudes is as follows: If P = final gage press in both
gage press at sea level and at higher altitude; V_1 and V_2 = fi
higher altitude; then, $V_2 + V_1 = [1 + (P + P_1)] \div [1 + (P$
Example: Let $P_2 = 10$ lb, $P = 90$ lb, and V_1 (Table 36) = 0.1

Table 36. Volumes, Press and Temp, in Compressin and 60° F (Sullivan Mach'y Co)

Gage press, lb	Abs press, lb	Press, atmos	Vol, air at constant temp	Vol, air not cooled	Mean pr per stro Air at constan temp,
50	64.7	4.40	0.227	0.35	21.79
55	69.7	4.74	0.211	0.331	22.77
60	74.7	5.08	0.197	0.314	23.84
65	79.7	5.42	0.184	0.301	24.77
70	84.7	5.76	0.173	0.288	26
75	89.7	6.10	0.164	0.276	26.65
80	94.7	6.44	0.155	0.267	27.33
85	99.7	6.78	0.147	0.256	28.05
90	104.7	7.12	0.140	0.248	28.78
95	109.7	7.46	0.134	0.24	29.53
100	114.7	7.80	0.128	0.232	30.07
105	119.7	8.14	0.123	0.225	30.81
110	124.7	8.48	0.118	0.219	31.39
115	129.7	8.82	0.113	0.213	31.98
120	134.7	9.16	0.109	0.207	32.54
125	139.7	9.50	0.105	0.202	33.07

netric Outputs of Compressors at Altitudes

lative put for e press		Mean effective press for gage press		Cu ft piston displace't per i h p for gage press		Cu ft compressed air per i h p for gage press		Decreased power required %
	90 lb	70 lb	90 lb	70 lb	90 lb	70 lb	90 lb	
o	1.000	33.1	38.2	6.93	5.99	1.144	0.801	0.0
7	0.966	32.6	37.6	7.03	6.09	1.123	0.787	1.8
5	0.933	32.1	36.9	7.15	6.20	1.103	0.773	3.5
4	0.900	31.5	36.3	7.27	6.31	1.084	0.759	5.2
3	0.869	31.0	35.6	7.39	6.43	1.065	0.746	6.9
3	0.839	30.5	35.0	7.51	6.55	1.046	0.733	8.5
3	0.809	30.0	34.3	7.65	6.67	1.028	0.720	10.1
5	0.780	29.4	33.7	7.80	6.79	1.011	0.708	11.6
8	0.751	28.9	33.1	7.94	6.92	0.994	0.695	13.1
1	0.723	28.3	32.5	8.09	7.06	0.976	0.683	14.6
5	0.696	27.8	31.8	8.24	7.20	0.959	0.670	16.1
o	0.671	27.4	31.2	8.39	7.34	0.942	0.658	17.6
6	0.647	26.9	30.6	8.54	7.49	0.925	0.646	19.1
2	0.623	26.3	30.0	8.71	7.64	0.908	0.635	20.6
8	0.600	25.8	29.4	8.88	7.80	0.891	0.624	22.1
5	0.576	25.3	28.8	9.06	7.96	0.875	0.613	23.5

high altitudes. In stage compressors the only clear- tput is that of the intake cyl, and, as loss due to clear- le, this loss per unit of work is less in stage than in

one at sea level (atmos press, 14.7 lb), the other at 5 000 ft . If the first is producing 6 compressions, its final abs press r 73.5 lb gage press. To produce the same press, the other lbs press of 73.5 + 12.2 = 85.7 lb, for which 85.7 ÷ 12.2 = But, at sea level, 7 compressions would produce (7 × 14.7) Since the quantity of heat increases with the final press, compressing from low initial press than from high. Hence, the cyl of stage-compressors is more valuable at high than

Compressed Air into Free Air (Sullivan Mach'y Co)

Gage pressure										
60	65	70	75	80	85	90	95	100	105	110
5.08	5.42	5.76	6.10	6.44	6.78	7.12	7.46	7.80	8.14	8.48
5.15	5.50	5.83	6.19	6.53	6.88	7.23	7.57	7.92	8.26	8.61
5.24	5.60	5.95	6.31	6.66	7.02	7.37	7.72	8.08	8.43	8.79
5.31	5.67	6.03	6.39	6.75	7.10	7.46	7.82	8.18	8.54	8.90
5.41	5.77	6.14	6.51	6.88	7.24	7.61	7.98	8.34	8.71	9.08
5.58	5.96	6.34	6.72	7.10	7.49	7.87	8.25	8.63	9.01	9.40
5.76	6.15	6.55	6.95	7.34	7.74	8.14	8.53	8.93	9.32	9.72
5.94	6.35	6.76	7.17	7.58	7.99	8.40	8.82	9.22	9.64	10.05
6.16	6.53	6.96	7.38	7.81	8.23	8.66	9.08	9.51	9.93	10.36
6.32	6.77	7.21	7.65	8.10	8.54	8.98	9.43	9.87	10.32	10.76
6.53	6.99	7.45	7.91	8.37	8.83	9.29	9.75	10.21	10.68	11.14
6.74	7.22	7.70	8.19	8.67	9.13	9.61	10.09	10.57	11.05	11.52
6.94	7.43	7.93	8.42	8.92	9.41	9.91	10.40	10.90	11.40	11.88

of air compressed to 95 lb at 4 000 ft altitude. Opposite re 8.53. 8.53 × 348 = 2 968.44 = vol of free air.

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Free air is air at normal atmos press, but the term has no preci
the atmos press varies with altitude and weather. In general, it
press and 60° F.

Table 39. Wt of 1 Cu Ft Free Air at Various Altitud

Temp, deg Fah	Vol of 1 lb, cu ft	Weight of 1 cu ft, lb				
		Sea-level, abs press, 14.72 lb	2 500 ft, abs press, 13.34 lb	5 000 ft, abs press, 12.14 lb	7 500 ft, abs press, 11.84 lb	10 000 ft abs press 10.04 lb
0	11.582	0.0863	0.0785	0.0713	0.0648	0.0589
10	11.834	0.0845	0.0768	0.0698	0.0635	0.0577
20	12.085	0.0827	0.0752	0.0683	0.0622	0.0565
30	12.336	0.0811	0.0737	0.0669	0.0609	0.0554
32	12.386	0.0807	0.0734	0.0667	0.0606	0.0551
40	12.587	0.0794	0.0722	0.0656	0.0596	0.0542
50	12.838	0.0779	0.0708	0.0643	0.0585	0.0532
60	13.089	0.0764	0.0695	0.0631	0.0574	0.0522
70	13.340	0.0750	0.0682	0.0619	0.0563	0.0512
80	13.592	0.0736	0.0669	0.0608	0.0552	0.0502
90	13.843	0.0723	0.0657	0.0596	0.0542	0.0493
100	14.094	0.0710	0.0645	0.0586	0.0533	0.0484
110	14.345	0.0697	0.0634	0.0578	0.0523	0.0476
120	14.596	0.0685	0.0623	0.0565	0.0514	0.0466
130	14.847	0.0674	0.0612	0.0556	0.0506	0.0459
140	15.098	0.0662	0.0602	0.0549	0.0497	0.0452
150	15.350	0.0651	0.0592	0.0538	0.0489	0.0444
160	15.601	0.0641	0.0583	0.0529	0.0481	0.0438
170	15.852	0.0631	0.0573	0.0521	0.0473	0.0431
180	16.103	0.0621	0.0565	0.0513	0.0466	0.0424
190	16.354	0.0612	0.0556	0.0504	0.0459	0.0417
200	16.605	0.0602	0.0547	0.0497	0.0452	0.0411
210	16.856	0.0593	0.0539	0.0489	0.0445	0.0405
212	16.907	0.0591	0.0538	0.0488	0.0444	0.0404

Since barom press often varies 6 or 7% from the normal, Tabl
eccentricity. (See Sec 39, Art 9.)

Table 40. Horse-power, with Allowance for Usu

for compressing 1 cu ft free air at various altitudes, from atmos p
(Initial temp of air in each cyl, 60° F; jacket cooling not

Altitude, ft	Gage press, lb					
	60	80	100	60	80	100
	Hp, simple compression			Hp, two-stage con		
0	0.1533	0.1824	0.2075	0.1354	0.1580	0.1765
1 000	0.1511	0.1795	0.2040	0.1332	0.1553	0.1734
2 000	0.1489	0.1766	0.2006	0.1310	0.1524	0.1700
3 000	0.1469	0.1739	0.1971	0.1286	0.1493	0.1666
4 000	0.1448	0.1712	0.1939	0.1263	0.1464	0.1635
5 000	0.1425	0.1685	0.1906	0.1241	0.1438	0.1600
6 000	0.1402	0.1656	0.1872	0.1218	0.1409	0.1566
7 000	0.1379	0.1628	0.1839	0.1197	0.1383	0.1536
8 000	0.1358	0.1600	0.1807	0.1173	0.1358	0.1504
9 000	0.1337	0.1572	0.1774	0.1151	0.1329	0.1473
10 000	0.1316	0.1547	0.1743	0.1132	0.1303	0.1442

Example. — Requir'd the vol of free air, which, when compres
9 000 ft, will be equivalent to 1 000 cu ft of free air at sea level; a

ge in 2-stage compression at this altitude. From Table 41, ivalent free air vol is therefore 1 000 × 1.356 = 1 356 cu ft. or compressing 1 cu ft of free air under these conditions is .1473 = 199.73 hp.

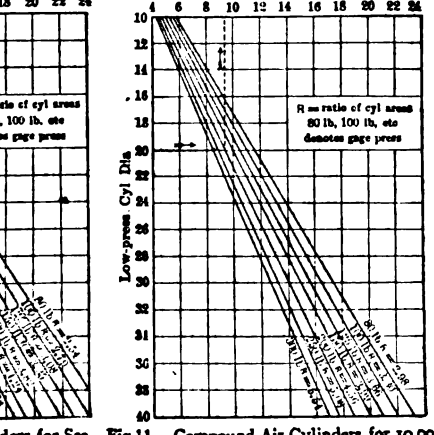
tipliers for Determining Volume of Free Air

, when Compressed, is Equivalent to a Given Vol of Free Air t Sea Level (Ingersoll-Rand Co)

r	Multipliers for gage press, lb				
	60	80	100	125	150
5	1.000	1.000	1.000	1.000	1.000
0	1.032	1.033	1.034	1.035	1.036
7	1.064	1.066	1.068	1.071	1.072
5	1.097	1.102	1.105	1.107	1.109
7	1.132	1.139	1.142	1.147	1.149
0	1.168	1.178	1.182	1.187	1.190
3	1.206	1.218	1.224	1.231	1.234
0	1.245	1.258	1.267	1.274	1.278
7	1.287	1.300	1.310	1.319	1.326
5	1.329	1.346	1.356	1.366	1.374
7	1.373	1.394	1.404	1.416	1.424

liam for stage compressors at different altitudes. ea level and altitude of 10 000 ft (*Comp Air Mag*, Oct,

oke are first computed for required piston displacement. procedure when l-p cyl diam is 20 in, and final press is 200 lb yl diam so found divides the load equally between the 2


High-press Cyl Dia
4 6 8 10 12 14 16 18 20 22 24
Low-press Cyl Dia
10 12 14 16 18 20 22 24 26 28 30 32 34 36 38 40
R = ratio of cyl areas
80 lb, 100 lb, etc
denotes gage press
R = 2.98
R = 3.84

ders for Sea b per sq in

Fig 11. Compound Air Cylinders for 10 000 ft Altitude. Atmos Press, 10.04 lb per sq in

ler abstracts all heat of compression in first stage. Ratios om the absolute ratio of compression. Example. —

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pression of 150 lb gage press is required. Abs press = 150 + 14.7 = of compression = 164.7 + 14.7 = 11.2. Ratio of l-p to h-p piston

Formula for determ

$$d = \sqrt{\frac{D^2}{\sqrt{(P_1 + P_2) \div}}}$$

of h-p cyl, D = diam press, P_2 = terminal

A common problem which will be approx c tudes and pressures. sions for the h-p cyl 30-in diam; also the 80 and 100-lb gage p 10 000 ft. Fig 10 sho level to be 18⅞ in fo

Table 42. Absolute Ratios of Compression

Gage press, lb	Sea-level	5 000 ft	10 000 ft
80	6.44	7.55	8.92
100	7.80	9.20	10.90
125	9.50	11.25	13.40
150	11.20	13.30	15.90
200	14.60	17.40	20.80
250	18.00	21.50	25.80
300	21.40	25.60	30.70

100 lb. According to Fig 11, at 10 000 ft the respective diam shou a satisfactory mean for general service would be 17 in diam.

18. Measurement of Air Consumptio

Apparatus for this purpose is varied. One consists of 2 l which can be filled with air at any desired press, the drill o be tested being connected to one of them. The tank also con to show in a gage glass. Height of water in the gage is noted, Tank press is kept constant by forcing in more water fro running long enough for accuracy, the drill is stopped, and t in the gage glass again noted. The increase in volume of w of air used, from which the cu ft of free air per min is compu tanks, one being used while the other is being filled with air, of considerable length.

Fig 12 shows a simpler and less expensive apparatus used companies.

Table 43. Discharge of Air Through an Orifice, in Cu per Min (Ingersoll-Rand Co)

Receiver gage press, lb	Diam of a round orifice in receiver, in							
	1/64	1/32	1/16	1/8	1/4	3/8	1/2	5/8
	Discharge in cu ft of free air per min							
2	0.038	0.153	0.65	2.4	9 7	21.0	39	61
5	0.059	0.242	0.96	3.8	15.4	34.6	61	96
10	0.084	0.342	0.13	5.4	21.8	49.0	87	136
15	0.103	0.418	1.67	6.6	26.7	60.0	107	167
20	0.119	0.485	1.93	7.7	30.8	69.0	123	193
25	0.133	0.540	2.16	8.6	34.5	77.0	138	216
30	0.156	0.632	2.52	10.0	40.0	90.0	161	252
35	0.173	0.710	2.80	11.2	44.7	100.0	179	280
40	0.190	0.770	3.07	12.3	49.1	110.4	196	307
45	0.208	0.843	3.36	13.4	53.8	121.0	215	336
50	0.225	0.914	3.64	14.5	58.2	130.0	232	364
60	0.260	1.050	4.20	16.8	67.0	151.0	268	420
70	0.295	1.190	4.76	19.0	76.0	171.0	304	476
80	0.330	1.330	5.32	21.2	85.0	191.0	340	532
90	0.364	1.470	5.87	23.5	94.0	211.0	376	587
100	0.400	1.610	6.45	25.8	103.0	231.0	412	645
125	0.486	1.970	7.85	31.4	125.0	282.0	502	785

4 to 5 ft long, are connected at the bottom by a 3-in pipe, and . Each has a gage glass for observing the water level, and in d 4-way valve. When the valve is set water in the right-hand tank is forced ess, and the air in the latter is driven he machine being tested. When the te point at the top of the gage glass valve is thrown to the position shown e process is reversed, the air in the the machine under test. The number ied in a given period shows the vol-d, which is expressed in cu ft of free $P = V$, in which P_1 = abs press of the ol, P and V = barom press and vol

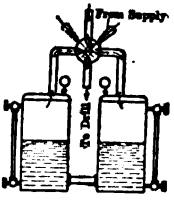

Fig 12. Displacement Tank Apparatus

ing of Moisture in Compressed Air

ordinary atmos conditions, amounts to about 1 lb water per E-CARRYING CAPACITY of air depends upon its temp, but is density. Hence, if saturated air be compressed isothermally e, its water capacity is reduced to 0.1, and 0.9 of the water air from the compressor cyl, on entering the receiver, deposits emainder is carried into the mains where it may freeze and so the pipe. Lastly, in the air engine, the intense cold of expan-noisture, which may obstruct ports and valves (3).

ng. To remove the moisture before the air reaches the may take place, several devices may be used, singly or after-cooler (Art 12), set as close to compressor as water traps, at low points in air line; (*c*) a second pressed-air machines, the air being transmitted under reduced before entering second receiver; (*d*) reheaters HODS for pumps: (*e*) lead a small jet of water from nozzle) into the exhaust port of the pump; (*f*) carry to exhaust port, if a steam line is conveniently near; d air by passing it through a coil of pipe placed in an pump column or suction pipe.

mmended (1914) by M. F. La Croix, Cleveland-Cliffs Iron Co: in it automatic drop oilers containing denatured alcohol or nt is used to make a mixture of low-freezing point (approx ft air at 90 lb press will keep a line entirely open). This lly frozen pipe, but, with an opening large enough to force be melted. It is said that so little alcohol is necessary that or pump exhausts.

Reheating Compressed Air

use: (*a*) passing the air through a cast-iron chamber or fire or current of hot gases or steam; (*b*) adding heat self, by combustion of fuel, injection of steam or hot ir line an electric-resistance coil (3). The first is com-

btainable. THERMAL COST OF AIR COMPRESSION (2): Wt of = 0.2089 lb; total heat in 1 lb steam at 75 lb, from water at tal heat in 1 cu ft steam at 75 lb = 1 151 × 0.2089 = 240.44 mpressor, about 2 cu ft of steam at 75 lb are required to pro- at 75 lb and 60°, the thermal cost being about 240.44 × here supposed to have lost its heat of compression by t

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stored or transmitted to a distance, so that the 480.88 heat-units are
THERMAL COST OF REHEATING: Wt of 1 cu ft free air at 60° = 0.076 l
pressed air at 75 lb and 60° = 0.456; heat r
vol of 1 lb free air at 60° = 123.84; heat requ
of 1 cu ft compressed air at 75 lb and 60° = 1

Comparing the cost of 1 cu ft of reheated ai
compressing 1 cu ft in a compressor cyl: 480.
that is, the theoretical cost in heat-units of
produced by reheating is about 12% of the
produced in a compressor. (Richards).

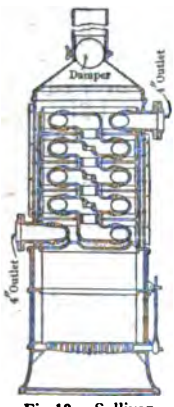

Fig 13. Sullivan Reheater

Table 44. Reheater De

Maker	Size no	Height, ft	Diam in
Sullivan.....	1	6.5	33
Sullivan.....	2	7.5	33
Sullivan.....	3	8.4	33
Sullivan.....	*	2.1	25 by
Ingersoll-Rand..	1	2.7	17
Ingersoll-Rand..	2	4.9	42

* Small portable hot-water h

Reheaters in general consist of a hollow shell, having inl
nections with the air main, and containing tubes, coils or d
bringing the compressed air in contact with the products
of combustion from a coal or coke fire below (3).

Actual results. Table 45 gives experiments by Riedler and Guttermuth (3) on consumption of reheated air by a single-cyl 80-hp Corliss engine. Initial press was 95.5 lb

Table 45. Efficiency of Reheating

Test	Temp of air		Cu ft free air used per hp hr	Indic hp	Effic, %
	Admission	Discharge			
1	264.2° F	69.8° F	462.77	72.3	89
2	305.6	84.4	431.09	72.3	90
3	320.0	95.0	418.55	72.3	91
4	338.0	120.2	432.12	65.0	87

Fig

abs, and temp of air in reheater did not exceed 338° F, at a
of 0.176 lb per hp-hr.

Reheating increases vol, rather than press, of the air. Usual
temp is from 250 to 340° F, giving practical increase of 20 to 3
general, 1 added hp is obtained by consumption of 0.3 to 0.4 lb
⅓ of fuel consumption of large compound compressors. Net sa
is from 15 to 20%. Incidentally, the use of heated air prevents
ing at the exhaust of pumps or other stationary engines. (See a

21. Compressed-air Hoists

Air hoists for mine service in shafts are dealt with in S

Air "lifts," for raising heavy weights through very small distanc
in connection with cranes in iron works and machine shops, are ma

nsists of a long-stroke cyl, suspended vertically from a crane or ed to lower end of piston rod. TELESCOPE TYPE, for low head- ng into each other. Oil or water, acted on by air press, may be , to prevent accidental lowering of the load through leakage of in the cyl by a stopcock, and passing into a reservoir for lowering.

plex hoists for small lifts, made by Chicago Pneumatic Tool tons; speed of lift, 28 to 4 ft per min; wt, 320 to 1 000 lb.

erial" hoist has a 3-cylinder "valveless" motor, driving a drum in dust-proof and oil-tight casing. There are 5 sizes; lifting nd 5 ton.

. Cost of Pneumatic Hoisting (Richards)
air: 5¢ for 1 000 cu ft free air at 100-lb gage)

Cu ft free air per 4-ft lift	Cost of air per 100 4-ft lifts, ¢	Diam of cyl, in	Effective piston area, sq in	Max wt lifted, lb	Cu ft free air per 4-ft lift	Cost of air per 100 4-ft lifts, ¢
1.67	0.84	8	48.87	4 398	11.88	5 94
2.97	1.49	9	61.85	5 566	15.03	7.52
4.64	2.32	10	76.36	6 872	18.56	9.28
6 68	3.34	11	92.39	8 315	22.46	11.23
9.09	4 55	12	109.96	9 896	26.73	13.37

handling timber in stopes, etc). INGERSOLL-RAND "LITTLE hoist with a 4-cyl, 2.5-hp motor; lifting capac, 0.5 ton; rope press, 80 lb. It fits a 4.5-in drill column or arm; weighs, with- olds 300 ft of 5/16 or 500 ft of 1/4-in rope. HOLMAN STRETCHER- d to a tunnel drill column. It has 1 or 2 cyl, can raise 500 to with 80-lb air press, and weighs 500, 650, and 1 000 lb, for 2, 4, $500. LEADVILLE COLUMN HOIST, similar to the above, has a o lb at 80 ft per min. Price, about $235.

de- om- ave arge ting the ving ines ssed cally

er re- rope, fric- 20%- skip con-

Table 47. Air Required to Hoist 1 Ton (2 240 lb) on Slopes (Ingersoll-Rand Co)

Angle of slope to horiz, deg	Cu ft free air per min of actual hoisting, @ 350 ft per min rope speed	Cu ft free air per 100 ft of hoist	Angle of slope to horiz, deg	Cu ft free air per min of actual hoisting, @ 350 ft per min rope speed	Cu ft free air per 100 ft of hoist
0	11	3.1	35	520	148
5	90	25.7	40	590	168
10	100	45.6	45	650	186
15	230	65.7	50	700	200
20	320	91.2	60	790	225
25	400	114	75	880	250
30	470	134	90	900	257

on a horiz mine track is 30 lb per ton and that there are 500 ft plain slide-valve, in good condition, and loaded to about full e air is throttled, more air is needed for same work.

ping by Compressed Air (See also Sec 13)

igned for steam are rarely well adapted to use of compressed air, expansion curves of steam and air; (*b*) the frequently improper the available air press (3, Chap 17, 24).

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Air consumption, based on head and vol of water and
HP = (lb water per min × height of lift in ft) ÷ 33 000.

Table 48. Static Water Potentials (Frank F

Gal	Volume, cu ft	Weight, lb	Potential hp in water raised 100 ft	Gal	Volume, cu ft
1	0.13368	8.355	0.02530	200	26.736
5	0.66840	41.775	0.12651	250	33.420
7.48	1	62.5	0.18928	300	40.104
10	1.3368	83.55	0.25303	400	53.472
20	2.6736	167.10	0.50606	500	66.840
25	3.342	208.87	0.63257	600	80.208
50	6.684	417.75	1.26515	700	93.576
75	10.026	626.62	1.89771	800	106.944
100	13.368	835.50	2.5303	900	120.312
150	20.052	1253	3.7954	1000	133.680

Volume of air for pumps working without expa
free air, cu ft per min; H = head under which pump work
raised per min; P = receiver air press, lb; V_2 = vol of fre
1 cu ft at press P; then $V = 0.093\ V_2\ (H \times G) \div P$. Th
100 ft piston speed, with 15% added volume of air to cov

Table 49. Values of V_2 and 0.093 V_2, Mean Press per Cu Ft Free Air

Air press P, lb	V_2	0.093 V_2	Mean air press per stroke	Hp per cu ft free air	Air press P, lb	V_2	o
25	2.70	0.2511	17.01	0.0743	60	5.08	0.
30	3.04	0.2827	19.40	0.0847	65	5.42	0.
35	3.38	0.3143	21.60	0.0943	70	5.76	0.
40	3.72	0.3459	23.66	0.1033	75	6.10	0.
45	4.06	0.3776	25.59	0.1117	80	6.44	0.
50	4.40	0.4092	27.39	0.1196	85	6.78	0.
55	4.74	0.4408	29.11	0.1270	90	7.12	0.

Example. — Required the vol of free air per min to raise 200
press being 80 lb. From Table 49, 0.093 V_2 corresponding to
$V = 0.5989\ (200 \times 150) \div 80 = 224.4$ cu ft free air. Hp devel
free air at 80 lb is $224.4 \times 0.1583 = 35.6$ hp. At 40 lb gage p
150) ÷ 40 = 259.0 cu ft free air, and $259.0 \times 0.1033 = 26.8$ hp
These examples show the advantage of using low-press air for

Since machine drills require air at high press, they ge
sign of the compressed-air plant. To get the best results, t
must be reduced below that in the mains. A receiver sho
the reducing valve and the pumps, to drain the air, lesse
ing, and to equalize the press (3).

Foot-gallons pumped per cu ft of free air range
efficiency is increased by stage compression, and by rehea

To use Table 50, divide diam of air cyl by diam of water cyl,
this ratio, and to the right of given head or lift, find the cu ft of
this, multiplied by total gal to be lifted, gives quantity of free
gage press is similarly found. Under pressures an allowance of
friction, and under volumes 15% for clearance losses and leak

of Air and Press Required for Direct-acting Pumps (1)

ss per sq in, lb				Cu ft free air per min to lift 1 gal water					
cylinder diams				Ratio of cylinder diams					
.5 to 1	2 to 1	2.5 to 1	3 to 1	1 to 1	1.25 to 1	1.5 to 1	2 to 1	2.5 to 1	3 to 1
12	7	....	....	0.44	0.53	0.65	0.94	....	....
14	8	....	....	0.49	0.58	0.70	0.99	....	....
16	9	....	....	0.54	0.63	0.75	1.03	....	....
18	11	....	....	0.61	0.68	0.79	1.11	....	....
21	12	....	....	0.66	0.75	0.87	1.15	....	....
23	13	....	....	0.72	0.82	0.91	1.20	....	....
29	16	10	....	0.86	0.95	1.06	1.33	1.67	....
35	20	13	9	1.00	1.08	1.20	1.50	1.88	2.31
46	26	17	12	1.28	1.37	1.47	1.75	2.14	2.60
58	33	21	15	....	1.64	1.75	2.06	2.41	2.89
68	39	25	17	....	1.92	2.00	2.31	2.68	3.08
80	45	29	20	....	....	2.28	2.57	2.95	3.37
92	52	33	23	....	....	2.57	2.87	3.22	3.66
05	58	37	26	....	....	2.88	3.13	3.48	3.95
...	65	42	29	....	....	....	3.42	3.82	4.24

educed in proportion to the ratio of abs temp. For compound a may be assumed at 75% of best results in the table.

pumps operate by displacement of a vol of water by an equal vol sponding to height of lift. Air is used without expansion. SINGLE-, *a*) are suited to low lifts. Compressed the chamber drives out the water, and e air escapes through the 3-way valve. e valve closes automatically, and water ough the bottom valve. The 3-way float. DOUBLE CHAMBER PUMPS (Fig operate alternately, the action of the constant. The Merrill and Latta- s type (3). AIR REQUIRED. The press lift. Press per sq in per foot of head height to which a given air press will s ÷ 0.434. 1 cu ft of water will be dis- at the required press, plus about 20% f these pumps is about 25%, which compares favorably with effic g pumps.

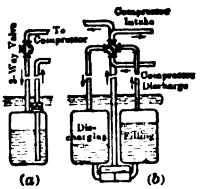

Fig 15. Direct Displacement Pumps

mption, Latta-Martin ap (Ingersoll-Rand)

Gage press, lb	Cu ft free air per gal water
80	0.861
90	0.952
100	1.043
110	1.134
120	1.225
.....	.....
.....	.....

Example. — Required to pump 75 gal per min against a vertical head of 60 ft, through 1 150 ft of 3-in pipe. Static press is 60 × 0.434 = 26 lb. Referring to tables, Art 16, 75 gal per min passing through 3-in pipe produce a friction loss of 0.74 lb per 100 ft of pipe; or, for 1 150 ft, 11.5 × 0.74 = 8.51 lb. Hence, working press at the pump = 26 + 8.5 = 34.5 lb. Interpolating in Table 51 between 30 and 40 lb press, 35 lb will require 0.452 cu ft free air per gal, or 33.9 cu ft free air for 75 gal. To estimate compressor capacity, add proper allowance for clearance and other losses (Art 1).

1 (3) may be used for displacement pumps, discharge air pipes from compr intake. Thus, same air is used repeatedly, with some deg 50%. Halsey pump is single-tank pump working on this princi-

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23. Air-lift Pump (See also Sec 13) (3, 29,

Principle. The delivery pipe is partly submerged in the
Compressed air is admitted to the foot of the pipe, and force
air and water.
pipe, the air
layers, betwee
in a large one,
a number of po
ing with water
The water is r
viva of compre
by aëration a
duction of sp

Submerge
surface of wa
pressed air is a
OF SUBMERGE
mergence to to
ery pipe; it m
low than for h
effic, range is a
lift (vert dista
water to point
for 500-ft lift;

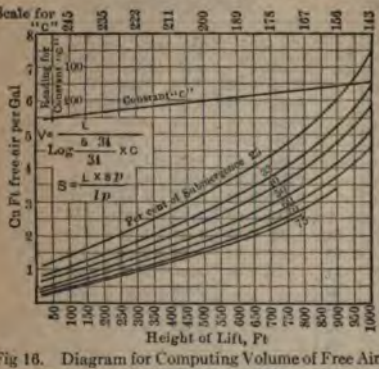
Scale for "C"
243 235 222 211 200 189 178 167 156 145
Reading for Constant "C"
100
200
Constant "C"
L
V = 5 34 / Log 34 × C
S = L × sp / lp
Pct cost of Submergence Air
Cu Ft free air per Gal
Height of Lift, Ft
50 100 150 200 250 300 350 400 450 500 550 600 650 700 750 800 850 900 950 1000

Fig 16. Diagram for Computing Volume of Free Air for Raising 1 Gal Water by Air-lift to Different Heights (Frank Richards) (Diag follows G. H. Richards' formula, except in taking constant *C* as variable, to agree with observed results)

Table 52. Approx Cu Ft Free Air and Working Press of Water by Air-lift (Sullivan Mach'y Co)

H ÷ *L* =	33% ½ to 1			43% ¾ to 1			50% 1 to 1			55% 1¼ to 1			60% 1½ to
Lift, ft	FA	WP	Hp	FA	WP	Hp	FA	WP	Hp	FA	WP	Hp	FA	WP
20	....	....	....	....	....	....	0.34	16	.018	0.29	18	.017	0.25	20
30	....	....	....	....	....	....	0.38	20	.024	0.32	24	.023	0.28	27
40	....	....	....	....	....	....	0.41	25	.031	0.35	29	.029	0.31	34
50	....	....	....	....	....	....	0.44	29	.036	0.38	35	.036	0.34	40
60	....	....	....	....	....	....	0.47	34	.043	0.41	40	.042	0.37	47
80	....	....	....	....	....	....	0.52	43	.056	0.46	52	.056	0.42	61
100	....	....	....	0.68	40	.070	0.58	52	.071	0.51	63	.062	0.47	74
120	....	....	....	0.73	47	.084	0.63	61	.075	0.56	74	.074	0.52	87
140	....	....	....	0.79	54	.099	0.68	70	.087	0.61	85	.087	0.56	101
160	....	....	....	0.84	61	.100	0.73	78	.099	0.66	96	.100	0.60	114
180	....	....	....	0.88	67	.110	0.77	88	.111	0.70	108	.112	0.65	127
200	1.15	52	.141	0.93	74	.123	0.82	96	.124	0.74	119	.121	0.69	141
250	1.27	63	.154	1.05	91	.154	0.92	119	.154	0.86	147	.159	0.79	174
300	1.40	74	.186	1.20	107	.191	1.03	141	.187	0.96	174	.192	0.89	208
350	1.50	85	.213	1.31	124	.224	1.16	163	.225	1.05	202	.229	0.98	241
400	1.63	96	.246	1.38	141	.250	1.23	186	.253	1.14	230	.238	1.08	273
450	1.74	107	.277	1.48	157	.282	1.33	208	.267	1.23	258	.269	1.17	308
500	1.86	119	.312	1.56	174	.312	1.43	230	.299	1.32	286	.299	1.26	342
550	1.96	130	.342	1.68	191	.349	1.52	253	.329	1.42	328	.338	1.34	375
600	2.05	141	.372	1.78	208	.358	1.61	275	.360	1.51	342	.365	1.42	409
650	2.18	152	.409	1.87	226	.388	1.74	297	.400	1.61	370	.400	1.52	442
700	2.27	163	.441	1.96	240	.416	1.81	320	.427	1.68	398	.428	1.60	476

Note. — *H* = submergence, ft; *L* = lift, ft; *V* = cu ft free air pe
ing air press, lb; *Hp* = horsepower; working power, ft lb = subm
7 lb; $V = L \div \log[(H + 34) \div 34] \times 292.5$; *FA* = free air, cu ft.

discharge is 20 to 25 ft per sec. At discharge, the air is Veloc of air in compressed-air pipe, 50 to 70 ft per sec.

n (30). When the cross-sec of the well is ay be placed alongside of the delivery pipe, n a nozzle directed vertically upward in the very pipe. This is the POHLÉ SIDE-INLET,

t with Pohlé Annular Foot-piece
(Ingersoll-Rand Co)

n Tail	Max econ capac mod-erate lift, gal per min	Foot-piece	
		Outside diam, in	Length, in
2	20	3 1/16	11
2½	35	4 3/32	12
3	60	5¼	12½
3½	100	5⅞	13
4	140	5 13/16	13
4½	190	7⅝	13½
5	225	7 9/16	13½
6	300	9¾	14

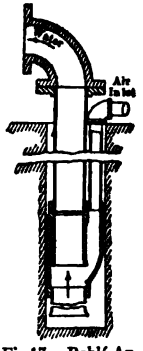

Fig 17. Pohlé Annular Foot-piece

r larger volumes of water, the POHLÉ ANNULAR FOOT-PIECE (Fig ng of small nozzles (Table 53).
ac" is based on 60% submergence, and a discharge of 12 gal discharge pipe for the smaller diam, and 15 gal for the larger. s the enlarged lower end of the delivery pipe. SAUNDERS AIR- d where the well will not admit placing 2 pipes side by side; the l air is admitted between the well casing and the delivery pipe. er min per sq in area of delivery pipe: for 25-ft lift, 15 to 20 gal; 25-ft lift, 12 to 15 gal. In the CENTRAL-AIR SYSTEM, giving max a given diam of well, the casing forms the delivery pipe. Proper e depend upon lift and submergence (Table 54). SULLIVAN FOOT-

Table 54. Central-air System (Saunders)

n	Size of air pipe, in	Capacity, gal per min	Size of casing, in	Size of air pipe, in	Capacity, gal per min
	1.25	80 to 100	6	2	275 to 375
	1.5	100 to 150	8	2.5	500 to 650
	2	150 to 250	10	2.5	775 to 1 000

18) is of bronze, with an air passage terminating in a perforated a contraction or "choke," for breaking air into small jets. This rough mixture of air and water, and velocity of flow is increased ke to produce an ejector effect. An opening below mixing tube nd or scale to drop through and not obstruct the perforations.

able 55. Sullivan Standard Air-lift

Diam well head, in	Approx shipping wt, lb	Diam disch, in	Approx capac, gal per min	Diam well head, in	Approx shipping wt, lb
4	85	4	200–300	8	185
6	125	4.5	300–350	8	200
6	135	5	350–500	10	250
8	150	6	500–800	10	300
8	175	7	800–1500	12	350

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In modifications of the preceding, air is admitted between casing an by a central air-pipe. They are made in sizes up to 15-in discharge 5 000 gal per min. In the SULLIVAN "BOOSTER," the discharged air and thu

Table 56. Tests of Air-lifts for Pumping Oil
(Evangeline, La) (30)

	Central Air-lift with Return Bend		Patented System
	Mannon	Evangeline	Evangeline
Mean i h p	99.1	122.56	89.19
Mean water hp	9.85	9.97	10.36
Mean air hp	82.5	107.38	79.16
Gal of fluid, per hr	2 499.6	1 953.6	2 188.2
Wt of 1 gal of fluid, lb	8.72	8.7	8.69
Oil in fluid, %	11.1	10.5	11.4
Sp gr of oil	0.9	0.9	0.9
Total depth of well, ft	1 901.0	1 805.0	1 805.0
Diam casing, in	6.0	6.0	6.0
Height above ground pumped, ft	3.33	18.5	18.5
Size of disch pipe, in	4.0	4.0	4.0
Length vert disch, ft	1 500.0	1 513.5	1 513.5
Length vert air pipe, ft	1 489.5	1 513.5	1 493.0
Size of compressors, in	7.5 by 18 by 16 by 16	10 by 22 by 16 by 20	7.5 by 18 by 16 by 20
Fuel (crude oil), gal per hr	44.22	48.36	35.27
Cost of fuel, ¢ per gal	2.03	2.14	2.14
Cost of fuel per bbl oil, ¢	12.8	21.2	12.6

Other applications. The air-lift is not limited to raising water. such work as pumping petroleum wells, and slimes, sands and solution, also for returning cooling water to elevated tanks, where the same wate

Table 57. Cost of Pumping with Air-lift (compiled from dif

Output, gal per min	Lift, ft	Cost of coal	Cost per 1 000 gal, ¢	Output, gal per min	Lift, ft	C
2 775	50		0.33	175	75	
2 775+	35	$2	0.15	695	50	
925	75	2	1.00	10 400	30	
2 000	75		4.50	100	60	
2 000	50		3.50			

Estimate for an air-lift plant to raise 1 250 gal per min, 75 ft. Su pressor and receiver; 85-hp boiler with accessories; 19 000-gal tank, ir framework to bring it 75 ft above water level; 2 12-in, 135-ft cased we light casing; 500 ft of 3-in air pipe in wells; 1 000 ft of 4-in air line fro 1 250 ft of 12, 10 and 8-in C-I water pipe from tank to works, laid cost, complete and ready to run, $8 750. OPERATING COST per day: @ $2.25, ½ time chargeable to pumping plant, $0.90; fireman, 2 shift of 1 man for each 250 hp of boiler) for 85 hp, $1.19; coal for 85 hp, @ $2, $8.50; oil and sundries, $0.60; int on $8 750 @ 6%, figuring o per year, $1.91; deprec, repairs, and providing for renewal of plant ev $3.18; insurance and taxes, 1%, $0.32; total daily cost, $16.60; cost + 1 500), 1.10¢.

Examples. I. Mining plant, Sinaloa, Mexico. Centrifugal pu 1910, wore rapidly due to gritty water, and were replaced, Oct, 1910,

65% submergence, 3.5-in well-pipe; (*b*) 3.5-in delivery, 38-ft well. Air press, 25 to 28 lb. Total cost, including 5-ft shaft and launders, but omitting the compressor, was less than nps used 25 hp per 24 hr; maintenance, $200 per mo; were The air-lifts have cost little for repairs, have worked steadily

ry Mines, S Africa. At Angelo and Cason plants are large- mes having sp gr of 1.1013. Lift, 27 to 32.5 ft; submersion, air press, 15 to 22 lb; free air per min, 796.5 to 2 256 cu ft; effic, 17.7 to 36.15 (3).
ter supply. 6 wells, 500 ft deep; one, 13-in diam, with 3-in a, with 2.5-in air line (Table 58).

Air-lifts, Memphis, Tenn (*Comp Air Mag*, Sep, 1909)

Sub-mergence, ft	Working press, lb	Draft of well, ft	Total lift, ft	Gal per min	Free air per min, cu ft	Free air per gal, cu ft
65.2	60.0	18.5	73.9	974	336	0.345
63.2	57.5	13.9	77.2	883	332	0.376
63.4	58.5	15 0	78.1	603	347	0.576
66.1	57.5	17.3	68.9	701	307	0.437
.....	73.0	41.6	120.6	1 410	514	0.364

ba, 1910. Water stood at 200-ft level in an 800-ft shaft, and was umps. 3 air-lift pumps were installed, having 10-in discharge 2 were used at a time, one being held in reserve. At start, submergence; at finish, head was 369 ft, with 39% submergence. r min per pipe; air consumption, 105 cu ft free air per 100 gal. oved @ 7.434 ¢ per 1 000 gal. Below 400-ft level a 2 000-gal l, and the discharge of the air-lifts turned into its sump. Cost 7 days by plunger and air-lift pumps was 2.32¢ per 1 000 gal. lunger pumps, 8.3¢ per 1 000 gal (*S of M Quart*, July, 1911).

Reciprocating Rock Drills (3, 39)

on. The bit is clamped in a chuck, attached to the piston mpressed air or steam (for electric drills, see Sec 16). Pis- d automatically by rifle-bar, ratchet and pawls. Cyl is fed forward by a hand crank and feed screw. Entire a tripod, or on a vertical or horiz bar (Art 27).

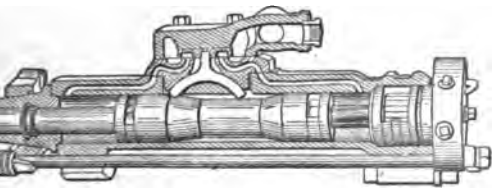

Fig 19. Sullivan Tappet Drill

t based on the type of valve motion: (*a*) TAPPET DRILLS, lve is thrown positively by a 3-armed tappet (Fig 19), h are caused directly by the reciprocations of the piston. (Fig 20) an arc-shaped tappet valve controls auxiliary use movement of the main valve; (*b*) AIR-VALVE DRILLS, vements control those of the valve by a system of auxiliary

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ports between cyl and chest. This valve motion resembles s
single-cyl pumps. (For details of valves, see Art 26.)

Air at full press is used by most drills. Expansive-working dri
ful; either the drilling speed is low (due to lighter blow) and ports a
ing, or repair cost is high. Saving in compressed air is insufficient t

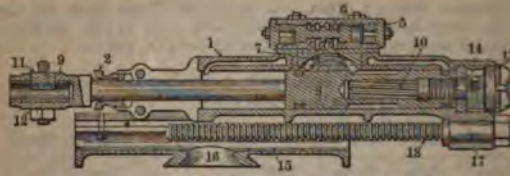


Fig 20. Sergeant Drill (Ingersoll-Rand)

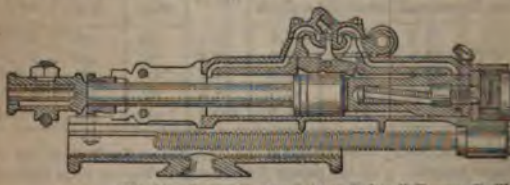


Fig 21. Ingersoll-Rand "Arc Valve," Tappet Drill

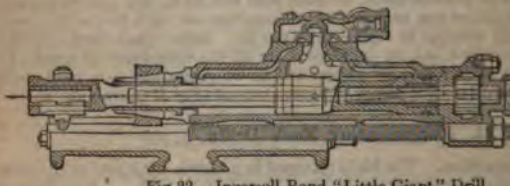


Fig 22. Ingersoll-Rand "Little Giant" Drill

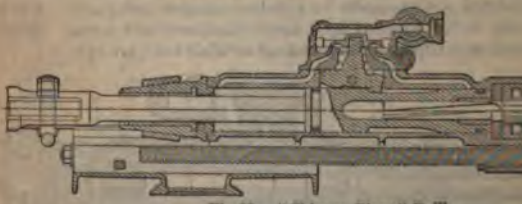


Fig 23. "Chicago Giant" Drill

Tappet-valve drills include: Ingersoll-Rand "Arc Valve
"Little Giant" (Fig 22); Chicago "Giant" (Fig 23) and "S
of the Chicago Pneumatic Tool Co; Sullivan "Tappet-valve
man; Stephens "Climax;" Rio Tinto; "Little Murphy" (Fig

The principal features of tappet drills are indicated in the cuts (s
ADVANTAGES: in dusty mines they generally wear better than air-thro
affected by moist compressed air, and are less apt to stick or "fitcher
ground, as the air caught by the valve reversal between piston and
return stroke. But, for this reason, the tappet drill does not deliver

the Rio Tinto, most tappet drills have B slide valves. In the pet and valve are combined.

pical drill (Fig 20): 1, cylinder; 2, front cyl head and stuffing box l head; 4, one of 2 side-rods, holding cyl heads in place; 5, valve rc tappet valve, oscillated by shoulders on the piston 8; 9, chuck;

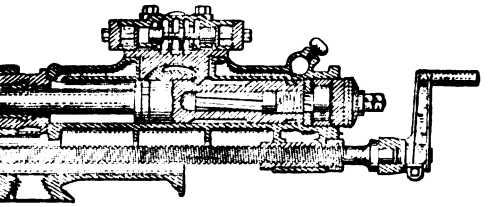
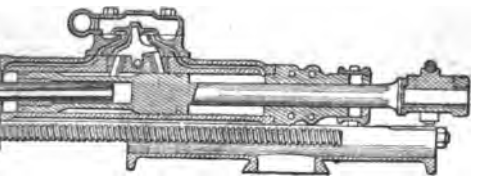


Fig 24. "Chicago Slogger" Drill



ig 25. Murphy "Little Champion Drill"

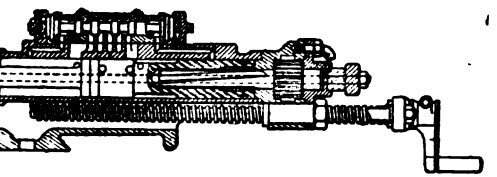
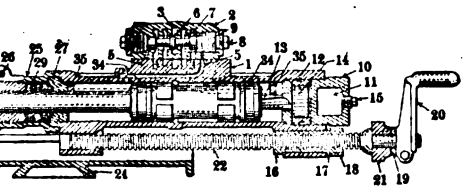


Fig 26. Ingersoll-Rand "Slugger"



ig 27. Sullivan Differential-valve Drill

piston; 11, 12, cottar and U-bolt for clamping shank of bit; 13, 2 h teeth of ratchet wheel; 14, ratchet wheel, preventing rifle-bar stroke, but allowing it to turn on back stroke; 15, guide shell; astened to mounting; 17, feed nut; 18, 19, feed screw and crank.

lude: Ingersoll-Rand "Eclipse," and "Slugger" (Fig 26); "Differential-valve" (Fig 27) and "Liteweight" (Fig 28);

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Stephens "Climax Imperial"; Wood (Fig 29); McKiernan (Fig 30); "Little Hercules"; Hardy; Ingersoll-Rand "New terfly-valve (Fig 42);" "Chicago Gatling" (Fig 31). (Fo of a number of air-valve drills, see Bib 3, 39.) ADVAN parts, due to elimination of the tappet, and they stand

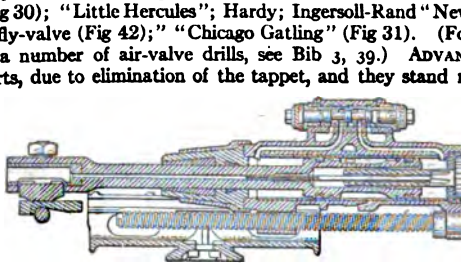


Fig 28. Sullivan "Liteweight" Drill

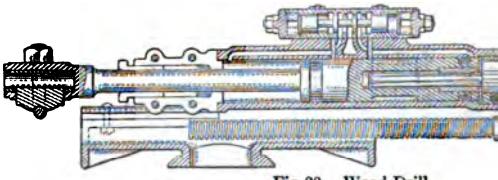


Fig 29. Wood Drill

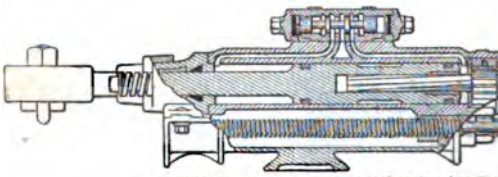


Fig 30. McKiernan-Terry Spool-valve Dr

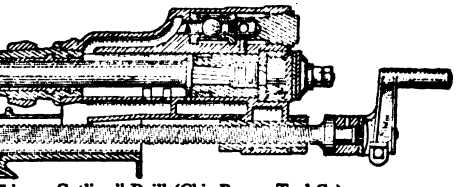
better. Valves are usually of the spool or piston type. of the slide-valve tappet drills is thus avoided until the machine becomes less efficient through irregular valve ac

Essential parts of a typical air-valve drill (Fig 27):

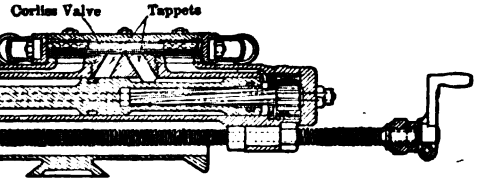
1. Cylinder.	10. Top head.	19. Washer.
2. Steam chest.	11. Oil chamber.	20. Feed handle.
3. Inlet port.	12. Ratchet ring.	21. Yoke.
4. Exhaust ports.	13. Rifle bar.	22. Feed screw.
5. Reverse ports.	14. Ratchet.	23. Shell.
6. Valve.	15. Plug.	24. Trunnion.
7. Valve bushing.	16. Feed nut.	25. Lower head.
8. Buffer.	17. Lock washer.	26. Clamp bolt.
9. Check nut.	18. Check nut.	27. Bushing.

Special features of air-valve drills. SULLIVAN "LITEWEIGH rock and high air press. It may have hollow piston rod and bit, of hole compressed air to clean out sludge. INGERSOLL-RAND work; should be run by air only. As the valve is thrown by stroke lengthens as valve wears. MCKIERNAN-TERRY S drill (uniform rock; works best with air, but can be run with dry steam has a ball valve, which throws promptly and wears well; well INGERSOLL-RAND "BUTTERFLY-VALVE" drill has a valve of smal with large exhaust ports. "Pull-back" is quick and strong, pro

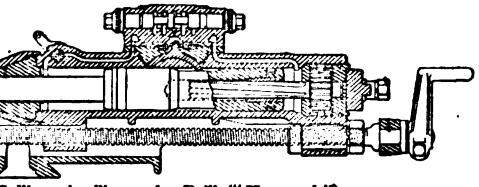
drills are for deep holes, as in subaqueous work. McKIER-32) has Corliss valve; strikes a heavy blow, and its strong it or fissured rock, likely to "fitcher."



hicago Gatling" Drill (Chic Pneum Tool Co)



2. McKiernan-Terry Corliss-valve Drill



Sullivan Auxiliary-valve Drill ("Hy-speed")

ve 2 valves; one, moved directly by the piston reciprocations, xhaust of air which actuates the other. Examples: Ingersoll-IPROVED SERGEANT"; "CHICAGO SLOGGER"; HOLMAN spool-d" (Fig 33) and FZ–31 (7-in). The Holman main valve is noved positively by the piston.

atures noted above are from makers' catalogues, as it is prac-unbiased criticism from users.)

o 62 give a comparative view of types and sizes of well-known rying drills, omitting large sizes used for subaqueous work. the tables indicate the following items:

	Unit			Unit
.....		13	Boiler for steam supply of 1 drill	Hp
.....		14	Diam of pipe to carry steam 100–200 ft	In
.....	In	15	Weight of drill unmounted, with fittings, unboxed	Lb
.....	In			
) 80 lb....	Cu ft	16	Same, boxed	Lb
.....	In	17	Wt of tripod without weights	Lb
.....	In	18	Wt of holding-down weights	Lb
illed easily	Ft	19	Wt complete, with fittings	Lb
bove depth	No	20	Price unmounted, with fittings	$
n.....	In	21	Price complete, with tripod and weights	$
.....	In			

x, being subject in some cases to discounts of 10 to 25%; they different sizes of same drill, but not drills of different makes.

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Table 59. Tappet-valve Drills

1	Ingersoll-Rand Co												Sullivan Machinery Co		
2	"Arc valve"						"Little Giant"						Tappet valve		
3	A 50	B 32	C 32	D 32	E 32	F 32	o D	1	2 D	3 D	3½ D	4 D	5 D	UC	UE 2	UF 3
4	2¼	2½	2¾	3⅛	3¼	3⅜	2	2¼	2¾	3⅛	3¼	3⅜	4¼	2¾	3⅛	3¼
5	5	5⅜	6⅛	6⅜	6⅞	7¼	5¼	5⅜	6⅛	6¼	6¾	7½	8	6⅛	6½	6⅜
6	76	86	104	120	127	143	63	76	104	120	127	143	164	108	119	127
7	⅞	1⅞	1⅛–1	1¼–1⅛	1¼–1⅛	1⅜–1¼	¾–⅞	¾–⅞	1–1⅛	1⅛–1¼	1⅛–1¼	1⅜	1½	1–1⅛	1⅛–1¼	1⅛–1¼
8	15	20	24	24	24	24	15	18	18–24	24	24	30	36	24	24	24
9	6	8	10	14	16	20	5	6	10	14	16	20	30	8	14	16
10	5	5	5	7	8	10	4	4	7–8	7	8	8	10	5	7	8
11	⅝–1¼	1–1½	1¼–2¼	1½–2¼	1¾–2¾	1¾–3	1	1 1/16	1½	1¾	1¾	2	2¼	1¼–2½	1¼–3	1¼–3
12	¾	¾	1	1	1	1	¾	¾	¾	1	1	1	1¼	¾	1	1
13	5	8	8	8	10	12	5	6	8	8	10	12	15	8	10	10
14	¾	1	1	1	1	1¼	¾	1	1¼	1½	1½	2	2	1	1¼	1¼
15	145	185	270	285	295	415	118	176	232	321	329	459	704	160	260	295
16	185	230	320	335	345	465	152	209	273	371	379	520	792	220	305	340
17	85	165	165	165	210	275	80	90	150	181	181	333	437	.....	.....	.....
18	120	270	285	285	330	375	116	228	331	368	368	426	600	.....	.....	.....
19	445	745	885	870	970	1 225	348	527	754	920	928	1279	1829	.....	.....	.....
20	195	225	250	275	300	320	170	195	250	275	300	320	375	.....	.....	.....
21	225	275	300	325	350	375	200	225	300	325	350	375	440		.....	.....

1, 2	Chicago Pneum Tool Co "Chicago Giant"					Wickes Bros "Murphy Little Champion"					Hardsocg Wonder Drill Co "Wonder"						
3	B	C	D	E	F	A	B	C	D	E	F	G	H	S	U	V	W
4	2¼	2¾	3⅛	3¼	3⅜	2¼	2½	2¾	3	3⅛	3¼	3½	3¾	2¼	2¾	3⅛	3¼
5	5¼	6¼	6¾	6¾	7½	5½	6	6½	6½	6¾	7	7½	8	5¼	6¼	6¼	6¾
6	.....	102	.....	130	141	80	90	100	115	120	130	140	145	.....	.....	.....	.....
7	⅞	1	1⅛–1¼	1⅛–1¼	1⅜	¾–⅞	⅞–1	1–1⅛	1⅛–1¼	1⅛–1¼	1⅛–1⅜	1¼–1⅜	1¼–1⅜	⅞	1	1⅛	1¼
8	20	26	26	26	32	15	20	24	24	24	24	24	30	20	26	26	26
9	9	14	18	20	25	5	6	10	12	14	16	20	25	8	14	18	20

14	¾	1	1	1	1	1¼	1¼	1¼	1½	1½	1	1¼	1½	1	1	1	1¼	1½
15	110	145	165	160	240	265	260	295	365	560	162	295	325	210	250	295	420	675
16	145	180	200	220	280	310	305	340	440	775	227	340	400	250	295	340	780	890
17	.....	.....	....	.....	.....	.....	.....	.....	.....	.....	215	240	370	160	160	210	210	300
18	.....	.....	....	.....	.....	.....	.....	.....	.....	.....	305	310	400	270	270	330	330	375
19	.....	.....	....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	760	805	965	1 105	1 710
20	.....	.....	....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	225	250	275	320	365
21	.....	.....	....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	275	300	325	375	430

1	Ingersoll-Rand Co					Wood Drill Works							McKiernan-Terry Drill Co						Chic Pneum Tool Co	Ingersoll-Rand Co
2	"Slugger," 13 Type and 33 Type				"Ing"	Wood Drill							"Spool-Valve"						"Chic Gat'g"	"Butterfly"
3	12	13	14	33	34	H 9	...	....	....	....	.....	.....	.....	S	S	S	S	S	S	C 22	C 110
4	2¾	3¼	3⅝	3⅛	3⅝	5½	2	2¼	2½	2¾	3	3¼	3⅜	2¼	2½	2¾	3	3¼	3⅜	2¾	2¾
5	6¼	6½	7¼	6½	7⅜	8	4½	4½	5	5½	6½	7	7½	4¼	5	5½	6½	7	7½	6¼	6¼
6	104	127	143	120	143	.....	63	79	85	107	114	126	147	76	86	104	114	127	143	102	104
7	1–1⅛	1⅛–1¼	1–⅜	1⅛–1¼	1–⅜	1–⅜	⅜	1	1	1⅛	1⅛	1⅛–1½	1¼–1⅜	¾–⅞	⅞–1	1–1⅛	1–⅛	1¼–1⅜	1¼–1⅜	1	⅞–1⅛
8	18	24	30	24	30	30	12	20	20	24	24	24	24–30	12	20	24	24	24	24–30	20–24	24
9	10	16	20	14	20	32	4	6–8	10	10–12	10–14	14	20–25	6	10	12	14	16	25	8	8
10	7	8	8	7	8	13	....	.....	.....	.....	.....	.....	.....	6	6	6	7	8	10	5	4
11	1½	1¾	2	1¾	2	3–6	.....	.....	.....	.....	.....	.....	.....	¾–1½	1¼–2	1¼–2	1½–2¼	1¾–3	1¾–3	1¼–1¾	1–1½
12	¾	1	1¼	1	1¼	1¼	½	½–¾	¾	¾–1	1	1	1	½	¾	1	1	1	1	¾	¾
13	....	.....	....	.....	....	18	5	6	8	8	8	10	12	5	8	8	8	10	10	.....	.....
14	....	.....	....	.....	....	1½	.....	.....	.....	.....	.....	.....	.....	95	170	210	255	290	407	145	137
15	236	340	424	288	416	930	85	130	165	205	265	295	390	135	205	270	310	350	475	200	180
16	277	396	485	338	477	1 170	120	165	205	240	310	340	445	81	162	162	223	223	334	200	174
17	150	181	333	181	333	300	68	84	170	170	170	220	300	171	219	219	309	309	432	300	326
18	309	345	399	345	399	375	150	159	195	225	225	300	450	.....	.....	.....	.....	.....	.....	.....	.....
19	758	939	1244	887	1236	1 990	.....	.....	.....	.....	.....	.....	.....	210	240	250	275	300	320	220	220
20	250	300	320	275	320	700	125	135	140	145	155	170	185	255	300	310	347	372	404	275	.....
21	300	350	375	325	375	765	.....	.....	.....	.....	.....	.....	.....	55	55	65	65	65	80	32	55

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Table 61. Corliss-valve Drills

1	McKiernan-Terry Drill Co				1	McKier
2	Corliss valve				2	
3	TC	TC	TC	TC	13	8
4	2¾	3	3¼	3⅜	14	1
5	6½	6½	6½	8¾	15	230
6	104	114	127	143	16	285
7	1–1⅛	1⅛–1¼	1¼–1⅜	1¼–1⅜	17	162
8	24	24	24	24–30	18	219
9	12	14	16	25	19	.....
10	6	7	8	10	20	250
11	1½–2	1½–2¼	1¾–3	1¾–3	21	310
12	1	1	1	1	....	.....

Table 62. Auxiliary-valve Drills

1	Ingersoll-Rand Co						
2	"Sergeant"						
3	A 86	B 24	C 24	D 24	E 24	F 24	F 94
4	2¼	2½	2¾	3	3¼	3½	3⅜
5	5	6	6½	6½	6½	7	7½
6	76	86	104	114	127	131	143
7	⅜–¾	⅜–1	1–1⅛	1⅛–1¼	1⅛–1¼	1¼–1⅜	1¼–1¾
8	15	20	24	24	24	24	24
9	6	8	10	14	16	20	20
10	5	5	5	7	8	10	10
11	¾–1½	1–1½	1¼–2¼	1¼–2¼	1¾–2¾	1¾–3	1¾–3
12	¾	1	1	1	1	1	1
13	6	8	8	8	10	10	10
14	¾	1	1	1	1	1¼	1¼
15	140	190	270	280	295	405	410
16	170	235	320	330	345	465	470
17	85	165	165	165	210	275	275
18	120	270	285	285	330	375	375
19	430	750	855	865	970	1 225	1 230
20	170	200	220	240	260	280	290
21	.....	.....	.....	.....	.....	.....	.....

1	Chicago Pneumatic Tool Co				Sullivan M
2	"Slogger"				"Hy-spee
3	C 2	D 2	E 2	F 2	PG 3	PJ 3	I
4	2¾	3	3¼	3⅜	2¾	3	
5	6½	6½	6½	7½	5½	6	
6	102	114	130	141	.....	.....	
7	1⅛–1¼	1⅛–1¼	1⅛–1¼	1¼–1⅜	1⅛–1¼	1⅛–1¼	1⅛
8	24	24	24	24	24	26	
9	10	14	16	20	14	17·3	
10	5	7	8	10	7	8	
11	2–2½	1¾–2½	1⅜–2½	1½–2⅜	.....	.....	
12	1	1	1	1	1	1	
13	8	8	10	10	8	8	
14	1	1	1	1¼	1	1¼	
15	285	295	330	400	250	275	
16	365	380	407	483	290	320	
17	233	233	233	376	215	215	
18	312	312	312	405	305	305	
19	1 042	1 084	1 084	1 375	.....	.....	
20	220	240	260	290	.....	.....	
21	285	305	325	365	220	240	

e Air per Min to Run One Drill (Ingersoll-Rand Co)

Diam drill cylinder, in									
2¾	3	3⅛	3 3/16	3¼	3½	3⅝	4¼	5	5½
82	90	95	97	100	108	113	130	150	164
93	102	108	110	113	124	129	147	170	181
104	114	120	123	127	131	143	164	190	207
115	126	133	136	141	152	159	182	210	230
126	138	146	149	154	166	174	199	240	252

rs to Find Compressor Capacity for 1 to 50 Drills at Different Altitudes

Number of drills												
5	6	7	8	9	10	12	15	20	25	30	40	50
Multipliers												
4.8	5.4	6.0	6.5	7.1	8.1	9.5	11.7	13.7	15.8	21.4	25.5	
4.94	5.56	6.18	6.67	7.3	8.34	9.78	12.05	14.1	16.3	22.0	26.26	
5.14	5.78	6.42	6.95	7.60	8.67	10.17	12.52	14.66	16.9	22.9	27.28	
5.28	5.94	6.6	7.15	7.81	8.91	10.45	12.87	15.07	17.38	23.54	28.05	
5.47	6.15	6.84	7.41	8.09	9.23	10.83	13.34	15.62	18.01	24.4	29.07	
5.62	6.32	7.02	7.61	8.31	9.48	11.12	13.69	16.03	18.49	25.04	29.84	
5.76	6.48	7.2	7.8	8.52	9.72	11.4	14.04	16.44	18.96	25.68	30.6	
5.9	6.64	7.38	7.99	8.73	9.96	11.68	14.39	16.85	19.43	26.32	31.36	
6.05	6.8	7.56	8.19	8.95	10.21	11.97	14.74	17.26	19.9	29.96	32.13	
6.19	6.96	7.74	8.38	9.16	10.45	12.26	15.09	17.67	20.38	27.6	32.9	
6.34	7.13	7.92	8.58	9.37	10.69	12.54	15.44	18.08	20.86	28.25	33.66	
6.57	7.4	8.22	8.9	9.73	11.1	13.02	16.03	18.77	21.62	29.32	34.94	
6.85	7.72	8.58	9.3	10.15	11.58	13.58	16.73	19.59	22.59	30.6	36.46	

free air to operate 30 5-in drills at 9 000 ft altitude, with air 3, one 5-in drill at sea level requires 190 cu ft per min. From rills at 9 000 ft is 20.38; multiplying 190 cu ft by 20.38 gives , which is the compressor displacement for aver conditions, n and leakage losses.

Electric-air" drill operates by pulsations of compressed ven by electric or other motor. The air is not exhausted,

Table 65. Electric-air Drill

Catalogue number.....	5-F	4-E	3-F
Cylinder diam, in.....	5⅝	4¾	3⅝
Stroke, in.....	8	7	6¾
Length of feed, in.....	30	24	20
Depth of holes drilled, ft....	20	12	6
Diam of holes, in.....	1.75 to 2.75	1.25 to 1.5	1.5 to 1.75
Hp required at motor.....	6	4	3
Wt of drill unmounted, lb...	410	228	125
Wt of pulsator mounted, lb..	1 050	950	600
Total wt complete, lb.....	1 755	1 690	925
Price.....	$1 050	$1 000	$ 750

of hose, each acting alternately as supply and exhaust. The ry rock drills.

st, with electric-air drills, model 5-F, at Kensico, N Y, dam a). Rock, gneiss, mainly hard and solid. One 8-hr shift per usually, about 25 in. Full speed, 400 strokes per min. Crew, 1 to 1.5 in. Bits: starter, 3.5 to 4 in, decreasing by ½ in to 1.5 in.

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Spacing of holes: at quarry, 15 by 12 ft or 20 by 20 ft; at dam, irreg
60%, averaging 0.5 lb per cu yd. Blacksmith coal: 0.25 ton per day
per shift. Drilling speed: at quarry, 0.135 ft per min; at dam, 0.1

Table 66. Drilling Time, Electric-air Drills (Constru

	Quarry				A
	Aver time, min sec	Total time, min sec	% of total		ti
mi
Actual drilling	14 18	228 54	51.1		11
Raising drill	1 05	16 13	3.6		0
Loosening chuck (*a*)	0 11	1 16	0.3		0
Loosening chuck (*b*)	1 27	(4 20)(*c*)			
Removing bit	0 32	6 26	1.4		0
Bailing hole	1 23	15 10	3.4		1
Bit in hole	0 22	4 20	1.0		0
Bit in chuck	0 23	6 12	1.4		0
Tightening chuck (*a*)	0 10	1 00	0.2		0
Tightening chuck (*b*)	0 53	8 46	2.0		
Getting started	1 01	17 13	3.8		0
Cycle totals	21 45	305 30	68.2		16
Shifting drill	52 00	73 18	16.4		12
Miscel delays	6 17	69 05	15.4		2
Totals	.....	447 53	100.0		...
Ft drilled	.....	.....	31.0		...
Aver depth of holes	.....	.....	22.0		...
Drilling speed, ft per min	.....	.....	0.135		...
Ratio drilling time to total	.....	.....	0.511		...

(*a*) Sleeve chuck. (*b*) Bolted chuck. (*c*) Not included in cy
formed by 1 man, while another raised the drill. (*d*) Percentage
at dam than at quarry because of prompt delivery of steels. In a

Table 67. Cost of "Electric-air" Drilling at Kensico Quarry

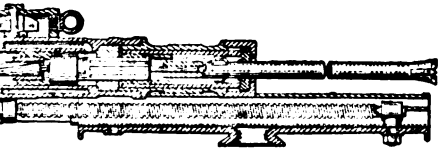
	Rate	Amount, $	Cost per lin ft, ¢	Cost per cu yd, ¢
6 drillers	$2.50	15.00		
6 helpers	1.75	10.50		
		25.50	12.75	1.23
1.5 blacksmith	3.00	4.50		
1.5 helpers	1.75	2.63		
2 nippers	1.50	3.00		
2 mules	1.50	3.00		
		13.13	6.57	0.64
Coal, 500 lb	3.50	0.87		
Oil, 3 qt per drill	0.30	1.35		
Power, 35 kw-hr	0.01	2.10		
		4.32	2.16	0.21
Total drilling cost		42.95	21.48	2.08
Int and deprec, 2% per mo		7.70	3.85	0.37
3 powdermen	2.00	6.00		
1 035 lb dynamite	0.12	124.20		
25 detonators	0.03	0.75		
Totals		181.60	90.80	8.77

Note. — Contractor's overhead charges, superintendence, storage costs, insurance, etc., are not included.

25. Hammer Drills (3, 39)

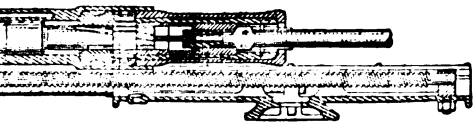
n. The bit does not reciprocate. Its shank projects
n, the forward end of cylinder, and is struck by a ham-
vil block may be placed between piston and bit shank.
ct with the bottom of the hole, except during the slight
ow of the hammer. (For drill details, see Art 26.)

e drills are held in position by a D-handle, or are mounted on
air-feed telescopic standard; in the latter case, they are used



4. "Leyner-Ingersoll" Hammer Drill

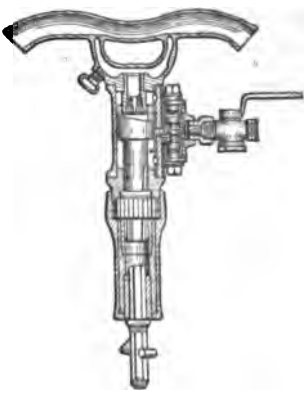
nown as "stopers." In some machines the air-feed and D-
as in the Whitcomb, and McKiernan-Terry A 345 rotating



35. Sullivan Mounted Hammer Drill

in any of the 3 forms in either the valveless or valve types.
is automatic.

drills, in which the differential piston acts also as a valve, in-
rial," Waugh "Valve-
"Little Wonder."
ersoll-Rand "Leyner-
hamer" ("butterfly"
). "Little Jap" (axial
ounted drill (Fig 35),
-feed" stoper (Fig 37);
mmer" (self-rotating);
iernan-Terry self-rota-
nd Stephens "Climax



is provided to receive
blow in the Ingersoll-
rial" and "Crown"
Chicago, Waugh and

s. Table 68 gives
hand hammer drills
tunneling, trimming
are well adapted to
lug and block holes.

Fig 36. Sullivan "Rotator," DP-33

andle drills are used chiefly in sinking, for which service, on
d rapid drilling, they often compare favorably with recipro-
es: INGERSOLL-RAND "JACKHAMER" uses the "Butterf-

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Table 68. Unmounted Machine Hammer Drills

Maker	Sullivan Machinery Co							Ingersoll-Rand Co							McKiernan-Terry Drill Co			
Catalogue name	Plug drill	Ft-hole drill	Hand hammer		Sinker	Hitch cutter	Rotator	Plug drill	"Imperial"		"Butterfly"		"Jackhamer"	"Bullmoose"	"Busy Bee"		Non-rotator	Sinker
Model	DF 3 (DA 15)	DK 3 (DA 19)	DF 31 (DB 19)	DC 19	DB 21	DB 13	DP 33	HA 14	MV 1	MV 2	BA	BC	BCR 430	BCR 53	C 3	B 1	B 4	A 3
Use	plug & feather	foot holes	block holes	block holes	shaft sinking	cutting hitches	general drilling	plug & feather	plug & feather		block holes		sinking	sinking	plug holes	sinking	sinking	sinking
Length, in	17.5	21.5	19	21.5	....	16	....	17.5	15	15	22.5	24	18	28	17	17	.....	27
Weight, lb	24	41	27	44	60	14.5	38	21	22	30	43–48	54–65	40	105	28	50	75	92
Cyl diam, in	1 9/16	1 3/4	1 9/16	1 3/4	2	1 5/16	2 1/4	1 3/8	1 3/4	1 11/16	1 1/2	2	2 1/4	2	1 11/16	1 3/8	1 11/16	2
Stroke, in	2 1/4	2 1/2	2 1/4	2 1/2	3 1/2	2 1/8	....	2 1/2	2 1/4	2 1/4	3 1/2	4	2	4	4	3	5	3 5/8
Diam of steel, in	1 1/4	2 7/32	7/8	1	1	.....	7/8	7/8	7/8	7/8	7/8–1	1	7/8	1	7/8	7/8	7/8–1	1
Kind of steel	solid	solid	hollow	hollow	hollow	solid	hollow	solid	hollow	hollow	hollow	hollow	hollow	hollow	hollow	hollow	hollow	hollow
Size shank, in	1 1/16×2 1/2	1 3/16×3 1/2	7/8×3	....	....	....	....	1 1/8×2 1/2	....	....	none	none	.....	none	taper	taper	taper	taper
Depth hole, ft	0.5	1	4	6	8	....	10	0.5	4	4	12	12	12	12	4	8	8–12	12
Diam hole, in	7/8–1	5/8–1	1 1/2–2	2	2	special	1 1/4	7/8–1 3/4	1–1 1/2	1–1 1/2	1 1/8	1 13/16	2	2	1 1/4	1 1/4	1 1/4	1 1/4
Free air, min, cu ft	29	41	30	38	59	13	50	25	23	23	57	74	69	85	50	60	70	75
Air press, lb	90	90	90	90	90	90	90	80	90	90	80	80	90	90	80–90	80–90	80–90	80
Rotation	hand	hand	hand	hand	hand	....	auto	hand	hand	hand	hand	hand	auto	auto	hand	hand	hand	auto
Approx price, $	55	75	65	85	100	45	100	50	60	60	75	100	100	200	50	75	100	130

Maker	Trow & Holden	G. Oldham & S	Kotten Co		Cleveland Rock Drill Co				Chicago Pneumatic Tool Co			Whitc'b Co	Dallett Co		Hardsocg Wonder Drill Co				Den R'k Dr Mf Co
Catalogue name	Plug drill	Plug drill	Plug drill	Hand rock	Hand sinkers				Plug & feather	V'vless h'd t'l	Sinker	H'm'r drill	Plug drill	Hand rock	Sinker	Sinker	Plug drill	Sinker	Waugh plugger
Model	.....	A and B			25 H	50 H	30 H	HR 10	No 5	K 1 3/4	B 46	....	Y'kee B	....	90	65 A	38	44	22 P
Use	plug holes	plug & feather	plug & feather	sinking	block holes & general drilling				plug & feather	block holes	sinking	block holes	plug & feather	block holes	sinking	sinking	plug & feather	sinking	block holes
Length, in	17.5	20.5	18	17	22	22	23	28	17	15	24	....	18 1/2	19 1/2	21	20	18	19	24
Weight, lb	21	35	21	30	30	60	42	80	22	29	43	20	24	34	40	29	22	40	33
Cyl diam, in	1 1/4	1 1/2	1 3/4	1 1/2	1 11/16	2	1 11/16	2	1 3/8	1 3/4	1 1/2	2	1 1/2	2	1 1/2	1 1/2	1 1/2	1 1/2	1 1/2

l features are: automatic rotation, and the spring-retained
. Designed as a hand hammer, but with the JM-6 mount-
feed), this drill may be used for drifting
McKIERNAN-TERRY A 3 self-rotating
valve, and hollow steel through which
to eject the cuttings. SULLIVAN DP 33
er hollow or solid steel; rotation is auto-
ntial, air-thrown type; the blow of the
rn stroke; drill bit is held in the chuck
. "CHICAGO HUMMER" (Chic Pneum
de in 3 sizes (A 66, B 66 and C 66) and
; designed especially for sinking and

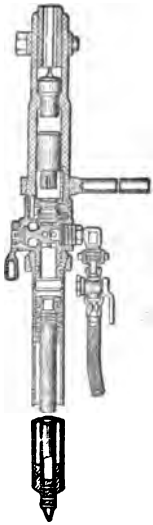

Fig 37. Sullivan Stoper, DA-21

ounted drills. A few hammer
ner-Ingersoll and new (1914) Sulli-
unted and operated like standard
drills.

can and some foreign drills, as Stephens
d the Kimber, are mounted, but in oper-
fer materially from the reciprocating
-INGERSOLL (formerly, "Water Ley-
: the combined air and water passing
through the hollow bit eliminate dust,
mper of bit, and by keeping the hole
increase drilling speed. Valve mech-
f the "Butterfly" type (Art 26). Bit is
uck without bolts. SULLIVAN MOUNTED
, for tunneling and drifting (Fig 35):
admits water or air, alone or together,
it, and air into the valve and cylinder.
nd of a run, the hole and steel are thor-
just before closing the throttle. Valve
rear of the cylinder; valve is hollow, with end seat, and oscil-
ent with the piston. The bit is rotated by grooves in piston,
bushing in the chuck. A water attachment allays dust.

hammer drills, or "stopers," have telescopic-feed

INGERSOLL-RAND "BUTTERFLY" drills are in 3 models. Fig
r and the chuck. (For valve motion, see Art 26.) Rotation
es to the cylinder. The air enters the drill through an "in-
he air-feed standard: 70, air-feed piston,
71, friction spring, which automatically
been run out; 73, air-feed cyl. Dust is
to the drill cyls. Model BC 22 may be
INGERSOLL-RAND "IMPERIAL" STOPER
eel (see Art 26). Anvil block is of tung-
and. There are 3 models: MC 30, with
oint; MC 31 with standard feed alone;
ey are not recommended for holes point-
LLIVAN STOPERS are of 3 models: DA 21
F 21 is best for soft or broken rock, and
They are rotated by hand. The DC 21
but has hollow bit. CHICAGO STOPER has
that in Fig 31. It is rotated by hand.
K DRILL CO) is of both valve and valve-
ed and automatic rotating mechanism.
s 40 C and 50 C stoper and 25 S "Little
e rotated by hand. The 25 CS model is like the 25 S, but
he 50 C model is recommended for hard rock; the other-

Fig 39. Ingersoll-Rand "Butterfly" Stoper, Showing Anvil *A*, Hammer *B*

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Table 69. Air-feed or Cradle-mounted Hammer Drills

Maker	Ingersoll-Rand Co				Sullivan Machinery Co					Cleveland Rock Drill Co			
Catalogue name	Water Leyner		"Butterfly"	"Imperial"	Mounted hammer	"Rotator"	Standard stoper	Heavy stoper	Air jet stoper	"Little miner"	Stoper	Stoper	Column drill
Model	No 16	No 18	BC 20,21,22	MC 30,31,32	DR 6	DP 33	DA 21	DG 21	DC 21	25 S	40 G	50 C	25 CS
Length, in	40	47¾	55½	56¼	.....	.....	.....	.....	.....	50	58	57	50
Length, air-feed extended, in	.....	.....	77½	78¼	.....	.....	.....	.....	.....	69	81	81	69
Feed, in	20	24	22	22	24	.....	.....	.....	.....	24	18	18	24
Weight, lb	95	150	72–83	70	146	38	84	89	86	50	70	85	55
Cyl diam, in	2¼	2½	2	1¾16, 2 1/16	2½	2¼	.....	.....	.....	1 1/16	2	2¼	1 1/16
Stroke, in	2½	3	4	3½	2 1/16	.....	.....	.....	.....	9	3	3	9
Shape of steel	round	round	any	cruc	round	hex	cruc	cruc	cruc	hex	cruc	cruc	hex
Diam of steel, in	⅞	1¼	1	⅞–1¼	1¾	⅞	1	1	1	⅞	1	1	⅞
Kind of steel	hollow	hollow	solid	hollow	hollow	hollow	solid	solid	hollow	solid	solid	solid	hollow
Depth hole, in	6–8	8–12	12	.....	12	6–10	8	8	8	10	10	10	8
Diam hole, in	1–1¼	1–2½	.....	.....	2¼	1¼	2	2	2	1¼	1¼	1¼	1¼
Free air per min, cu ft	76	87	64	.....	70	50	.....	.....	.....	55	60	80	55
Air press, lb	.....	.....	80	85	85	90	.....	.....	.....	80	80	80	80
Rotation	auto	auto	hand	hand	auto	auto	hand	hand	hand	hand	hand	hand	hand
Approx price, $	145	300	145	135	.....	100	135	135	150	125	135	135	125 (col.$35)

Maker	Chicago	McKiernan-Terry Co rotating hammer				Denver Rock Drill Co						
Catalogue name	Stoper	Stoper	Drifter	Combination stoper		Stopers			Waugh stopers			& drifter
Model	B 44	A 4	A 5	A 345	B 5	8 C	12 A	14 A	7 V	16 V	17 V	3 D
Length, in	54	63	63	27–63	.....	54	56	56	55	53	55	54
Length, air-feed extended, in	76	92	92	92	.....	78	80	79	79	77	79	80
Feed, in	22	29	29	29	.....	24	24	23	24	24	24	26

rock. McKIERNAN-TERRY models A 4 and A 5 (Fig 38) d mounted; A 5 is designed for column or tripod. Model hments for hand, cradle, or air-feed opera- No 5, 6, 19, 26, 29 and 36, are air-feed, hand- e among oldest of the hammer drills.

r working at altitudes, and with more and 69.

Valves and Other Details

ions of valves and other parts of typical

) **drills** are those in which the piston drills only). In soft or medium ground, han the valve machines, because they ort, quick blows; the valve drills have der.

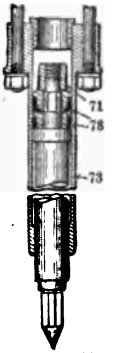
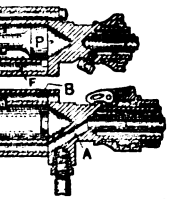


Fig 40. Air-Feed of Ingersoll-Rand Stoper

Imperial" (Fig 41). Air at full press con- the front of piston *P*. The lower cut shows the exhaust passing out through *B*. When per cut), live air is admitted through *F* to area of this face is greater than the other, In passing over *E*, the piston cuts off admis- d by expansion of the air. When exhaust on is formed in chamber *C*, preventing injury to the front what similar designs.

25) are used only in reciprocating drills.

alve (Fig 19) is a curved rocker, the lower ends of which are lders of the piston. The rocker is thus oscillated, and its upper arm throws the slide valve, which controls admission and exhaust of air. The duty of the rocker is severe, and its proper operation demands the best quality of materials and workmanship, and efficient maintenance.



erial" Stoper (diagram)

Air-thrown valves, of many designs, are used in both hammer and reciprocating drills.

Examples: SULLIVAN DIFFERENTIAL valve (Fig 27). The cen- a larger diam (and area) than the ends, and the alternate ston of the reverse ports 5, 5, cause the valve to reciprocate The piston will reverse under a very short stroke, permitting close and deliver light in seamy rock. Fig 26, own valves. Full press oke, insuring a powerful ND "BUTTERFLY" valve . In Fig 42, piston *P* is Since it covers rear ex- let air holds valve *V* to dvancing piston uncovers ess on valve wings. But, valve does not move until forward When piston passes over forward exhaust port, cushion pres- n acts through forward inlet port, and throws valve, opening E_1. Back stroke then begins, exhaust escaping through E_2. Fig 31, 43) consists of a 1-oz hollow hardened steel ball, a"

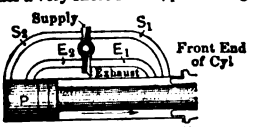


Fig 42. Ingersoll-Rand "Butterfly" Valve

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apertured steel pocket to confine it, and 2 steel seats limi
thrown by drop in press occurring when piston uncovers e

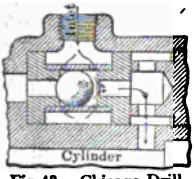

Fig 43. Chicago Drill Ball Valve

not dependent on compression.
seat until piston moves far eno
port, which moves ball to that

Corliss type valve, used on
(Fig 32), is oscillated by tappet
other tappet valves, it will no
thrown valves, but that the str
by feeding the cyl forward until
It is said to be strong, and, bei

Auxiliary valves are typifie
20, described in Art 24) and C
Main valve is a spool; the auxiliary, an arc-shaped tapp
valve (diagrammatically shown as flat) has been thrown t
shoulders (Fig 24). Hence, live air passes through auxiliary port *A* to left-hand end of main valve, throwing it also to the right, and admitting air to back end of cyl. On nearly completing forward stroke, the other piston shoulder reverses the auxiliary valve to the position in Fig 44 *b*; when, as shown by the arrows, the main valve is thrown to left, for making the back stroke of the piston. The hollow plug *B*, having a very small opening, admits a little live air to back end of main valve chest, and
of the main valve, and prevent piston from striking the

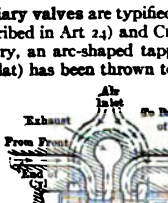

Fig 44. Diagram of

Cushion valves, claimed to prevent the piston from st
wide open, are used on Sullivan "Hy-speed" (Fig 33) and su
motion, the drill can operate at practically full speed when
or on an inclined surface; deflected holes can more readil
pumping out a mudded hole preparatory to charging. T
spring, and a cord can be attached to the lever, so that
the operator's reach, the valve may be thrown to its effe
cord. It is a combined plug and poppet valve, near
arranged that, when open, a free passage is left through
without cushion. To operate with cushion, the valve h
communication between the main port and front end of cy
action. This admits air to the other end of the cyl, but w
With the valve in this position, the drill runs as before,
lower end of cyl cannot escape except by leakage. In ru
up cushion press, which is returned to the back end of th
to exhaust. Small losses by leakage are automatically re

Rotation devices. "SERGEANT" SLIP ROTATION (Fig
ting the drill to free itself in broken or "fitchering" groun
The ratchet is held in the back cyl hea
of an adjustable cushion spring (Fig
the bit "glances" or twists backward
small steel cylinders for pawls, and lo
ery of the ratchet head. "SERGEANT

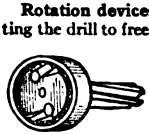

Fig 45. "Sergeant" Slip Rotation

freeing the piston from rotation, or n
"broaching" between rows of holes,
ment with the ratchet teeth by a re
to turn freely on both strokes. A handle is furnished fo
the plug is not used the rotation is the same as if the dri
tion," described above. Drills having the "release rota
cushion springs, pawls and rifle bar. "LITTLE GIANT" a

nout studs. They fit in recesses in the ratchet box with flat g surfaces. SULLIVAN DP 33 self-rotating hammer drill (Fig vice near the front end of the machine. There is no separate ved to engage a retaining bushing in the chuck. Grooves are mmer, some straight and some spiral. The straight grooves retaining bushing, which is screwed into the e ratchet collar. Positive rotation is thus ifle bar.

Fig 46. Device for Holding Bit in "Jackhamer" Drill

ng screw, turned by a crank, and engaging shell. AUTOMATIC FEED (Ingersoll-Rand making deep holes in a uniform, free-cutting nd feed, these machines have a tappet enter- hich is struck by the piston on approaching an exterior, longitudinal bar, turns a ratchet SULLIVAN FP 33 ENGINE-FEED is designed ed, being used for deep holes in open cuts, gh for an auto-traction drill (Sec 5). The peed" drill, but the cyl stands out from its evolve freely when drawn back in front of the e unusually long, and a little 2-cyl reversing the shell for raising and lowering.

k (Fig 19–33) is commonest. Square nuts are better than less easily. Fig 23 indicates use of a slip bushing for octagon 28, 30, 31 show wedge keys which facilitate changing bits. In he bit, on which lugs are forged, is inserted loosely in the chuck, until the lugs strike against stops. CHUCKS FOR SHANKLESS he Chicago hammer drill, the bit is held by a plate against ing is pressed. For the Ingersoll-Rand "Jackhamer," a steel ld against the bit by notches in drill head (Fig 46).

prays. Fig 47 illustrates a dust arrester, operated by com- and applicable to Ingersoll-Rand BC hammer drills. Other e similar devices.

rs, for removing cuttings from the hole, are sometimes used, ollow and solid steel. With solid steel, the spray is held in against its mouth, by a collar around the drill steel.

27. Drill Mountings and Carriages

Use of tunnel columns and carriages, and shaft bars, is treated t 4; tripods, quarry bars, gadders, channelers, and towers for drilling in Sec 5.

rts (Fig 48): *a*, saddle clamp; *b*, clamp-jaw; *c*, saddle through- ; *e*, side and back-leg bolts; *F*, right hanger; *J*, *K*, leg collar and set- veight. VARIATIONS of the tripod: parallel holes close together; quad- rilling from a platform. With the e when changing bits.

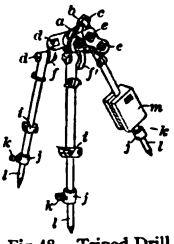

Fig 48. Tripod Drill Mounting

le-screw columns (Fig 49): 1, col- jack screw for double-screw column; ase-block bolt (small); 6, jack-screw cs); 8, jack-screw nut, double-screw , column arm; 12, column-arm cap 16, jack-screw cap; 17, jack screw; nut. COLUMN CLAMP: *b*, clamp- *P*, *Q*, column clamp, top and bot- lamp bolts. Columns weigh from o lb (3⅝-in drill or larger); cost, $25 to $40 without arm.

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Hammer drill mountings: TELESCOPIC AIR FEED (Art 25) is exte
firmly on the floor or wall, with the drill bit against the rock. Fi
light, telescopic column, the piston of which ends in a hook. The
its side rods from the hook, its weight being carried by a spring und

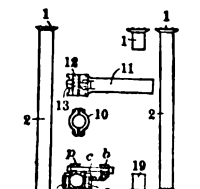

Fig 49. Double and Single-screw Columns

umn. In operation, the drill is adjust
by manipulating an upper hand set-scr
set-screw is released, so that the spri
CRADLE MOUNTING is a more permanen
pole." It consists of a skeleton cradle
hinged clamps, one at side of the cradl
cylinder, the other engaging the drill h
drill is held firmly in position by tighter
screws, and the cradle is designed for att
column or tripod saddle, except those
On this mounting the drill is handled lik

28. Compressed-air Co

Coal cutters are widely used to
for undercutting the face of coal, pr
ing it by blasting or wedging (Fi
economize in cost of mining; to decrease proportion of fines;
production from a given extent of workings (3).

[**Note.** — Air-driven pick machines alone considered here; for elect

Fig 50. Undercutting with Pick Machine

"New Ingersoll" coal puncher or pick is for underc
pitching seams, the angle of which does not exceed 10° to
in coal no rotation of the bit is necessary, as for drills.

Table 70. "New Ingersoll" Coal Punchers o

Model	Size			Dimensions, incl extensions & picks				Size unde
	Diam of cyl, in	Stroke, in	Diam of wheels, in	Width over all, in	Max length, ft and in	Min length, ft and in	Height, floor to top of chest, in	Depth, ft and in
F 4, wide gage	4½	14	14	22½	8 0	6 9	18⅝	4 6
F 4, narrow gage	4½	14	14	19½	8 0	6 9	18⅛	4 6
H 4, wide gage	5	17	17	24¼	8 6	7 0	20⅛	5 0
H 4, narrow gage	5	17	17	21¼	8 6	7 0	20⅛	5 0
H 7, wide gage	5	17	16	24¼	8 6	7 0	19¼	5 0
H 7, narrow gage	5	17	16	21¼	8 6	7 0	19¼	5 0
H 10, wide gage	5½	17	17	24¼	8 6	7 0	21¼	5 0
H 10, narrow gage	5½	17	17	21¼	8 6	7 0	21¼	5 0
H 12, wide gage	5½	17	17	24¼	9 6	8 0	21¼	6 0
H 12, narrow gage	5½	17	17	21¼	9 6	8 0	21¼	6 0

extension, wrenches, hammer and shovel, boxed; hose

to depth of 4½ ft. It is claimed by its makers that it is successful 12° to 15°, and can work in a 3-ft seam. Not suitable for very ell in soft, medium, or hard coal. H 7 is intermediate in wt being a larger cyl than F 4, it is adapted for harder coal. H 10 is aving a larger cyl, it strikes a harder blow, and is for use in very eavy machine, for thick, very hard seams.

cks on large, "SHEARING" WHEELS (see last columns, Table 70), as for driving entries or gang-ways.

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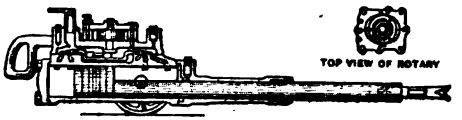

Fig 51. Harrison Pick Machine, Class W

groove close to floor; double ports for cushioning at each end of N in classes W and GGG is independent of piston movements, being rotary engine, its piston being heavy enough to act as a flywheel. valve by a crank and connecting rod. Air is cut off at about d and force of blow can be regulated by the operator.

Table 71. Harrison Pick Machines

h of e, in	Length of rod, in	Diam of wheel, in	Wt of striking tool, lb	Wt of machine, lb	Depth of undercut, ft
	60	16	175	770	4.5 to 5
	66	16–18	187	785	5 to 5.5
	72	16–18	200	810	5.5 to 6
	68	16–18	212	825	6 to 6.5
	54	14–16	135	715	4 to 4.5
	60	14–16	147	727	4.5 to 5
	66	16–18	159	740	5 to 5.5

onclad Compressed-air Chain al Cutters

	"CE7" room and pillar	"CH8" long-wall
n.....	66, 78, 90,	24, 36, 42, 54, 60, 66,
cl bits, in	21	20½
, in.....	140 to 164	95½
props, in	70	33
in.....	23⅞	17⅞
eels), in.	34⅜	28⅜
ar (12-in		
.....	30⅜	24⅜
ut), in. .	5¾	5
.....	30	30
.....	800	800
.....	15, 18, 21, 24, 27, 30, 33, 36, 39,	12, 15, 18, 21, 24, 27, 30, 33, 36,
pan, lb.	4 700	4 700
plete, lb.	1 000	.....
lete, lb.	800	800
ent, lb.	900	900
car, lb.	7 500	.....
car, lb.	7 200	6 900

In class GGG, piston rod is symmetrical with piston, and is prevented from rotating by a spline. Class PW has same size cyl as class W, and though slightly lighter will undercut to about same depth. It is built with 3 lengths of piston rod, and is designed for 4-ft veins and thicker, and for hard mining. It has a piston valve, and cyl and chest are cast in one piece. Class PG has 4.75 by 16-in cyl, is built with 3 lengths of rod, and has same valve motion as class PW. It cuts to same depths as class GGG.

Sullivan coal picks are made with 4½, 4¾ and 5⅛-in cyls; they undercut from 4.5 to 6 ft, and weigh 650 to 850 lb (3).

Chain undercutters, usually electric-driven (Sec 16), but Jeffrey Mfg, Link-Belt Mach'y, Sullivan Mach'y Co and others, make air-driven chain cutters for use in gaseous collieries

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Table 73. Ingersoll-Rand "Radialaxe" Coal C

	Outfit for shearing and undercutting, 24-in feed			Outfit for shearing and undercutting, 18-in feed		"48 for 18
Model.....	C-92	E-92	F-92	E-92	F-92	
Diam of cylinder, in.....	2¾	3¼	3½	3¼	3½	3
Length from crank end to chuck (closed), ft, in.....	3 9	3 11½	4 0	3 11½	4 0	3
Depth of cut, with 1 bit, in ...	24	24	24	18	18	
Total depth of cut, ft.....	8	8	8	7.5	7.5	7
Net weights						
Drill and shell, lb.....	215	250	262	250	262	2
Mounting, lb.....	136	136	136	136	136	1
6-ft column, lb.....	130	130	130	130	130	1
Extension steels, 4 to 8 ft, lb	89	89	89	116	116	1

"Radialaxe" coal cutter (Ingersoll-Rand Co) is a rock drill, w mounted on a column, and provided with worm gear, so that the n radially in undercutting or shearing.

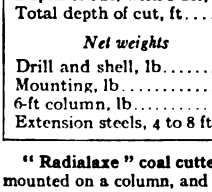


Fig 52. Bits for "Radialaxe" Coal Cutter

of 2 forms (Fig 52). Solid bits, forged used only for the machine with the "9 shearing and undercutting (Table 73). are of the combination "fish-tail" type and sharpened by the mine smith.

23. Compressed-air Lo

Note.—Sec 11 deals with all means port, including electric and gasolene l treats of compressed-air locomotives o

Field of work. This type of lo to all mines having compressed-air to gassy collieries, in which electric or fuel-burning machines ADVANTAGES: since the power is stored in the locomotive i wherever track is laid, returning at intervals to the chargin is consumed when the locomotive is idle; the exhaust tend power, greatly in excess of the average, is obtainable by te tank press in the cylinders (3).

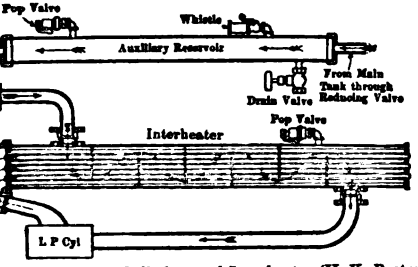
General construction. There are 1 or 2 storage tanks etc, are carried on a frame mounted on 4 or 6 driving wh usually simple, sometimes compound. The high-press air passes into a small auxiliary tank, where it is reduced to p before going to the cyls.

Air pressure in storage tanks, 700 to 1 200 lb. Cyl press: simpl compound engine, 200 to 250 lb. Intermediate press for compoun

Tank capacity, for hauls of 1 to 1.5 mile, varies from 50 to 150 it is better to increase the press, rather than the size of tanks.

Reheating (Art 20) is desirable to restore the heat lost in reduci the auxiliary tank; it is essential for compound cyls, the temp of high-press cyl being far below freezing. Fig 53 shows the H. K. Po for compound locomotives. Between the cyls is a shell filled with which the warm atmospheric air is drawn by the ejector action of

in the high-press cyl is thus restored, expanding the air about
t the use of the interheater effects a reduction of about 30% in



of Locomotive Cylinders and Interheater (H. K. Porter Co)

boilers, and in power costs, and of about 15% in first cost of
ase of about 30% in length of haul on 1 charge of air.
imple method of reheating is to inject into the auxiliary tank
time the locomotive is charged. In non-gaseous mines, a small
ace is used.

le 74. Compressed-air Haulage Plants

Tanks			Cyls		Locos	Cars, wt						
Capac,	Air press	Working press	Diam	Stroke	Height / Length / Width ft	Empty	Load	Cars in train	Max grade	Length round trip		Charging time
cu ft	lb	lb	in	in		lb	lb		c%	ft		sec
130	550	...	7X14		5.2X19X —	3400	7000	12	4.5	5100	...	...
130	550	160	7X14		.....	2800	7000	12–20	2.8	8000 / 4200	...	...
30	500	...	....		4.3X5X2.5	....	2000	5	....	....	....	....
....	500	125	5X10		4.5X0.5X5.8	8500		6	2 5	9000		20
....	600	180	....		.....	3300	5500	15–38	0.8	5200		60
....	600	180	....		.....	....	....	4–10	5.5	3200		....
....	600	200	....		6.5X14X6.3	9000		32	1.7	....		....
160	600	125	9X14		.....	5850		30	1.0	9000		85
47	550	125	5X10		5X10.4X5	3450		6	0+	....		60
218	750	150	12X18		.....	....	....	....	....	1000 to 7000	...	...
132	....	....	9.5X14									
....	....	....	....		5X10.5X3.3	....	....	....	....	....		....
....	700	140	7X12		5.2X12X4.2	1800	1800	20	....	1200 to 4000		45

livered 150 cars in 10 hr, formerly requiring 10 mules. Aver
, gross, 1.875¢; net, 3.77 ¢; by mule, gross, 3.94¢; net, 7.92¢.
$r 800. Max speed, 10 to 12 miles per hr. Capac of main stor-
lb press. Press in pipe line, 735 lb. (c) (See "Cost of Plants.")
ar: daily operating cost, $14.69; fixed charges @ 10% and cost of
cost per day per car for 666 cars, 3.6¢ (previous cost by mule, 5.1¢
cos are 4-cyl compound, 5 and 8 by 12 in. (e) Pipe-line press, 700
er gross ton-mile, 1.25¢. (f) 2 round trips per charge on cold air;
ocos have detachable rear end, to permit loading on to a 9-ft

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Tractive force of compressed-air locomotives may be foun
$\times$ 0.98 P) $\div$ d: where T = tractive force, lb; D = diam of pist
in; P = initial press, lb; 0.98 P = effective cyl press; d = di
in (33). For compound cyls, the effective cyl press is taken
actually the difference between the initial press in high a
and it may be assumed that there are either 2 low-press or 2
the same power is generated in both.

Table 75. Compound Compressed-air Locomotives (H.

Class	Cylinders, in			Rigid wheel base, ft, in	Length over bumpers, ft, in	Extreme width outside of gage, in	Height, ft, in	Main tank capac, cu ft	Weight in working order, lb
	Diam, h-p	Diam, l-p	Stroke						
B-P-O	4.5	9	10	2 9	9 0 12 6	12¼	(a)	20 60	7000 10000
"	5.5	11	10	2 9	10 0 12 6	14¾	(a)	30 75	10000 13000
"	6	12	12	3 0 4 0	12 0 15 6	15	(a)	60 104	14000 17000
"	7	14	14	4 0	12 0 18 6	16½	(a)	60 120	18000 22000
P-B	4.5	9	10	2 9	9 0 12 6	12¼	8 0	20 60	5000 10000
"	5.5	11	10	2 9	10 0 12 6	14¾	8 6	30 75	10000 13000
"	6	12	12	3 0	12 0 15 6	15	"	60 104	14000 17000
"	7	14	14	4 0	12 6 18 6	16½	9 0	60 120	18000 22000
B-PP-O	7	14	14	4 0	14 0 20 0	16½	5 0 5 10	97 160(c)	20000 27000
"	8.5	12&12	14	4 6	16 0 20 0	16½	"	114 120(c)	28000 36000
"	9.5	14&14	14	4 6	18 0 21 0	18	"	160 275(c)	32000 42000
"	10	14&14	14	5 0	20 0 23 0	18	5 5 5 10	160 290(c)	43000 46000
B-PP	8.5	17	16	5 6	14 0 18 0	16½	10 0	114 120(d)	28000 36000
"	9.5	19	18	5 6	18 0 21 0	18	11 0	160 290(d)	32000 42000
"	11	22	18	5 9	19 0 22 0	25¼	12 0	260 375(d)	55000 60000
"	13	26	24	6 6	22 0 24 0	29¾	"	" "	69000 73000
"	15	30	24	6 6	22 0 26 0	33	"	" "	95000 100000
C-PP	7	14	14	5 6	14 0 20 0	16½	5 0 5 10	97 160(c)	20000 27000
"	8.5	12&12	14	5 6	16 0 20 0	16½	"	114 210(c)	28000 36000
"	9.5	14&14	14	5 6	18 0 21 0	18	"	160 275(c)	32000 42000
"	10	14&14	14	5 6	20 0 23 0	18	"	160 290(c)	43000 46000

(*a*) Depends on gage. (*b*) Radius advised, 25–50% greater. (*c*
3 tanks. All main tanks are built for charging press of 700 or 1 2
press for all, 250 lb.

ers (Art 4, 7) are 3 or 4-stage, to divide the work of compressing several cyls, for more efficient cooling. Also, with high press, eep valves and pistons tight, to use only a small quantity of high and to insure that the cooling water runs freely and fills every whenever the compressor is running. It is well to provide the le plug, which will melt in case of excessive temp, and warn the nd remove the cause of trouble.

harging stations. With short hauls no pipe line is re- es being charged at the receiver. On long hauls, one or is are necessary.

added storage capacity. STORAGE CAPACITY and press of pipe line in locomotive tanks, are determined by $V(P-p) = v(p-p')$, in required, cu ft; v = vol of locomotive tanks, cu ft; P = pipe- = desired press in loco- in; p' = residual press st before charging, lb

3 to 6-in, lap-welded, ats. Charging station set in the main line, one or more ball and cting with locomotive ; time is from say 0.5 to

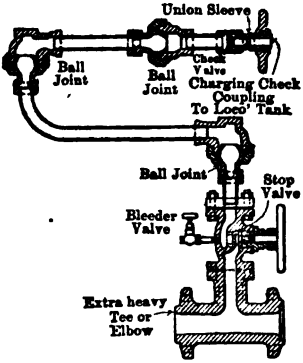

Fig 54. Charging Station, H. K. Porter Co

ompressed-air locomo- ms, Westphalia. The out 262 ton-miles per ce ranging from 2 295 -stage compressor sup- r min at 1 500 lb press. r goes to a 35-cu ft re- t down the shaft in a vels, 3 other 35-cu ft idistant, for charging educing valves give a locomotive tank press of 750 lb. Locomo- 7 cu ft capacity, corresponding, at 750 lb, to 2 910 ft free air. In press is reduced to 150 lb for the cylinders. The locomotive has total wt 5.5 tons. Normal power is 8 to 12 hp; max, 24 hp; haul- ed cars, weighing 2 180 lb gross each, on an up-grade of 1 in 300 at a

Actual running time per 8-hr shift for 2 locomotives was 313 min, in shunting and stoppages. Total cost of haulage, about 3.5¢ per g cost, inlcuding interest and sinking fund, $8 750 per year.

Buck Mtn colliery, Pa (1897) (3):

ves		$5 505.00
4-in pipe	$2 894.00	
ing stations	360.00	
s and valves	382.00	
cost of erection	998.00	4 634.00
	2 880.00	
ction, $246; steam line, $152	398.00	
e	256.00	3 534.00
		$13 673.00

ding Coal & Iron Co (3):

otives and fittings	$4 904.00
5 in, and 3 charging stations	2 951.00
age compressor, erected	5 181.00
ngways	665.00
	$13 701.00

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C. Consolidation Coal Co, Georges Creek (1902) (34):

2 main locomotives, 4-ton	
5 gathering locomotives, 4-ton	
Pipe line: 5 600 ft 5 in	$5
3 100 ft 2.5 in	1
1 000 ft 1.5 in	
3-stage, comp compressor, $5300; 2 80 h-p boilers, $1000	
Installation, labor and material	

D. No 6 Colliery, Glen Lyon, Pa, installed 1895–96 (31):

1 Porter locomotive for shaft, 7 by 14 in, 18 ton	$2
1 do for slope	2
Extra for motor repairs	
Air line: 5-in (6 000 ft)	2
3-in (4 000 ft)	1
Foster equalizing valves	
Compressor: 1 Norwalk 3-stage, 12½ by 9½ by 5-in air cyls, and 20-in steam cyls, by 24-in stroke; 296 cu ft free air per min @ 600 lb	2
Steam connections to compressor, etc	
Material for compressor foundations, house, and air-washer	
Labor for installation	1

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SECTION 16

ELECTRIC POWER FOR SERVICE

BY

GEORGE R. WOOD

ELECTRICAL ENGINEER, BERWIND-WHITE COAL MININ

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Note. — Numbers in parentheses refer to Bibliography at end of

1. Purchased Power

Electric power may now be purchased from Public Servi most coal and metal mining regions, an arrangement which is ical than the operation of an isolated plant (1). Cost of SUB-ST FORMING APPARATUS will be from $25 to $30 per kw of rated with $60 to $80 per kw for power plant. This comparison as station, and a power plant of equivalent d-c capacity, but with one or more sub-stations would be required, the purcha save the entire investment for power plant.

Cost of purchased power depends largely on load factor aver to max load. INSTALLED CAPACITY is fixed by peak or plant must supply max demand of all connected loads. OVERH ing interest, deprec, taxes and insurance, is a fixed percentage, vestment. This amount, divided by output in kw-hr, represen kw-hr, which obviously will be a minimum for the highest ratio

On account of distribution of peaks over different periods of the da central stations, serving customers in various lines, have load factors pared with 10 to 20% for the aver isolated mining plant. In the Ka coal fields, W Va, 56 plants, averaging 200 kw, had a load factor of

ity, or 16.4%, based on ratio of aver load to the 5-min integrated per kw is assumed for the central plant, including transmission 60% and overhead cost of 15%, the overhead cost per kw-hr will the usual mine plant, costing $80 per kw, with load factor of 10%, 0.7¢ with a 20% load factor. In these examples, the peak load is capacity. Special rates are made by a number of power com- g their existing plants by purchased power, when the purchaser d steam plant so as to be available in case of breakdown.

is usually in the form of 3-phase a c, and, while haulage ormation to d c at low voltage, other apparatus may be with a c, at low or medium voltage. Induction motors ch are rugged and efficient, and have no commutators or c is also better adapted for lighting, as it can be trans- ge for use in buildings.

yond the power or range of an existing d-c plant are required, operate one or more sub-stations, and if of proper design these the d-c plant, and at night or on idle days either may be shut is especially economical when strikes, breakdowns, or lack of ince a sub-station may be started in a few min, and pumps, fans be operated for a few hr at a cost only in proportion to work plant must be kept in continuous operation. When ADDITIONS quired, an added sub-station unit may readily and cheaply be in- ted plant added boilers, piping, engine, and generator are needed. is an important consideration in deciding between purchased n of a generating plant, and the possibility of failure of the pur- nsmission troubles must be carefully investigated.

r ton of product generally determines whether to use erect a plant; few mine owners have accurate data on r in existing plants, and in case of new operations this must purchased power may be estimated closely from published n in this case the peak load and probable total consump- Where it is proposed to substitute purchased power for an rding watt-hr meter will show the total consumption, and rvals will give the integrated peak for that interval; or a be installed, which will show the sustained peak for any

purchased power is usually based on a FIXED CHARGE of per kw of peak load, plus a METER RATE, which is reduced es. Other rates base the fixed charge on a certain per- e instantaneous peak, or upon the installed sub-station se, the fixed charge is taken as the minimum charge per purchased power in coal-mining districts ranges from 8 for installations above 100 kw. At some western hydro- osts range from 2¢ to 5¢ per kw-hr.

epresent the aver load for a specified time, as 5, 10, 15, or PEAKS are sometimes used, of 3, 5, or 10 min duration, itions.

intermittent apparatus only, as hoists, locomotives, and coal be such that no load would be sustained as long as 5 min. k, or one arbitrarily determined as a certain percentage of the ation capacity, is generally used as a basis. With mining loads capacity, if the 15-min integrated peak is 100%, the 10-min peak e 5-min peak, 110%. For larger capacity, 300 to 500 kw, the , and 106.5%, and for a number of mines (10 or more), the range %, 103%, and 105%. Aver of 56 plants showed 5-min inte- installed capacity, and this arbitrary percentage is often used

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Feed pumps in mining service are subject to the general rule ordinary power plants, where the service requires heavy drafts on t tively short periods. It should be remembered, when selecting p often remote from repair shops; therefore, rugged construction a pumps should be in duplicate.

Feed-water regulators are desirable in mine service, provided will not operate within too narrow limits of low and high water. draft on the boiler would cause rapid addition of a large volume of sequent drop in capacity when it is most needed.

Feed-water heaters are preferably of the OPEN TYPE, as they a moving scale-forming elements, are much easier to clean, and will of the exhaust, which condensate mingles with the feed-water and the boiler. This type has the advantage that waste water, as fr turbine glands, etc, may be piped into it, as it is not under press the closed type may be used, and may be located at any conveni open heater must be not less than 3 ft higher than the feed pump

Water-purifying systems are sometimes installed in connectio and it is claimed are more economical than the cold-water type, as readily in hot water. Purifying plants for mine service must co impurities, as the water supply will often vary from "hard" to relative proportions of surface water and mine drainage.

Fuel supply for colliery boilers may come direct from m breaker, or part from each, or may be received in R R cars. supply and storage therefore vary widely.

If machine cuttings, bottom or roof coal, or road cleanings are loaded separately and delivered in mine cars. If the fuel is scre tipple or breaker, the screenings may often be loaded into mine cars venient in either case to dump the cars into storage bins from a track room floor. But it is often necessary, where fuel is separated at some continuous conveying apparatus for delivering it into bins refuse coal is used it is generally best to crush it to about 0.5 in, as distributes bone, slate, and other impurities, and makes combustion

Utilization of waste heat from bee-hive or Belgian types of coke ov There are a number of waste-heat plants near Uniontown and Brow Run, Pa, 100 bee-hive ovens supply 2 boiler plants, one at middle thus drawing from 25 ovens each way. Radial-brick stacks, 150 ft hp developed per oven depends on character of coal, time of coking With high-volatile coking coals, as the Connellsville, the yield from 48-hr coke, 1.2-in draft at bridge wall, is 16 to 20 hp per oven, with tity or quality of coke yield. With induced draft, as at Ellsworth Ellsworth, Pa, the yield is 20 to 24 hp per oven. With low-volatil Pocahontas and Clearfield, there is no excess gas in the coal, com the oven, and the power obtainable is reduced to 8 to 10 hp per ove to justify the large investment required. At the York Run plant, re are equivalent to use of coal at 70¢ per ton.

Ash handling and disposal at large power plants usually r conveyer (Sec 27). This service is very severe on account of possible it is better to dump the ash by gravity into a dump car

Railroad companies often supply cars and remove ash for ballast roads in mines and around the plant. In flat regions, a tower may cars handled toward its top on a suspended track, filling in track toward the tower. When the top is reached, cars may be run out more directions, and if tracks are properly graded, a small hoisti room will handle cars over a large dump. Fireman or water tender and if car can be run under boilers to receive ashes by gravity, one ash from a large plant. In most small plants, firemen can handl car or a wheelbarrow, but in such case their range of travel must b hauling ash for roads from a fixed dump or by handling it mechanic

8. Engine Room

.ery is determined by: capacity of the plant; whether a c
ted; first cost of installation; operating cost; reliability of
e required.

gine vs turbine. For a plant developing less than 300 kw,
lly preferable. Following tabulation assumes costs as they
change in cost would not alter the relation, the tabulation
comparison. The figures cover standard makes of turbine
engine, direct-coupled to d-c and a-c 300-volt generators.

att, direct current				Cost per kilowatt, alternating current			
0	200 kw	300 kw	500 to 750 kw	100 kw	200 kw	300 kw	500 to 750 kw
	$33	$30	$30	$45	$35	$27	$20 to $15
	22	20	$20 to 30	25	22	20	20 " 30

e the choice of reciprocating engine-driven units for installations
first cost and steam consumption are smaller. For larger units,
d space indicate choice of the turbine. In practice, there is almost
ts as large as 500 kw for mining purposes, and a-c generators,
natural choice for large installations.

dion. Either turbine or Corliss engine can be maintained at good
nitely, if properly cared for, though most types of turbine show
consumption as blades erode or corrode, clearances increase and
Single-cyl slide-valve engines require constant repair and attention
ny. For installations of 500 kw or less, it is usual to operate non-
er plants, condensing is necessary for economy.

Art 14, 15). Choice of type, whether surface, barometric, or low-
kind of impurities in the cooling water. Surface condenser is gen-
it returns the condensate to the feed-water heater. If feed-water
is a great advantage, since only the make-up water, small in
reatment. Barometric or low-jet condenser must be used when
is such as to make it inadvisable to pass it through the small tubes
Acid, dirty, or salt water forbids use of surface condenser. As
t mines is dirty and acid, barometric or low-jet is usually installed.
res a larger ratio of cooling water to steam than other types.

ondensing water (Sec 40, Art 15). When the supply of water is
sive power for pumping, or is too dirty and acid for use, even with
, a cooling system may be installed. The water loss in a well-
only the evaporation; it may aver as low as 1.4% throughout the
eed 3%. SPRAY PONDS are usually lower in first cost, are cheaper
and have longer life than cooling towers. Tests show that the
ipated by a properly designed spray-cooling pond is 127 B t u per
fference. Installed cost of a spray system to provide a 28-in aver
year is approx $2.50 to $3.50 per kw of turbine generator served;
nature of the ground makes construction of a pond difficult.

ction of generators for mine service is governed by careful
ocal conditions. In other kinds of service, the practice is
on the basis of continuous load, and to limit the driving
e continuous capacity of the generator can not be exceeded.
y overloads of 100% are often caused by large single units,
and crushers, and the length of the peak will depend on the
rvice. Starting peaks of locomotives or hoists are seldom
an 1 min. The load factor is very low, especially in " "

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plants; overheating is not to be feared, and the perm
more by commutation and capacity of the prime mov
therefore desirable to select generators with a capacit
(as high as 200%) for 5 min. Generators with COMMU
larly adapted to d-c mine service.

Switchboard (Sec 42, Art 12) in main or sub-stations fo
natural black slate, which is cheap, neat, does not show dirt,
cleaned with oil. Capacity of circuit breakers, switches, o
of indicating instruments should be ample to handle or in
OVERLOADS of the plant. For D-C PLANTS the required equ
ammeter, watt-hr meter, main switch, field-switch, circuit-
One voltmeter with potential plug is sufficient for each station
meter may be used to record the entire output of the statio
and pump circuits be kept separate from hoisting and haula
overloads may be expected, as the former are often operated
fan and pump motors can not be supplied direct from bus-
starter may be used. Use of a watt-hr meter is recommen
output of each station, as this is essential to proper cost keepi
often be effected by locating the cause of increased power con
per TON OF PRODUCT is a direct check on the effic of all ele
A-C PLANTS the station panel should carry a-c voltmeter, freq
(for more than 1 unit), and Tirrill regulator, to maintain a-c
Each generator panel should carry a-c ammeter (or indicatin
power-factor indicator, field rheostat, field switch, and
Feeder panels should carry a-c ammeter (or indicating wattm
watt-hr meter, and automatic switch and circuit breaker.
sub-stations or a number of motors, should have a TIME-LI
operation of the overload trip, to permit individual circuit
short circuit without interrupting supply to other branches.

Ample space should be left behind all switchboards, fo
repairs. D-c boards should be not less than 4 ft from wall.
oil switches and high-tension connections against wall, wit
clear between panels and switches, a total of about 7 ft.

Station wiring between machinery and switchbo
in a conduit under or in the floor; this makes a neater
ance or damage, and does not interfere with work on o
should be insulated with rubber or varnished cloth, and
ments, should be lead-covered. For wire gages and cu
Art 3, Table 4. For insulations other than rubber,
Am Inst Elec Engs.

Station records can be made adequate and accur
meters, and their importance increases with capacity
fuel consumption and kw-hr output, which, with the
will show lb coal per kw-hr, kw-hr per ton of product
records, the cost per kw-hr and of power per ton pro

Large plants, supplying several mines, can also afford to
water, feed-water temp, power used at each mine, power
graphic records of steam press, station voltage, and total
instruments should be provided. Portable instruments sh
ammeter, and larger plants should also have a-c and d-c watt
cycles of apparatus and check up switchboard instruments.

Examples. (*A*) A c operates 3 car hauls and hoists, 100
1 fan motor, 25 hp; 2 transformers for lights, 30 hp; and or
furnishing d c for: 13 pumps, 37 hp; 7 hoists, 21 hp; 6 punc
ing machines, 90 hp; one 10-ton loco, 100 hp; two 12-ton locos
750 hp. During 13 mo, max demand ranged from 55 to 112; a
tion, 27 400 to 38 500 kw-hr; aver, 32 815 kw-hr. Load fact

, 60-cycle) operates one 50-hp, 2-speed (450 and 900 rev per hich delivers 112 000 to 115 000 cu ft air per min at 1-in wg, g working hours only; one 22-hp slip-ring motor on picking ; and 2 synchronous motor-generator sets having combined lts, operating one 13-ton loco, two 10-ton locos, three 6-ton and 2 small pumps. Power consumption during 1 yr totalled tput of 236 842 tons coal, and was distributed as follows:

	Motor-generator, d c	Fan, a c	Picking table, a c	Lights, a c	Total	Kw-hr per ton mined (a) 7 months
kw-hr.....	306 880	118 860	(a) 7558	1 253	434 551	.....
per mo, kw-hr	25 573	9 905	1 080	104	36 211	1.835
.....	70.5	27.4	1.8	0.3	.....	.....
.....	1.294	0.502	0.034	0.005	1.835	.....

, 2-speed motor driving a fan which delivers 66 000 cu ft air one 10-hp motor driving another fan; also one 150-kw syn- et supplying 250-volt d c to: three 8-ton locos, 240 hp; one -hp pump motors, 26 hp; 4 hoists, 20 hp; and 5 short-wall nected load, 471 hp. During 1 yr, total consumption was of 226 935 tons, or 1.715 kw-hr per ton.

b-stations (See also Sec 42, Art 13)

ns for the sub-station, to transform from a c to d c, ng whether rotary converters or motor-generator sets

2, Art 11) is self-starting and self-exciting, and for short loads. It may be started as an induction motor, thrown on ull, voltage, in which case it is self-synchronizing. When the ed by this is objectionable, the converter may be started by hich case it must be synchronized at full speed before being

o methods has been successfully used in mine service, which rrent at low power factor of the first method and the necessity uires skilled attendance, of the second method. The rotary y an induction motor; then thrown on the a-c line in series then cut out by a short-circuiting switch. This method is st Penn system in the Connellsville, Pa, field and by the Ber- Pa.

c 42, Art 11) can be operated directly on a-c lines at voltages hout use of transformers. The generator end can be com- for line loss.

, including transformers, is approx equal to that of a motor- ers. EFFIC of a rotary converter, including transformers, is f a motor-generator set without transformers.

. For 220 to 300-volt service, it is particularly im- rator be COMPOUNDED, to compensate for line loss. sub-station apparatus, but it is a decided advantage, hen serve a larger territory and maintain more satis- nd. Hence, a motor-generator set is to be preferred

ems, the range of each station is largely increased, and com- . The higher effic of the rotary converter recommends it, d c, unless considerable power-factor correction is requir-d

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This can not be obtained from a rotary converter, since it must not vary from u factor at full load and over, on account of excessive heating of the "tap" coil suitably designed, a synchronous motor may be run as high as 80% leading po

Commutating-pole type of rotary converter (Sec 42, Art 11) is adapted to mine service, because of its enormous overload capacity. factors of 30% and less, which are usual at mines, it is common to use s apparatus which will average half load and occasionally carry peaks of load. With commutating-pole rotaries, the machine may be operate rated capacity, carrying the peaks at overloads of 200%. This gi higher all-day effic and satisfactory service, with a machine of half the These machines may be obtained as standard in sizes of 100 kw and cycles and as small as 300 kw for 25 cycles. They are usually connect and have remarkably good effic; the 150-kw, 60-cycle, 275-volt machi ing 94.7% at 1.5 load, and 93.5% at 0.75 load. Price, $1600 f o b, facto

7. Transmission Lines (See also Sec 42, Art 13)

Calculation of d-c 2-wire circuits: if I = current in amperes, or wat L = length of transmission, ft; V = volts lost in transmission; then cir in conductor = $(22 \times I \times L) \div V$.

For haulage work, where the track forms the return circuit, the rail capacity at least equal to that in trolley and feeders. Steel rails in general have 12 t sistance of copper, and for 2 rails will be equal to 20 000 cir mils of copper per Thus, a track of 40-lb rail will be equivalent to 800 000 cir mils. When capac and rail are unequal, the drop in each should be figured separately, using $V = 11 \times I \times L \div$ cir mils, in which case V is the drop in half the circuit. is distributed, L represents aver distance.

Calculation of single-phase a-c circuits. Formula given above ma as reactance may ordinarily be neglected. But the current, represented h watts $\div E \times$ pf, where E = volts between conductors and pf is power fa circuit. For 3-phase circuits, universally used for mines, the current I i will be kw $\times$ 580 $\div E \times$ pf, and the size of wire in cir mils = $19 \times I \times$

Example. To transmit 500 kw, 3-phase, 6 600 volts, 85% pf a distance of 20 300 volts loss. Current in lines will be $500 \times 580 \div 6\,300 \times 0.85 = 54$ am mils = $19 \times 54 \times 20\,000 \div 300 = 68\,400$, which is nearest a No 2 wire. W ties are large, reactance must also be considered (2), as the losses from this exceed resistance losses (Sec 42, Art 13).

Permissible losses may be determined from standpoint of economy mission or from operating conditions. It is common practice to fig lines with a loss of say 10%; but large saving in transmission cost may by considering that, as a general rule, for best economy the value of should approximately equal the INTEREST charges on the conductor.

Example: Assume 500-kw aver load at 500 volts, delivered 1 000 ft with 5% power costing 2¢ per kw-hr, this loss amounts to $4 320 per year. The requi from above data, is 5 200 lb, which, at 15¢, will cost $780, and interest on this $36.80. Total yearly cost of delivering this power is $4 367 per year, of which charge is about 1%. It is evident that a much larger investment in copper wi power loss would be more economical. For trial, assume 3 times as much c 0.2 as much power loss. Interest charges will then be $234, power loss $86, transmission cost is $1 098, a saving of $3 269 per year. Evidently a saving of might be effected by still further reducing the loss to about 0.5%, but this wo an investment of about $4 000 for added copper. On BASIS OF COST the in above cases would be large, as the copper required for least annual cost woul 9 000 000 cir mils. An even larger installation would theoretically be justifi power cost higher or copper cost lower; on the other hand, with power cost size of conductors would be halved.

OF VIEW, it is advisable to allow not more than 33% drop at max
actory operation of mining machinery under the worst conditions,
ons will be greatly improved. This appears an excessive drop, but
ge number of mines show max drop of 50% and over. With 10-hr
this max loss represents an aver loss of 6.6 to 10%. With poten-
the mine openings, the minimum voltages underground will be 200
isfactory operation of machinery designed for 220 and 500 volts.
ial in coal mining, makes a large copper investment inadvisable.
full-rated voltage should be maintained, since output and torque
as the square of the applied voltage. With step-down trans-
ed voltage taps may be used, line drops of 10 to 15% may be
l, and particularly with synchronous converters, it is best to
for not over 5% drop at rated capacity, which should be taken
ated peak. Where this is not known the continuous rating of
ted may be used.

ation of transmission lines for either d c or a c should be
accessibility, economy, and safety.

ction and repairs is of first importance, and if conditions permit
ays to make a considerable detour, as it will facilitate finding
se of ground or short circuit, a repairman can cover the ground
horseback or with automobile. In many cases, particularly in
le or more added length of line will give advantages worth many
ich location is not available, lines should be straight as possible;
t least twice the line spacing over or under any other lines; and
should be kept high (30 to 40 ft), because if low, short circuits
aused by objects thrown over the wire.

ms (Sec 42, Art 14). Steel cross-arms and pins are best,
ns is short, unless treated, in which case the cost will ap-
of steel. The Bo-arrow and similar constructions are
nd do not require gaining and weakening a wood pole.
re common for long high-voltage lines, with spans of 200
ry mining work, wood poles are satisfactory for spans of
ires are not larger than No 0.

ground, the butts of wood poles should be treated with creosote
ll joints and wherever poles are drilled or cut should be similarly
grass fires are expected, place concrete around pole to height of
rock, or, if conditions justify, use steel poles.

continually increasing, and probably steel poles will eventually
ission lines. In central Penn, 35-ft chestnut poles, 22-in circum
cost (1915) $4 to $6; cedar poles, $5.50 to $8 each. Steel poles
nerally costing about 3¢ per lb.

s (Sec 42, Art 13). ELECTROLYTIC or aluminum-tray
commended for a-c circuits of 2 200 volts and upwards,
us operation through electric disturbances lasting 1 hr or
and similar types break down.

Table 1. Cost of Lightning Arresters

Volts	Phase	Cost	Note
2 200	Single	$2.93	
2 200	Single	2.93	Graded shunt resistance
2 200	Three	8.87	" " "
6 600	Single	54.00	" " "
6 600	Three	76.50	" " "
2 200	Three	117.00	With horn-gaps & disc switch
6 600	Three	155.25	" " " " "
13 200	Three	294.75	" " " " "

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In operation, the aluminum type demands considerable attendanc regular and frequent intervals, the interval being much shorter in warm arresters are in a warm place. Though protection of primary apparat arresters, if step-up or step-down transformers are used these largely and a cheaper form of arrester, as the horn-gap, in connection with cl substituted for the electrolytic.

Choke coils are desirable for protection of all classes of electrical lines. They should be connected between lightning arrester taps an protected, on both main and sub-station ends of a-c lines, and also in co d-c feed lines. When in series with cables, the inductance of choke c cable may produce a condition of resonance, building up very high pote with an arcing ground, as in charging an aluminum arrester.

Cost of choke coils: 100-ampere, 6 600-volt, single pole $15.75 eac 200-ampere, 13 200-volt, single pole, 26.10 "

Ground wires are an efficient protection for transmission lin are used, partially surrounding the lines, they afford complete lightning, though arresters will still be necessary to prevent tro grounds. Pole lines usually have a single ground wire, support top on a bayonet or ridge iron. On steel-pole lines, with 2 or m on pole lines in exposed locations, 2 or even 3 ground wires sho above the lines, one at top in the center and one at each side.

In moist climates or where exposed to fumes and gases, as from coke ground wires should be of copper; elsewhere they may be of galvanized s dipped, 3/8 or 7/16-in diam, and should be tied with wire of the same mate WITH STEEL. Good groundings may be made at power plants, as to wa at sub-stations and on transmission lines it is often necessary to drive ground, or to bury copper ground plates. Salt around pipes or plate sistance and keep ground moist. Galvanized iron pipes, 0.75 to 1-in d ft in earth, will have 15 to 20 ohms resistance. Where good grounds c for a considerable distance, larger and better ones should be provided each end of such section (2).

8. Underground Wiring

Underground wiring must in general be supported from n walls, be safeguarded against falls of roof, and be so installed opportunities for contact by workmen and horses.

Feeder wires and cables should be insulated, preferably varnished cloth, though for potentials under 300 volts braided be used, filled or impregnated with waterproof compound.

As all types of insulation are affected by ACID WATER (rubber least, var feeder cables must be protected from water and supported on insulator sides, as though bare. Wet-process porcelain or glass insulators mus process knobs are attacked by mine acids. Where roof is of conve height, feeders may be supported from it by expansion bolts, or by lag plugs. If roof is high or irregular, or apt to fall, wires may be carried driven into walls or ribs, or on 1.5-in iron pipe, driven into holes bor neither roof nor rib is suitable, posts may be set against rib to support while in extreme cases, with frequent falls or caves, armored cable may bottom. Feeders should preferably be located on haulage roads beh than on air courses or parallel headings, so they may be readily inspec

Alternating-current lines should be of 3-phase insulated high voltages (above 600) must be lead-covered or armored. S have the lead or armor grounded, and are usually carried on mess feeders, even when supported from roof, should be carried clos where least liable to injury from falls of roof or rock.

ing material, as pins, expansion bolts, pipe hangers, etc, suitable are shown in catalogs of Electric Service Supplies Co, Ohio Brass , and Westinghouse Co.

rd-drawn GROOVED wire has superseded round or figure-8 ears leave a smooth under-run, and a symmetrical section han figure-8 sec. It is made in oo, ooo, and oooo sizes, mployed in all but smaller installations, on account of e to wear, even where its carrying capacity is not required.

supported from roof, walls, timbers, or pipe, as conditions require, outside the gage line, on the side opposite to safety or manholes, Sec 23, Art 13). MINIMUM HEIGHT is about 6 in clearance above eral range, 32 to 56 in; max height, usually 7 ft. Where the wire rail, it must be protected at all cross-headings, on partings and en or mules are required or permitted to cross underneath or work n consists in placing the wire in an inverted trough of wood or ides project at least 2 in below wire, or by cutting a groove in the supported 2 in above roof level. Wooden troughs may be sup- r held by trolley wire suspensions, spaced not over 10 ft apart. een devised for clamping, insulating, supporting, and stretching d, as shown in catalogs of makers (see above). As much of the e is not permanently located, soldered ears, frogs, etc, are not used.

be sectionalized, so that any grounded section may be isolated and rence with other sections. It is usual to install SECTION INSULA- anel heading of the mine, their number and location being deter- ns. Where roof is very bad, it may be desirable to sectionalize branch. Jumpers are prohibited in many states, chiefly on ac- rom a loose, arcing contact. A combination of the usual section witch puts the switch on the side of the insulator, so that when forms a con- in, eliminat- ng caused by on insulator suitable for here, except on each side as this type open, except

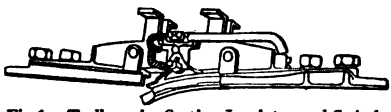

Fig 1. Trolley-wire Section Insulator and Switch (Automatic Type)

other useful cut-out switch (Fig 1) is operated by the trolley wheel, on as locomotive enters, and throwing it off as it leaves. It can not rs (Art 14) or pumps are operated from trolley beyond this switch.

nel pins, if carefully done, is satisfactory for currents up to oth rails are bonded and cross bonded.

e freshly drilled or reamed; oil should not be used in drilling, and ably be tinned. Channel pins are of soft steel, copper coated, and ole. Sizes of wire used with channel pins: o, oo, ooo, and oooo; rails is $\frac{9}{16}$, $\frac{5}{8}$, $\frac{23}{32}$, and $\frac{3}{4}$ in.

th rails above 40 lb per yard and wherever high effic is desired, OR PIN-TERMINAL BONDS are used, either concealed under the splice ONCEALED BONDS can be used satisfactorily only where the rail is ighter rail there is insufficient room between splice bar and rail to ting the bond. The copper stud of the pin-terminal bond should in rail. A small hole is drilled in stud, and after stud is placed in riven through, expanding the stud to a tight fit. Following the l steel pin is driven and remains in the stud, maintaining contact compressed-terminal bond, the stud should be expanded in the ulic compressor, and enough pressure applied to cause the copper ake intimate contact with the rail metal. A head is formed on aling it against moisture and maintaining contact of the b-

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The compressed-terminal bond is the most efficient, with lower resist
long life, but the pin-terminal type is more easily applied and with int
gives excellent results in mining work. EARTH RESISTANCE in coal
usually very low, because of acid water, and in many mines a large pa

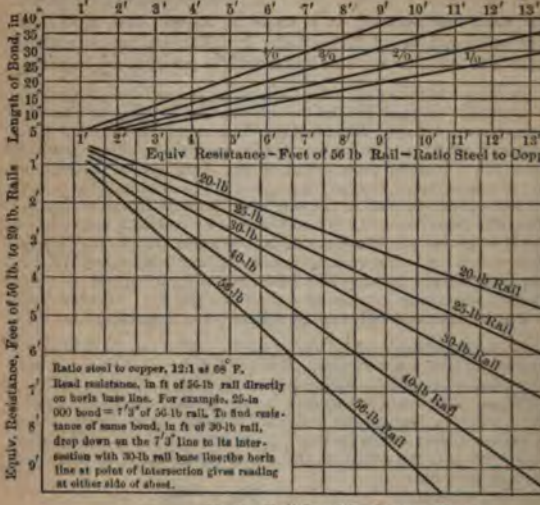
Length of Bond, in
Equiv. Resistance, Feet of 50 lb. to 80 lb. Rails
Equiv Resistance—Feet of 56 lb Rail—Ratio Steel to Copp
20-lb
25-lb
30-lb
40-lb
56-lb
20-lb Rail
25-lb Rail
30-lb Rail
40-lb Rail
56-lb Rail
Ratio steel to copper, 12:1 at 68° F.
Read resistance, in ft of 56-lb rail directly on horiz base line. For example, 25-in 000 bond = 1′3″ of 56-lb rail. To find resistance of same bond, in ft of 30-lb rail, drop down on the 1′3″ line to its intersection with 30-lb rail base line; the horiz line at point of intersection gives reading at either side of sheet.

Fig 2. Rail Bond Resistance Chart

travels through the earth. This causes ELECTROLYSIS of rails, pipes reduced or eliminated by good bonding of track, with return feeders bonding track rails to air or water piping at intervals of say 500 ft.

Capacity of bonds should be 0.5 that of trolley and feeders, but circular mils per ampere should be allowed, based on aver load. Area = 0.1 the wt per yd, and conductivity is about ½ that of copper. circular mils (nearly), and a short cut from lb per yd to equivalent cir is: 1 lb per yd equals 10 000 circular mils.

Table 2. List Prices per 100 Pin-terminal Bonds (Jan

Size	Stud	20 in	22 in	24 in	26 in	28 in	30 in	32 i
0	⅝ in	$ 67.60	$ 70.60	$ 73.60	$ 76.60	$ 79.60	$82.60	$ 85
00	¾ "	77.60	81.20	84.80	88.40	92.00	95.60	99
000	¾ "	93.50	98.10	102.70	107.30	111.90	116.50	121
0000	⅞ "	106.40	111.80	117.20	122.60	128.00	133.40	138

Discount 20%. Same list per 100 bonds for compressed terminals

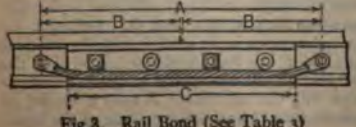

Fig 3. Rail Bond (See Table 3)

Cost of channel Table 2 shows cost of terminal bond. An 1915) of 1 525, ¾-i bonds, while mine $1 226 for material a $1.09 per bond, inclu

Standard Bond-drilling for Mine Rails (Fig 3)

tion nd es,	Size bond holes, in	Dist bet bond holes, in, *A*	Size bonds recommended		
			Capacity	Size terminal, in	Minimum length, in
	½	22⅛	No 0	½	24
	⅝	22⅛	No 00	⅝	24
	¾	26⅛	No 0000	¾	28
	¾	30⅛	No 0000	¾	32

iring must have special attention, because, on account ired, the conductors are mechanically weak and subject to

r shaft bottoms or pit mouths, it is true economy to install l-waterproof, strong and flexible, non-metallic conduit. Along here a conduit would be too expensive, rubber-insulated wire ist be supported on insulators and kept clear of coal, rock or nough it were bare, since insulation cannot be depended upon CIRCUIT should be used for telephone service, to avoid noise ver circuits, and lines must either be transposed frequently or

or these the same precautions as above should be observed es along haulage roads, potentials should not exceed 10 or 15 volts elsewhere.

sting (see Sec 12 for the general subject of hoisting)

re economical than steam, except where fuel is cheap and cost e a c is available, either purchased or produced at the mine's lectricity eliminates boiler plant and the stand-by losses attend- n. Choice of TYPE OF HOIST is largely influenced by power re- under which power is furnished. Slight changes in duty cycle requirements. A cycle consisting entirely of acceleration and asteful with any type of hoist. Fig 4 shows hoist diagrams nging from 2 to 10 sec acceleration to full speed, for a hoisting nical drum. Fig 4 shows the effect of too rapid or too slow y $\sqrt{\text{Mean}^2}$ hp ("root mean square hp") is obtained by mul- current input in each cycle period by the number of sec dura- these and dividing by total elapsed time (sec), and extracting 23).

(4) commonly used are: (*a*) INDUCTION MOTOR, geared , direct-coupled to drum, with variable voltage supply otor-generator set; (*c*) D-C DRIVE, same as above, with erator set, supplying variable d-c voltage.

cost, with effic equal to or greater than type (*b*), and is usually o 600 hp, under power conditions where heavy starting peaks lized.

tage of ease and certainty of control (5), adjustable hoisting rs. Reduced speed will reduce peaks while developing a mine, similar service. It is more expensive than type (*a*), and has g the operation of 2 separate machines, *viz*, the motor-gr

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tor set and the hoist. Motor-generator set may include an additio use, to which part of the stand-by loss may be charged.

Type (*c*) is suitable only for large hoists, where the power cont

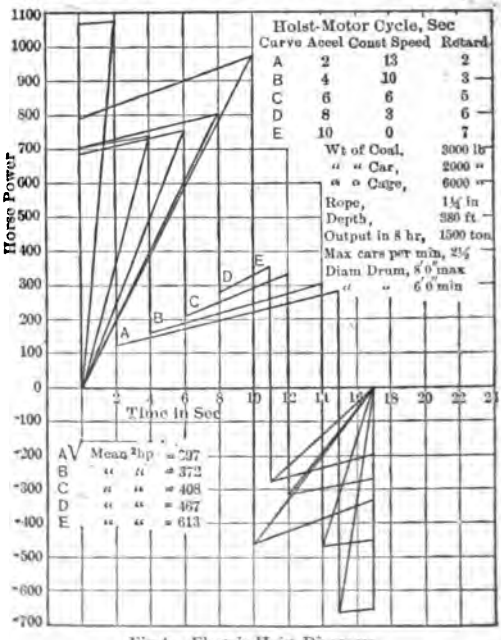
Horse Power
Time in Sec
Holst-Motor Cycle, Sec
Curve Accel Const Speed Retard
A 2 13 2
B 4 10 3
C 6 6 5
D 8 3 6
E 10 0 7
Wt of Coal, 3000 lb
" " Car, 2000 "
" " Cage, 6000 "
Rope, 1½ in
Depth, 980 ft
Output in 8 hr, 1500 ton
Max cars per min, 2½
Diam Drum, 8 0" max
" " 6 0" min
A Mean hp = 297
B " " = 372
C " " = 408
D " " = 467
E " " = 613

Fig 4. Electric Hoist Diagrams

Fig 6 is a diagram for obtaining effective wt of flywheels. The heavy lines assumes 20% drop in speed to deliver 25 000 hp-sec, w assumed as 4 ft, with set running initially at 600 rev per min, and shows that an effective wt of 39 000 lb will be required. Use of conical drums often improves the load curve by reducing the acceleration peak and compensating for wt of rope, which is a large factor in deep shafts (Sec 12). Electrical shaft hoisting is chiefly an electrical problem, and the design of drums will depend on the most economical duty cycle.

Fig 5. E A

Control of induction-motor hoists may be by liquid rheostat or magnetic contactors.

Liquid rheostat consists of a tank, filled with electrolyte (as soda or salt water) arranged to carry electrodes, usually triangular or trapezoidal. Electrodes are preferably fixed in position, and the electrolyte, circulated by a small pump, rises and falls in the tank. Electrodes are connected in the wound rotor secondary circuits, and motor torq the submerged area of electrodes. As the secondary circuit must trode tips remain submerged. The primary switch of the motor is operating a weir; the first movement of this lever closes the prim

eir controlling overflow of the electrolyte, allowing the liquid to Assuming the weir to be raised at once to its full height, the rise

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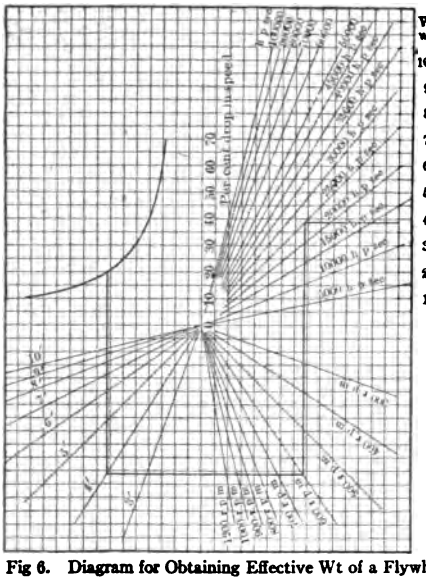

Fig 6. Diagram for Obtaining Effective Wt of a Flywheel

automatically in succession as the current falls to a predetermined
STER CONTROLLER determines the number of contactors which may close, and operator may thus start more slowly, but can not exceed the fixed limit of acceleration.

uid Rheostats

Table B

Acid wt	Resist. ohms
274	4.12
435	1.75
724	1.10
985	0.85

Control of d-c hoist motors by field control of variable-voltage generator is by rheostat in generator field, which is separately excited. Generator is of interpole type, and readily commutates current from o to max voltage, positive or negative, while running at a constant speed. Hoist-motor field is separately excited, so that its speed varies directly as impressed voltage. Same mode of control is used for IGNER SYSTEM, with addition of a slip regulator in the induction g the set to slow down when heavily loaded, thus drawing for all loads above a fixed capacity.

mple form consists of a liquid rheostat, in series with main in-y, with electrodes raised by a small induction motor in series with rrent, usually through series transformers. These electrodes are t, with normal current in main motor, they are completely sub-resistance in main-motor secondary. When line current exceeds the torque of the small regulator motor is correspondingly in-trodes and thereby increasing the secondary resistance of main and allowing flywheel to deliver energy to the d-c generator.

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Table 5. Induction-motor, Single-gear Hoists

Company	Location	Depth, ft	Material	Tons per hr	Rope speed, ft per min	Motor, hp	Speed	Control	Drum	Cycles	Volts
Anaconda Copper Co.....	Butte, Mont.....	2 000	Copper	.....	1 475	275	600	Contactor	Cyl	60	2 200
Pittsburg Coal Co.....	Montour, Pa.....	220	Coal	480	800	300	300	Contactor	Cone	60	2 300
New River Co.....	Cranberry, W Va.....	467	Coal	216	2 000	400	600	Contactor	Cone	60	2 300
Witherbee-Sherman Co (*a*)	Mineville, N Y.....	1 000	Iron	120	900	500	250	Liquid	Cyl	25	440
Vandalia Coal Co.....	W. Terre Haute, Ind..	280	Coal	540	1 560	500	450	Contactor	Cyl	60	2 200
Lehigh C & N Co (*b*).....	Lansford, Pa.....	900	Coal	800	.....	750	360	Liquid	Cyl	25	2 200

Table 6. Direct-current Mine Hoists

Company	Location	Depth, ft	Material	Tons per hr	Rope speed	Fly-wheel, tons	Hoist motor, hp	Speed	Motor-gen set, hp	Speed	Cycles	Volts
Hecla Mining Co (*c*).....	Burke, Ida.....	1 200	Lead	150	2 000	14.5	375	60	420–450	720	60	2 200
Cleveland-Cliffs Iron Co (*d*).....	Negaunee, Mich...	1 000	Iron	150	1 500	12.0	500	60	350–400	690	60	2 300
Christopher Coal Co.....	Christopher, Ill...	687	Coal	643	2 400	11.3	1 150	109	650–880	600	60	2 300
Va-Pocahontas Coal Co.....	Coalwood, W Va ..	650	Coal	500	2 600	15.0	1 100	73	700–1 000	500	60	2 200
Peabody Coal Co.....	Peabody, Ill.....	682	Coal	874	1 950	16.5	1 300	57	950–1 000	500	60	2 300

rol is most readily applied to d-c hoists with field control, on of the controller the speed is practically constant, even It follows that an interlock may be arranged, which will in a definite relation to the position of the cage in the shaft. ration and retardation may be automatic, and an overwind in case of overbalance or rapid retardation, current will be e. With INDUCTION-MOTOR HOIST, the speed at all resistance the load.

ng, some centrifugal device should be used, in connection with a the circuit and apply the brakes, if the speed at certain points is not the operator approaches the landing at proper speed, the device will e proper speed is exceeded at any one of a number of points, the cur- d brakes applied.

is applied in all electric hoists by electromagnetic attachment, when rrupted. A small switch or push button may also be arranged at cer- ings or dump, whereby the cager can immediately stop the hoist in an oisting men, with field control, a fixed resistance may be cut into ting the max speed to ½ or ⅓ the hoisting speed. This may be r at bottom of the shaft, thus obviating accidents through misun- s.

articularly adapted to electric drive, which eliminates losses of the , while lowering unbalanced, waiting for trips or on account of acci- m torque for starting, and maintains uniform speed with varying grades). Motor-driven hoists may be located at any desirable point t might not be economical or possible to operate otherwise. Practi- have either single or double reduction gearing; the former is pre- nce the perfection of the double-helical gear.

e hoists: Pennsylvania Coal Co, Dunmore, Pa, has three 300-hp nction-motor hoists; Woodward Iron Co, Woodward, Ala, has one ingle-reduction, induction-motor hoists; Keystone Coal & Coke Co, one 700-hp single-reduction Wuest gear, induction motor, liquid l Lack & W Coal Co, Nanticoke, Pa, has one 600-hp single-reduc- , liquid rheostat.

ts are largely employed in local dips in collieries. With rope pull d with 3-hp motor, these have been standardized by Pneumelectric e, N Y, and by the Lidgerwood Co, N Y; price, less than $300. hoist may serve 2 or more dip headings or rooms. Typical application ndber, Pa, where over 100 of them are used in 6 large mines.

10. Rope Haulage and Conveyers

l-rope haulages (Sec 11, Art 17) are readily adapted for ugh endless rope, with rope speeds of about 3 to 4 miles per ear reduction. The greater flexibility of locomotive haulage e haulages except where grades are very heavy. In such tor drive permits location of haulage engines at any point, und.

eyers, and elevators are almost always motor driven (see

notor must be selected to suit the character of work; whether induc- pound or series-wound d-c motors are chosen, depends on the speed tics of load. Enclosed and ventilated motors should be selected for Since the motors are run continuously without attendance, full over- otection and preferably current-limit starters should be provided. retarding conveyers, power is required for starting, after which In such case, with a shunt-wound motor suitably connected, power e line. Frequently it is desirable to install emergency control

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that in case of trouble the conveyer or car-haul may be instantly stopped f
number of points. A small switch short-circuits the trip coil on a no-volt
ing circuit breaker. The motor thus stopped can not be started until this

11. Electric Locomotives (For "Underground Haulage,"

Standard mine locomotive (Fig 7) has a heavy steel frame, on
over axles, gear-driven by series-wound motors, which are su
from axles and partly from frame. A rheostatic control, usua
paral
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ance
start
in e
Han
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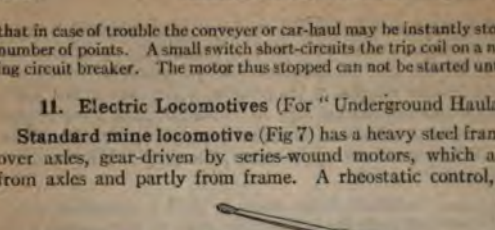

Fig 7. Electric Mine Locomotive

Details. FRAMES may be of C I, riveted steel plates, solid steel slabs,
type. Riveted-plate and slab frames are unbreakable, as in collision.
frame has the advantage of affording ready access to brakes, sand ri
apparatus, and gives best ventilation. Bearings, for journals, axle
motors, and armature shafts, must be carefully designed to exclude du
to give good service with minimum attention. JOURNAL BEARINGS are
removable when weight is taken off, and lubricated by waste-packed oi
BEARING LININGS may be of bronze or babbit, lubricated in the same
BEARINGS are preferably BALL BEARINGS, though bronze-lined waste-pa
also successful. It is highly important that the wear on these bearings
cause wear shifts the air gap, affects commutation and gear mesh, and
armature by rubbing the pole faces. Well-designed and constructed b
eliminated such troubles almost completely on small locomotives, a
standard practice on large ones of 75 hp and over. The limited spac
locomotives makes ball bearings desirable. MOTORS for mine locomoti
in general design from standard railway motors, though because of t
used, the coils, commutators, and brushes are often heavier for the
should be form-wound, interchangeable, impregnated, and baked, makin
waterproof and increasing the conduction of heat from the copper cond
narrow track gages require very compact motors, making it increasingly
sufficient heat-radiating surface. GOOD COMMUTATION is difficult to obt
rough track, many frogs and switches, and severe peak loads, both on
on heavy grades. COMMUTATING-POLE MOTORS are very advantageous,
ing due to overloads and minimizing effect of jar and vibration on
brushes. They also have a higher continuous rating, due to lower ma
armature core and pole face (8). Where gear faces are reduced on v
special heat-treated alloy steels may be used, and would doubtless be
standard practice. Tool-steel gears and pinions, costing about 1.5 to 2
ordinary forged steel, are guaranteed for 3.5 times greater mileage.

Controllers are of the magnetic blowout type, with separate li
ders, for speed control and for reversing.

Most controllers have both series and parallel connections on reverse c
connection is of no special value except for very light work, as motors wi
factorily in series unless mechanically connected, as with side-rods (9).
lers use double contacts, with separate blowout magnets for each resistance

Resistances for speed control are practically all of the C-I grid typ
alloy ribbon, packed with asbestos, is still used on some small sizes.

ced. Resistances should be placed at the ends of locomotives, d, and not above or near motors or cables.

tion of electric locomotives depends primarily on motor also on type of wheel, wt of rail, amount of sand used, etc.

Maximum available for starting depends on the wt of locome cases, with sharp sand, is as high as 50% of the wt; . Without sand a COEFFIC OF ADHESION of 25% may be heels, or 20% for chilled C-I wheels.

ion is 25%, and motors are available which will exert this drawbar hr with a temp rise of 75° C. This so-called RAILWAY RATING is h of haul is short and grades not excessive, so that sand is unnecesand approximates 10 hp per ton, at 8 miles per hr. On long h adverse grades requiring much sand and where motors can not the standard 1-hr capacity will not suffice. In such cases, the ous service is specified, and should not be less than 6 hp per ton.

Locomotive Drawbar Pull and Haulage Capacities

tives with steel wheels; for C-I wheels, take 80% of values in table.

ull and gross train load (tons). Coeffic of friction, 30 lb per vel track, and 20 lb per ton for each per cent of grade

rade	2% grade		3% grade		4% grade		5% grade		6% grade	
Haulage capac, tons	Drawbar pull, lb	Haulage capac, tons	Drawbar pull, lb	Haulage capac, tons	Drawbar pull, lb	Haulage capac, tons	Drawbar pull, lb	Haulage capac, tons	Drawbar, pull, lb	Haulage capac, tons
29	1 380	20	1 320	15	1 260	12	1 200	9	1 140	8
39	1 840	26	1 760	20	1 680	15	1 600	12	1 520	10
48	2 300	33	2 200	24	2 100	19	2 000	15	1 900	13
58	2 760	39	2 640	29	2 520	23	2 400	18	2 280	15
67	3 220	46	3 080	34	2 940	27	2 800	22	2 660	18
77	3 680	53	3 520	39	3 360	30	3 200	25	3 040	20
96	4 600	66	4 400	49	4 200	38	4 000	31	3 800	26
126	5 980	85	5 720	63	5 460	50	5 200	40	4 940	33
144	6 900	99	6 600	73	6 300	57	6 000	46	5 700	38
163	7 820	112	7 480	83	7 140	65	6 800	52	6 460	43
192	9 200	132	8 800	98	8 400	76	8 000	62	7 600	51
240	11 500	164	11 000	122	10 500	96	10 000	77	9 500	64

nclosed, series motors, as used on locomotives, ranges between 35 ing. FORCED VENTILATION, by fan blades on armature shaft or the casing, increases this by 60 to 85%. Chief objection is that ay be drawn or forced into the motor windings, but, by taking in g it at bottom of motor casing, this difficulty is largely avoided. with narrow-gage tracks and limited height of gangways, ventiry, and experiments made on the above lines show reduction in

8. Forced Ventilation for Haulage Motors

ength aul, ft	Temp, deg C, without fan		Temp, deg C, with fan		Mine temp	Remarks
	Com	Wind'gs	Com	Wind'gs		
2 000	111	104	76	72	14°	Air inlet at gear end
5 000	124	106	64	70	14°	Air inlet over commutat

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Use of sand increases the tractive resistance of mine car smallest possible quantity of the best quality obtainable should being developed to feed automatically a small stream of sand, of 50% in amount of sand used, with considerable decrease in in haulage speed. It is a small motor-driven air pump, whic through a nozzle placed in the sand-pipe outlet.

Cost of locomotives of same wt and rating appears to of the builders, based on pooling of patents (Table 9).

Table 9. Standard Mine Haulage Locomotives

Wt, tons	Min gage	No of motors	Min gage outside wheel	Price	Wt, tons	Min gage
5	24	2	35·5	$2 030	17	35·5
7·5	35	"	42	2 475	20	35·5
10	35·5	"	36	2 970	25	36
13	35·5	"	42	3 620	30	36
15	35·5	"	"	3 805	35	36

Standard equipment includes single-end control, 1 trolley, c incandescent headlights, with wheels inside frame. 3-motor l with wheels outside frame. All prices include freight east of Miss headlights, $40 ea; rolled-steel wheels, 5-ton loco, $40, 7.5-ton $60 to $80.

Gathering locomotives are of the same general type locomotives, with the addition of a trailing-cable or rope, motive to enter rooms or headings where no trolley wire

These locomotives require much less head room than mules, men will gather from 2 to 5 times as many cars as 1 mule with dr greatest where grades are worst. Where the seam is only 3 to great, because for mule haulage the top must be taken down head room. Gathering locomotives should handle 85 to 150 ca where several cars may be placed at once in each room. The h so that cars are delivered and removed regularly; it is usually n ings in pairs so that on one trip empties may be placed in one an the other, rather than to handle empties and loaded cars in the

Cable-reel type has a 2-conductor twin or concentric trailin trolley and to rail at the room neck, and paid out as the locomoti the cable as reeled out lying on the floor. A single-conductor with the room track forming the return circuit. On returning, on the reel, a tension of 10 to 20 lb being maintained to prevent the cable. In some types the reel is driven by a friction clutch but a later form drives the reel by a small series-wound motor tinuous connection across the line, maintaining uniform tensio length. Cables may be overrun and injured by the friction-clu cause wheels to skid, or by the motor type should the current b be used for pushing empty cars or pulling loaded cars up grade

Traction-cable or crab type carries a small and very compact eration, the "spragger" or trip-rider pulls the wire rope into the motorman winds up rope, pulling car out to locomotive on head cars may thus be hauled up grades too steep for locomotives. Th grades are against loaded cars only; in special cases, locomotive

Note (for following table). — Standard equipment includes 2 cast-chilled wheels, and 1 trolley. Cable reel carries 500 ft sing Crab reel carries 300 ft wire rope, and is driven by separate 7.5-hp on crab reel, add $125 to crab and combination type prices. for 3 to 6-ton locos, $40 per set, for 7 and 8-ton, $60 per set; slee $60, for 7 and 8-ton, $80.

ering Locomotives, Sizes and Prices (Apr 1, 1916)

ype	Price	Wt, tons	Min gage of wheels, in: In-side	Min gage of wheels, in: Out-side	Type	Price
-reel	$1 940	6	24	35.5	Crab	$2 650
	2 070	7	35	42	"	2 925
	2 230	8	"	"	"	3 110
	2 400	4	24	35.5	Combin	2 520
	2 675	5	"	"	"	2 680
	2 860	6	"	"	"	2 850
b	2 290	7	35	42	"	3 125
	2 320	8	"	"	"	3 310
	2 480					

motives, for light main-line haulage and for gathering umber of sizes and types, and, while largely in the experi- teed for specific installations. Battery equipment forms otive wt, and the frame, motors, and other parts are as ith strength. High-speed automobile-type motors are uble-reduction or worm gearing, the voltage used rang- a few at 150 volts and higher.

al types, the kaline nickel- lead battery, Co, and the have monopo- ch has advan- tions, but in 30% lighter, the Ironclad fficient and is

ry capacity Fig 8 con- locomotive; e against the h grade with ased on fric- per ton on each per cent tive and mo- l to be 60%. l for calculat- kind of duty; umber of cars tance, and wt wt of locomo- nt to haul the es, is selected from Table 11, which gives max drawbar pull ns) for a given wt of locomotive equipped with steel wheels. n and resistance of 20 lb per ton for each per cent of grade.

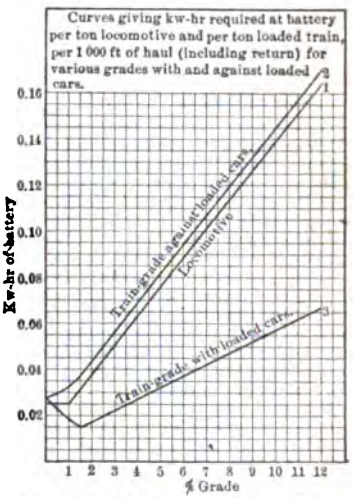
Curves giving kw-hr required at battery per ton locomotive and per ton loaded train, per 1 000 ft of haul (including return) for various grades with and against loaded cars.
Kw-hr at battery
0.16
0.14
0.12
0.10
0.08
0.06
0.04
0.02
Train grade against loaded cars
Locomotive
Train grade with loaded cars
1 2 3 4 5 6 7 8 9 10 11 12
% Grade

Fig 8. Battery Capacity for Storage-Battery Locomotive

ry capacity for a gathering locomotive. It is con- ngth of haul into 3 parts: distance from side track to nce on entry in front of rooms; max depth of room (total ion to allow for switching, etc).

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Table 11. Haulage Capacity of Storage-battery

Grade	Level		1%		2%		3%		
Loco wt, tons	Drawbar pull, lb	Haulage capac, tons	Drawbar pull, lb	Haulage capac, tons	Drawbar pull, lb	Haulage capac, tons	Drawbar pull, lb	Haulage capac, tons	
3	1 500	50	1 440	29	1 380	20	1 320	15	
4	2 000	70	1 920	39	1 840	26	1 760	20	
5	2 500	84	2 400	48	2 300	33	2 200	24	
6	3 000	100	2 880	58	2 760	39	2 640	29	
7	3 500	117	3 360	67	3 220	46	3 080	34	
8	4 000	133	3 840	77	3 680	53	3 520	39	

Example. Assume the locomotive must gather 120 cars in 8 hr, to the face. Trains are of 10 cars, making total number of trip 5 200 lb; length of haul from side track to first room, 600 ft; dep of haul on entry in front of rooms, 400 ft. Rooms are level an against the load. Distance the locomotive travels, and distan different parts of a trip, are given in tabular form:

	Equiv length of haul, ft	
	Loco	Loaded train
Side track to first room....	600	600
On entry in front of rooms.	400	400÷2=200
In rooms.....	10 × 250 = 2 500	
Loaded train weighs	$\frac{5200 \times 10}{2000}$ = 26 tons	

Kw-hr required for each part of haul over which LOCOMOTIVE t 1 000) × loco wt × kw-hr (read from curve). Per trip for level 5 × 0.025 = 0.312. Per trip for 1.5% grade, (1 000 ÷ 1 000) ×

Kw-hr required for each part of haul over which TRAIN travels × tons of loaded train × kw-hr (read from curve). Per trip on le 26 × 0.026 = 0.17. Per trip on 1.5% grade, (800 ÷ 1 000) × 26 × 0 per trip: locomotive, 0.467; train, 0.952; or 1.419. Hence, tota

To allow for decreased capacity of battery when discharging s in gathering service, switching extra cars, pulling empties and locomotives from charging station to working place, and running kw-hr obtained by calculation is multiplied by 1.35, to give the capacity at normal discharge rate, which in this case is 23 kw-hr. Consult Table 12 for battery capacities.

Table 12. Kw Dis
(Normal batt lo

Edison	
Type	Kw-h
63 A–4	11.
63 A–6	16.
63 A–8	22.
126 A–4	22.

Note that for above example the nearest battery capacity with least number of cells is 63 A–8 Edison, or 48 MV 15 Ironclad. Either will need at least 1.5-hr boosting charge, whenever the battery has been in use for 4 hr, in which case Edison battery will have 15% greater capacity, and Ironclad 25% greater capacity than given in table. Example shows how Fig 8 is used when calculating battery capacity required for gathering, and it is evident that this method can be used for any kind of installation on any kind of haul.

Successful installations of storage-battery locomotives have be Mining Co, Terre Haute, Ind; Bunsen Coal Co, Clinton, Ind; Pitt Pa; Consolidation Coal Co, Fairmont, W Va; Sterling Coal C n Mining Co, Kellogg, Id; Anaconda Copper Mini

tric Pumps (For "Mine Drainage," see Sec 13)

s best adapted for motor drive are the triplex single-acting, g, quintuplex single-acting and triplex double-acting. Single- er types, or duplex single-acting, may be motor-driven, but should be smoothed out by large air chambers on discharge t should be low, or vacuum chambers used on suction line.

r pumps (Sec 13, 40) are driven through single or double reduction s, pinion shaft should have an outboard bearing, or preferably be ft and provided with independent bearings to eliminate chattering oid, cloth, or rawhide pinions should be used on small pumps with s, as these reduce noise and vibration and eliminate crystallization

red pumps should be 85 to 90%, not including motor, but it uced by tight packing, worn gears and improper lubrication.

turbine pumps (Sec 13, 40) are driven direct from motors, exible couplings, the latter being used only where units are e.

high rotative speeds, bearings must usually be of self-oiling type, notors, and though pumps are preferably designed for hydraulic ust bearings should be provided, of ball or marine type. When nature must be in electrical balance under load, otherwise end thrust st bearing. EFFIC of well-designed centrifugal pumps ranges from pumps, 70 to 80% (10). SUCTION LIFTS for centrifugal and turbine eed 20 ft for best results. These pumps are most suitable for single rawing water from a number of points, with long and branching r plunger types are necessary unless lines can be kept tight and all

ng pumps usually require constant speed; hence, a-c induc- r compound types are suitable. Induction motors are prefer- r large main-station service. On account of the general use it is convenient to use self-starting compound-wound d-c mps, and in case of interruption by overload or short circuit rt without attendance, when current is restored. COSTS of suitable for pumping service, are given in Tables 13, 14 and

ction motors for centrifugal and turbine pumps may be by COMPEN- starting torque will not be high, and, unless motors are of large size starting current should not be objectionable. For larger motors, h a number of starting taps, connected through a DRUM-TYPE CON- or, wound rotor-type motor may be used, with starting resistance in gh for underground use the SQUIRREL-CAGE induction motor is prefer- plunger pumps, with gear drive, compensators may be used on small ticularly if cloth or rawhide pinions and flexible couplings are used for ears at starting. For larger pumps and high press, some MULTIPLE- , with a by-pass valve in discharge, to relieve press when starting.

und-wound and with interpoles, may be thrown in circuit without 5 hp, in which case the starting torque corresponds to that of squirrel- , and strain on gears and pumps is very great. SERIES CURRENT- commended for sizes larger than 20 hp. These are made in sizes and 200 hp, 500 volts, and are especially applicable to operation in r SWITCHES; these switches may also be used with self-starting apacity, and in larger capacities operate the pilot switch only for the it starter. Use of float switch with series current-limit control r attendance (except for occasional inspection), and insures proper arting current can not exceed a predetermined amount, required to d.

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Acid mine water (Sec 13, Art 15) is very destructive
water velocity is high, due to combination of erosion and
are most serious in centrifugal and turbine pumps. Cen
anti-acid bronze, composed approx of 75% copper, 15%
good results in waters up to 30 gr free acid per gal. Sha
protected and bushed to avoid electrolytic action. Tu
RECOMMENDED for acid water, as the thin edges of diffus
highest water velocity, are especially liable to destru
plunger pumps are best protected against acid water by
lead-covered plungers, though C-I plungers well greased

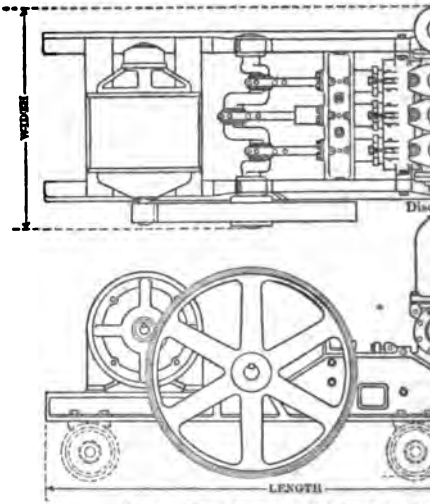

Fig 9. Truck-mounted Triplex Mine Pu

APPROXIMATE DIMENSIONS

Size of pump, 5 by 6 in	Depth stuffing box, 5 in	P
Capacity, gal per min, 150	Crank pin diam, 3 in	B
Maximum lift, 100 ft	Wrist pin diam, 1¼ in	L
Rev per min of pump, 98	Length connecting rod, 15 in	W
Motor hp, 7½	Crankshaft diam, 3 in	S
Motor, rev per min, 1150	Size main journals, 2½ by 4 in	D
Size packing, ½ in	Gear, 38 in diam, 3 in face	T

Special portable pump, truck-mounted for mine servi

Fig 9 is a triplex plunger pump, on the C-I base of which wh
running on mine track. It is useful in advancing headings an
water from a number of sumps where accumulations of a day
hr or less. It has a capacity of 150 gal per min against heads up
with motor, $650 (Jan, 1916).

Fig 10 is a standard type of small gathering pump, 3 by 6-in
min, against 100 ft head, with 3-hp self-starting motor, self-conta
per min: motor, 1 250; pump, 60. It is designed for acid wat
cyl lining, and extra large valve area and lift for passing small c
complete with motor (Jan, 1916).

c pumping installation:

o, Iron Mountain, Mich. 2 3-stage turbine pumps, each 3 000 driven by 450-hp, 2 200-volt, 60-cycle induction motor, at 1 200 c., 78.5%. Also 2 6-stage pumps, 1 000 ft head, driven by 1 050-

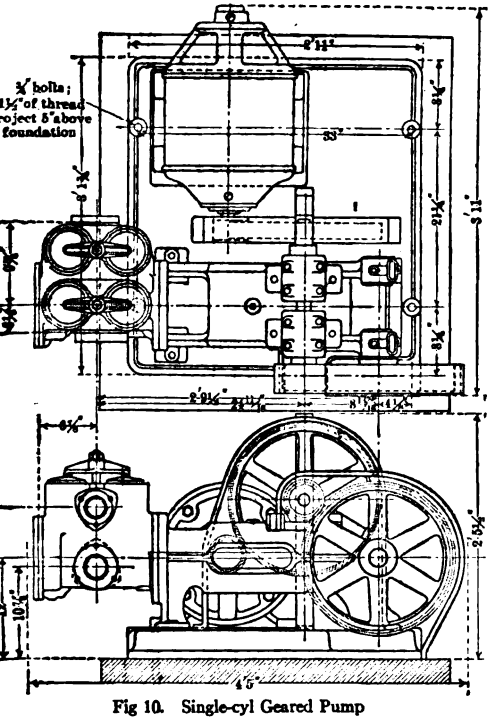

Fig 10. Single-cyl Geared Pump

Coal Co, Scranton, Pa. Besides a 4 000-gal per min motor-re are two 5 000-gal per min 6-stage centrifugal pumps, 500 ft head, o-volt, 720-rev per min, induction motors, handling acid water. Mining Co, Windber, Pa. 3 2-stage centrifugal pumps, 4 000 gal iven by 500-hp, 440-volt induction motors, at 720 rev per min, hese pumps raise 80 to 120 million gal per month, at total cost of ling power, labor, and repairs.

e for Ventilating Fans (See also Sec 14, Art 12–16)

rive may be either a c or d c. A-c motors are generally if no d c is available without conversion from a c. Full-otor is 88 to 93%, if connected to primary lines, and 85 to are required. It is simple, rugged, eliminates brush and (except where variable speed is wanted), and first cost is tors have effic of 88 to 91%, if d-c service is available with-erter or motor-generator; but if a rotary converter is used %, and if a motor-generator, only 75%. They are well

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Table 13. Cons

35 to 50 hp, 25 cycles, 3 and 2-phase								75 to 200 h
Hp	Speed sync, rpm	Volts	Motor with compensator, pulley and base	Ship'g wt, lb approx (complete)	Pulley dimensions, in			Hp	Speed sync rpm	
					Diam	Width	Bore			
35	750	220–440–550	$460	2300	13	10	2 3/8	75	750	220
		2200	581	2450	13	10	2 3/8			
35	500	220–440–550	560	2680	13	13	2 3/4			
		2200	686	3330	15	15	3 1/4	75	500	220
35	375	220–440–550	700	3800	16	17	3 1/2			
50	1500	220–440–550	Δ 563	2440	Δ	Δ	...			
		2200	Δ 669	2590	Δ	Δ	...	75	375	220
50	750	220–440–550	580	2680	13	13	2 3/4			
		2200	686	28,30	13	13	2 3/4			
50	500	220–440–550	700	4200	15	15	3 1/4	75	300	220
		2200	826	4360	16	17	3 1/2			
50	375	220–440–550	850	4490	21	19	4			
		2200	1014	4640	21	19	4	100	750	220
50	300	220–440–550	970	5140	24	21	4 1/2			
		2200	1114	5295	24	21	4 1/2	100	500	220
								100	375	220
								100	300	220
								150	750	44
								150	500	44
								150	375	44
								200	750	44
								200	500	440
								For gearing, u		

Δ Owing to high speed, 50-hp, 1 500-rev per min motor, is listed without base or pulley, and should not be furnished for belt drive.

For gearing, use slow-speed motors, with extended shafts and outboard bearings.

adapted to variable speed requirements, and are co during mine development, where frequent speed chang

A-c motor types comprise: squirrel-cage induction synchronous; three-phase commutator.

Squirrel-cage induction motor has no moving contacts, r or 550 volts in all sizes, for 2 200 volts from 20 hp up, and for but usually these limits are doubled. Starting torque and curre load torque and current. In large sizes this heavy starting c is objectionable, and wound-rotor induction motors should 2-speed motors are economical where fan may be run at half sp

Wound-rotor induction motor has an insulated phase-wo use of resistance, usually external, during starting or for sp Starting characteristics are good, the line current usually not

t or maintain power factor;
RTING CHARACTERISTICS, and
They are special for every

ed for VARIABLE SPEED, give
e no external accessories, and
rush stud per phase per pole,
s for commutator and brush
r is 35% in excess of wound-
ade in sizes from 50 to 250 hp.
with or without interpoles.
ecause of wide variations in

tors (Jan 1, 1916)
ng poles)

Shunt-wound

Shipg wt, lb, motor with rheo, pulley and base	Standard pulley dimensions, in		
	Diam	Width	Bore
3900	14	11	2¾
3900	14	11	2¾
3900	14	11	2¾
5920	16	16	3¾
5920	16	16	3¾
5920	16	16	3¾
4300	15	13	3¼
4300	15	13	3¼
4300	15	13	3¼
7355	17	19	4¼
7355	17	19	4¼
7355	17	19	4¼
6100	16	16	3¾
6100	16	16	3¾
6100	16	16	3¾
9230	20	23	5
9230	20	23	5
9230	20	23	5
7360	17	19	4¼
7360	17	19	4¼
7360	17	19	4¼
1300	21	24	5
1300	21	24	5
1300	21	24	5
9230	20	23	5
9230	20	23	5
9230	20	23	5
1300	21	24	5
1300	21	24	5
1300	21	24	5

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motor direct-coupled to a 3-phase commutator motor, the forn
to the supply mains, the latter being supplied with secondary
main motor. In this way, energy is returned to the line inste
heat in a secondary rheostat. With 40% speed reduction, for
to the fan would be 21.6% of that at rated speed, and the powe
14.4% being returned to regulating set. The relative effic is 70
with rheostatic control. Cost of the motor with regulating s
motor alone, but at 1¢ per kw-hr the excess cost would be sav

Costs of variable-speed induction motors are given in Table

Table 14. Variable-speed Induction Motors, Sizes and
(Continuous service. 75 to 200 hp. Slip-ri

25 cycles, 3 and 2-phase								60 cycl		
			Retail prices							
				Motor, with controlling apparatus, pulley and base						
Hp	Speed sync. rpm	Volts	Motor only	3-phase	2-phase	3 or 2-phase	Ship'g wt, lb (approx) motor with cont., res pulley & base	Hp	Speed sync. rpm	Volts
			$	$	$	$				
75	750	220	800	1118	1127		5020	75	900	220
		440–550	800	1046	1055		5020			440–55
		2200	890			1123	5020			220
75	500	220	919	1239	1248		6000	75	720	220
		440–550	919	1167	1176		6000			440–55
		2200	950			1185	6000			220
75	375	220					6050	75	600	220
		440–550					6050			440–55
		2200					7520			220
75	300	220					7520	75	514	220
		440–550					7520			440–55
		2200					7520			220
100	750	220–440–550	912	1246	1255		6200	100	900	220–440–55
		2200	1034			1449	6200	100	720	220–440–55
100	500	220–440–550	1136	1483	1492		7030			220
		2200	1186			1614	7030	100	600	220–440–55
100	375	220–440–55					8200			220
		2200					8200	100	450	220–440–55
100	300	220–440–550					8650			220
		2200					8650	150	900	220
150	750	220–440–550	1200	1605	1614		7380	150	720	440–55
		2200	1340			1826	7380			220
150	500	220–440–550	1353	1761	1770		8550	150	600	440–55
		2200	1425			1914	8550			220
150	375	220–440–550					9880	150	450	440–55
		2200					9880			220
150	300	220–440–550						200	600	440–55
		2200								220
200	750	220–440–550	1480	1965	1974		9070	200	450	440–55
		2200	1600			2166	9070			220
200	500	220–440–550	1534	2038	2047		10300			
		2200	1714			2280	10300			
200	375	220–440–550								
		2200								

* Price is for
motor is not re
Price does not in

e particularly desirable to correct or maintain power factor; t high in first cost, have POOR STARTING CHARACTERISTICS, and not susceptible of speed control. They are special for every s are available.

or motors are especially designed for VARIABLE SPEED, give l effic at wide range of speed; have no external accessories, and ifting brushes. They have one brush stud per phase per pole, and are comparable to d-c motors for commutator and brush notors for small clearances. COST is 35% in excess of wound- or same speed range. They are made in sizes from 50 to 250 hp.

series, shunt, and compound, with or without interpoles.

n used for fan drives, chiefly because of wide variations in age.

ct-current Constant Speed Motors (Jan 1, 1916)
ous service, 230-volt, commutating poles)

pound	55 to 175 hp. Shunt-wound								
	Contin'ous rating		Price, shunt-wound motor with rheo, pulley and base	Addition for compound winding	Ship'g wt, lb, motor with rheo, pulley and base	Standard pulley dimensions, in		
Ship'g wt, lb, motor with rheo, pulley and base	Hp	Rated speed, rev per min				Diam	Width	Bore
390	55	1 250	$590	$16	3900	14	11	2¾
390		1 000	605	16	3900	14	11	2¾
525		875	655	18	3900	14	11	2¾
525	65	780	804	23	5920	16	16	3¾
525		625	879	24	5920	16	16	3¾
525		550	949	26	5920	16	16	3¾
565	70	1 200	659	18	4300	15	13	3¼
565		950	714	19	4300	15	13	3¼
690		825	779	21	4300	15	13	3¼
690								
650	80	750	923	25	7355	17	19	4¼
650		600	1013	28	7355	17	19	4¼
770		525	1103	30	7355	17	19	4¼
770	90	1 125	853	23	6100	16	16	3¾
930		900	858	23	6100	16	16	3¾
930		775	938	26	6100	16	16	3¾
930	100	725	1138	28	9230	20	23	5
930		575	1223	33	9230	20	23	5
1 160		500	1288	36	9230	20	23	5
1 160	115	1 050	917	25	7360	17	19	4¼
1 160		850	1022	28	7360	17	19	4¼
1 160		750	1097	30	7360	17	19	4¼
1 730	125	675	1207	34	11300	21	24	5
1 940		550	1387	38	11300	21	24	5
1 800		475	1527	43	11300	21	24	5
2 380								
2 020	150	1 000	1271	35	9230	20	23	5
2 970		800	1291	32	9230	20	23	5
2 400		700	1311	35	9230	20	23	5
3 780	175	925	1406	32	11300	21	24	5
		750	1476	39	11300	21	24	5
ot rec-		650	1556	41	11300	21	24	5
tion.								

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Compound-wound motors are largely used, the series windi
STARTING TORQUE is required. Speed characteristics approach
they are recommended where voltage variations are rapid and

Shunt motors run at practically UNIFORM SPEED, with varyi
with interpoles, may be used on all circuits except where volta
On account of immense inertia of large fans, with their air col
speed requires high power input, tending to cause sparking and

Costs of d-c motors of various types are given in Table 15.

Connection between motor and fan is governed by c
effic, reliability and upkeep; it may be by direct drive,

Direct drive is highest in first cost, chiefly because of special
although with some high-speed fans of large capacity, first co
that of other drives. It is most efficient and reliable, since ther
any bearing friction except that due to wt of fan and motor.
FLEXIBLE COUPLING, to eliminate trouble from settling founda
Rubber buffers, leather lacings and links, and steel wire rope
to allow relative motion between coupling discs.

Rope drive is efficient, flexible, reliable, and is coming into
large fans, say 100 to 400 hp. American or CONTINUOUS syste
and its effic is 95 to 97%. Rope speed should not exceed 5 0
small sheave should be 40 times that of the rope. Makers of r
Co, Mishawaka, Ind, and Link-Belt Co, Phila, Pa.

Belt drive is generally used for small fans, because of low fir
designed has effic of 85 to 92%. Speed ratio should not be
belt should have liberal overload capacity. Driving pulley fo
smaller than 6 in, and, for a double belt, 18 in diam (Sec 41,

Examples. Changeable pole or 2-speed induction motors a
in use at Gary, W Va, by the U S Coal and Coke Co. Large w
variable and fixed speed, rope drive and direct-connected, are us
Mining Co, Windber, Pa.

Pittsburg Coal Co, Pittsburg, Pa, has in use two 150-hp, 600-
cycle motors, with 20% speed control, with regulating set of 30
hp, 450-rev per min, 2200-volt, 60-cycle motors, 20% speed ran
sets; also three 50-hp, 900-rev per min, 440-volt, and one 100-h
volt 3-phase, 60-cycle motors, with brush shift control giving
Rochester & Pittsburg Coal & Iron Co, Punxsutawney, Pa, o
per min, 440-volt, 3-phase, 25-cycle motors, with brush shift

11. Electric-driven Coal Cutter

(For compressed-air coal cutters, see Sec

Electric coal cutters are of 3 principal types: chain
turret, or top-cutter. In all, the cutting is done by bits o
driven chain, traveling on the cutter frame at a speed o
Rotary disk coal cutters are largely used in Great Brit

Chain-breast machine is either electric or compressed-air
cutter frame, with driving motor and gearing, travels forwar
pinion feed in a stationary rectangular frame, which is held fix
roof, or props. Cutting is done at front end or head of the
around idler wheels. Cutter head is 3.5 to 4 ft wide; height o
of cut, 4 to 7 ft, according to size of machine. On completing
back and whole machine must be shifted sideways for next c
peated until entire width of face is undercut. Owing to disa
operation, chain-breast cutter has been largely superseded by

Short-wall machine has no stationary guide frame.
is a long narrow rectangle, with driving sprocket in one

ain feed, the machine pulls itself into undercut, to the r, and then travels sidewise, cutting close to the bottom ation is continuous, as no time is lost in backing out and s manual labor is involved; under normal conditions, this 100% more tonnage than the chain-breast.

r is similar to a short-wall machine mounted on a pivoted er frame to swing in a half circle. It is carried to the face ithout leaving the track. This type is used especially to te partings, or to cut them out entirely (Fig 11). It is nd can cut a large number of narrow places per shift, as ically no setting is required. Width of cut is not greater hard cutting the strain on track is great.

verned by local mining conditions. CHAIN-BREAST: for entry-driving; for very rough, rolling or soft bottom; for

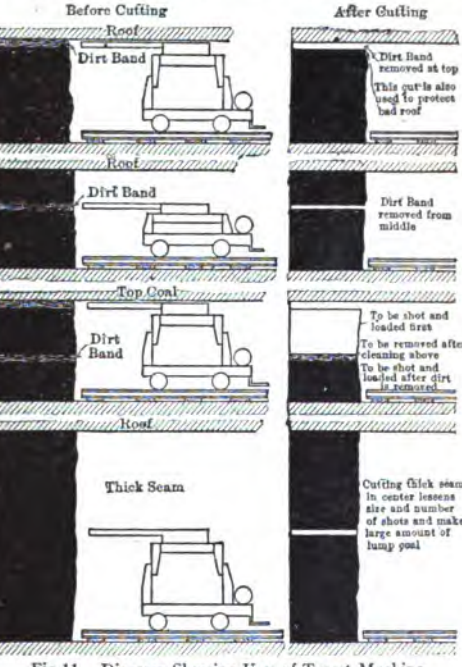
Before Cutting
After Cutting
Roof
Dirt Band
Dirt Band removed at top
This cut is also used to protect bad roof
Roof
Dirt Band
Dirt Band removed from middle
Top Coal
Dirt Band
To be shot and loaded first
To be removed after cleaning above
To be shot and loaded after dirt is removed
Roof
Thick Seam
Cutting thick seams in center lessens size and number of shots and make large amount of lump coal

Fig 11. Diagram Showing Uses of Turret Machine

dirt bands, kerf should be thicker than the band, so that the d, otherwise the machine will "climb" into softer cutting. At ing 5 to 8 in thick, about 3 ft above the bottom, is cut out with achine, keeping the coal clean, and increasing the proportion of he bone when undercutting required very heavy charges (12).

ds on character of coal and roof, as well as on thickness ule that depth of cut should not exceed thickness of se

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of machines in which the reciprocations are caused by combi
and springs driven by a small electric motor (Box, Siemens-Hal

Later makes are Pneumelectric (Sec 15, Art 24), which is
small self-contained motor-driven compressor and air-operat
(cost, $750) and the Fort Wayne drill (Fort Wayne Elec Wks,
The latter consists of a motor-driven helve, carrying 2 air-cu
which are thrown out by centrifugal force, and strike 1 600 bl
is rotated by gearing. The helve is revolved by a belt, with a
reducing tension and thus regulating speed, as for starting
out mounting, 450 lb. Cost, $750.

16. Mine Lighting by Electricity

Lighting of mine buildings, as offices, stores, shops, and dw
at low potentials, preferably not over 125 volts, on account of
and greater safety. This involves use of a-c, with distribution at
of 2 200, and step-down transformers of suitable sizes at cente

Large buildings, as stores, should use 3-wire system at 110 to 220 vo
costs. Dwellings may be grouped in blocks of 10 to 20 or more for
transformer. TUNGSTEN LAMPS are recommended, as giving better
half the current used by carbon lamps (Sec 42, Art 15).

Underground lighting system is commonly taken from mine
in which case 110 to 125-volt lamps should be used, connecte
quired. CARBON LAMPS, rated at about 3.6 watts per c p, giv
the fluctuating voltage of mine circuits.

Storage-battery miner's lamps have been widely introd
since 1914; on Jan 1, 1916, over 15 000 were in use.

Lead-battery lamps are made by the Witherbee Igniter Co, Springf
Electric Co, Schenectady, N Y; Mannesman Light Co of America,
nickel-iron alkaline battery lamp by the Edison Co, East Orange, N J
safe for use in gaseous mines, since the circuit opens if lens or bulb i
much better light than any type of Davy safety lamp, and, as the lam
miner's cap, leave both hands free for work, while directing the light on
reflector angle is 130°, illuminating a circle 7 ft diam at 3 ft distanc
battery and lamp is about 3.75 lb, of which 3.5 lb is carried on the mine
COST OF LAMP complete is $7 to $8.50 each, according to type. Charg
racks, meters, and accessories, cost $3 to $4 per lamp capacity. To
installation ranges from $10 to $12 per lamp, in quantities of 250 or

Lamps may be charged in groups of 80 or 90, from 250-volt mine cir
able rheostats. Small motor-generator charging sets, of 1 kw capacit
stalled, cost from $300, for 3-kw, to $500 for 6-kw capacity. COST o
labor, repairs, and charging current, should not exceed 4¢ per 12-hr
charge to miner per lamp is 5¢ per shift, leaving a balance to offset
and to cover interest on investment. Large installations have been m
Reading Co, Pottsville, Pa, Keystone Coal & Coke Co, Greensburg,
Co, Pittsburg, Pa, and Rocky Mountain Fuel Co, Newcastle, Cal.

17. Miscellaneous Power

For application of electric power to crushers, air compress
and screens, jigs and other apparatus in concentrating plants a
shop machinery and similar work, the a-c induction motor is pr
except where variable speed is required. Induction motors
less care and attention in dusty or damp locations than any
higher aver effic.

Direct drive, through flexible coupling, is especially desirable for
crushers, on account of the rapid change in load causing belts, chains,

ain alinement. EXAMPLES: Large crusher motor drives, 300 and
e crushers (Sec 35, Art 11) and 0.25-in screen, are used by the
ary, W Va. Compressor motors, direct-connected or belted, of
by the Rochester & Pittsburg Coal & Irón Co, near Indiana, Pa,
, with synchronous motors, direct-connected, by the Anacónda
e, Mont.

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SECTION 17

SURVEYING

BY

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BOSTON, MASS.

SURVEYING AND DRAFTING INSTRUMENTS

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AND DRAFTING INSTRUMENTS

1. Tapes and Rods

commonly 50, 100, 200, 300, or 500 ft long. The 50 and ally thin steel ribbons; the longer tapes are thicker and ability of kinking and to stand rougher usage. The lighter throughout in feet, tenths, and hundredths; the heavier are otched sleeves, etchings, or brass rivets at every 1-, 5-, 10-, st foot on each end being often divided into tenths. When s alone are marked, the last 5, 10, or 20 ft is usually graduated e 200 or 300-ft tapes are made 201 ft or 301 ft long, the addi- aduated into tenths. In most lighter tapes the zero-point some at the end of the steel ribbon, while in others it is 0.2 The heavier tapes are equipped with handle for each end, being marked. Bronze tapes are manufactured for use in mines. Metric tapes are in lengths of 10, 15, 20, 25, 30, hem are graduated in feet and inches on reverse side. The ight metric tapes are 15 and 30 m, the heavier, 25 or 50 m pes are carried on reels; tape reel should be of ample size, tape fills a larger reel than a clean one, and long tapes used at mines are also often used underground, where they are nd dirty.

th thermometer for temperature corrections, and with spring bal- ring amount of pull, are used chiefly in city surveying. Broken y riveting on back of tape a piece of old tape of same width; com- s are sold for this purpose. Tapes graduated in feet and inches building construction.

tapes are common; the former stretch readily and are useless llic tape is of cloth with fine brass wires woven into it to prevent usually graduated into feet, tenths, and half-tenths, and made in ; sometimes with feet divided into inches. A 50-ft metallic tape h as 0.5 ft, owing to rough usage and alternate wetting and drying; only for rough work.

-ft tape and box is from $5 to $15. The heavier tapes, exclusive per 100 ft, and can be obtained up to 1 000 ft; the ordinary reels 50 to $15; certain special reels for mine surveying cost as high as o ft long cost $2 to $3. Bronze tapes are a little more expensive es.

from 3 ft up are useful in the field, for accurate measuring when y at the 5-, 10-, or 20-ft points, are used.

rements recorded in tenths and hundredths of a foot into inches wing equivalents may be used:

oot = 0.01	0.08	0.17	0.25	0.50	0.75
= 1/8 −	1 −	2 +	3	6	9

morized, decimals of a foot may be quickly transposed into inches, 0.75 ft − 0.03 ft = 9 in − 3/8 in = 8 5/8 in.

trument attached to a wheel of a vehicle, to record number of wheel in traversing between two points; revolutions × circum-

trument to record the distance traversed by a person carrying it; erent lengths of pace.

easuring distance (see Art 22).

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Leveling rods are of two types, TARGET and SELF-REAI rodman, latter by levelman. Commonest forms of targe the New York, and the Philadelphia; the Philadelphia self-reading rod (Fig 1). Most rods are made of two they may be extended. The target of Boston rod is pe one strip of the rod; this requires that for readings highe rod the rod must be inverted, which introduces an erro so that the target is not in correct place with reference scale. The other target rods are always used right-side

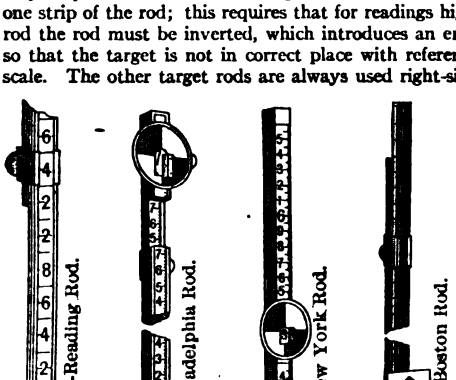


Fig 1. Leveling Rods

that it can be read through telescope by the levelman of rod are usually of a definite height (0.06 or 0.08 ft) 0.01 or 0.02 ft, thus aiding levelman to estimate hundredths of a foot or even closer. Self-reading rod for ordinary work has many advantages over target rod. A rod equipped with clamps having screw-heads, which lie close to rod and are protected by a metal guard, is preferable to a rod with outstanding screws, especially if rod is to be shipped from place to place. Ordinary leveling rods cost $9 to $16.

Stadia (or telemeter) rods are usually of soft pine, with a face 3 to 4 in wide, on which distinctive diagrams are painted, to be clearly distinguished at distances of 1 000 ft or more. For stadia sights not greater than 300 ft, the ordinary Philadelphia rod is suitable; for sights up to about 600 ft, either the first or second rod shown in Fig 2 is suitable; for longer sights the third rod in Fig 2 is best. Many surveyors make their own stadia rods, because those sold have too narrow a face, and graduations, and are not marked with clear, distinct

tion, stadia rods are frequently in two parts, hinged at hinged, it is well to provide a hard-wood cleat for back of ed joint. In Fig 2 two of the rods have cross-hairs marked . they are read (Arts 22 and 23).

2. Verniers

determining fractional part of smallest division of a scale it can be estimated by eye. The simplest form (used) is shown in Fig 3, where one unit on rod is divided The entire length of vernier equals space occupied by 9 hen the vernier length be subdivided ne division on vernier will be 9/10 as n scale. If the index (zero-point of 3 on the scale, then distance *ab* must e divisions and *cd* is 3/10 of a scale be raised until *a* comes opposite *b*, 01; if *c* comes opposite *d*, the reading case of a decimal vernier, the number which coincides with any line on scale aths of the smallest division on scale oint lies beyond the last preceding ample, reading at right of Fig 3 is 326.

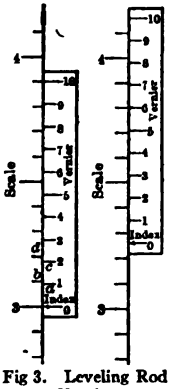
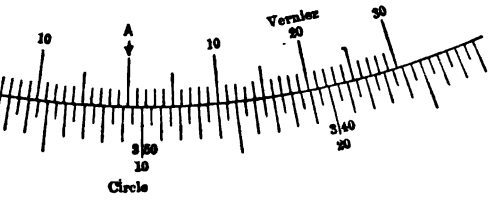


Fig 3. Leveling Rod Vernier

e usually double, one on each side of les can be measured either clockwise Against the horizontal circle of ordiusually four verniers, in pairs (Art 4). has its horizontal circle divided in e are 30 divisions on the vernier; a circle divided to 20 min, with 40 divi-

ec instrument has circle divided to 20 min, with 60 divisions sec instrument has circle divided to 10 min, with 60 divi-

min instrument is constructed by dividing into 30 equal ied by 29 half-degrees on circle; the difference between and one of vernier is therefore 1 min of arc. A pair of ructed on both sides of a zero-point (or index). If the



4. Horizontal Circle, with Double Vernier

n placed so that their common zero-point coincides with say 0° point on circle, then the 30-min line on each ide with the 14° 30′ line on circle, and the first lines on vernier will fail to coincide with the corresponding lines e second line on either side of the zero of the vernier will

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fail to coincide with its nearest mark on circle by 2 min, a
vernier be moved so that the next line to its zero-point coi
line on circle, one vernier will read 1 min and the other 29
will be either 0° 01′ or 359° 59′, depending upon whether
or counter-clockwise. Therefore, to read an angle on a
division on circle beyond which vernier index has passe
vernier in same direction to the first line which coincides
and add the number of this line to circle reading. Thu
1-min horizontal circle with double vernier, which reads (
the vernier that moves) 350° 44′ clockwise, or 9° 16′ coun

Single verniers are used with the horizontal circle of some
double vernier would be in the way of the telescope standards;
transit a single vernier corresponds in length to 20° of arc.
zero-points, one at each end, and is numbered to read from eit
angle one or other vernier index-point is set at 0° of the circle, d
tion in which angle is to be measured.

Folded vernier is a short, single vernier, sometimes used
transits and plane tables. It is designed to shorten the single ve
length, so that it can be placed between legs of the standards.
middle division; the numbering goes from middle to left end,
right end and runs to middle. Similar numbers run in reve
middle and running to right. In reading such a vernier, if a c
between middle and farther end, return to the other end of vern
direction, toward the center, until coincidence is found.

Retrograde verniers are those in which vernier divisions are
ions. They are found only on old surveying instruments.
described above, a vernier division is always smaller than a
called DIRECT VERNIERS.

3. The Compass

Surveyor's compass is used to determine the MAGNE
that is, the horizontal angle between the line and a magnet
is balanced on a pivot in center of a circular box, the ed
graduated in half-degrees and is numbered from 0° to
from N and S ends of compass-box; the needle swings
plane. Attached to the compass-box, in exact line with
two vertical sights (sometimes a telescope) used for po
The compass is connected to a tripod by ball-and-socket j
after instrument has been made horizontal by two spirit le
The frame has a spindle which fits into the ball-and-socket jo
ment has been leveled it can be swung around in a hori
counterweight is attached to S end of needle (in the no
prevent needle from dipping and thus rubbing against gla
For compasses used in mines, and their limitations, see S

To use the compass, first set it up at the point, level it, le
pivot, then point the sights along the line of which the bearin
needle points north, and the box turns under it, the letters E an
monly reversed from their natural position so that the readin
proper quadrant directly. The BEARING of a line, sometimes calle
is the angle it makes with needle when looking along line AB
bearing of same line, looking from B toward A, is called the REVE
the bearing of a line, always read N end of the needle if N e
the station of which the bearing is desired; when S end of bo
read S end of needle (the one to which counterbalance is attach
sphere). If wrong end of needle be read, the bearing noted wi
For difference between MAGNETIC and TRUE BEARINGS see Art 1

n not in use, needle should be kept raised from pivot by the screw that purpose, to avoid dulling pivot-point. Before reading bear- has been let down on its pivot, and tap the glass lightly over end s not touching under side of glass. If needle appears to cling to the robably become electrified; this can be corrected by placing a glass. The clinging of the needle to the glass may also be due to the compass, to loss of magnetism, or to shifting of counterweight. care that no iron or steel is near instrument, to attract needle from pe, pins, ax, pocket-knife, or the iron wire in a stiff hat are sources of ts affect needle so seriously that a compass is of little use in cities, or haulage ways, for obtaining even an approximate magnetic merid- have a movable inner circle, on which the declination can be laid off; hat this circle is set at 0° if declination is not to be laid off (Art 10). take both forward and reverse bearings of all important lines, be- dle to be deflected from magnetic meridian by presence of ferrugi- on material; this deflection is called LOCAL ATTRACTION. If the rom Sta A, and the bearing of *BA*, from Sta B, do not agree, local ther *A* or *B*. To determine at which station it is present, take mpass at *B*, and then with compass at *C* determine bearing of *CB*. raction is indicated at *A* but not at *B*.

mpass. (*a*) The bubbles are adjusted in same manner as those a transit (Art 4). (*b*) Straightening needle and centering pivot- ich is detected by noting whether S end reads 180° from N end; e due to a bent needle or a bent pivot-point, or both. A bend in d by reading S end of needle when N end points at some selected eading N end when S end points at same graduation. Bend the where metal is soft until ends read the same in either position r reading 180° apart at either trial). The pivot may be centered e position of maximum difference between end readings in different ding pivot until any two end readings agree. (*c*) Re-magnetizing it with a bar magnet from middle outward to ends, applying N S end of needle, and *vice versa*.

ss is a hand instrument for obtaining roughly the bearing of a inds, PLAIN and PRISMATIC. Former is much like surveyor's com- ally has no sights. In prismatic compass the graduations, instead ox, may be on a card fastened to needle (like a mariner's compass) This compass is generally provided with sights; bearing is read or folding mirror, at same instant that compass is sighted along es for surveying range in price from $10 to $60, the latter being ructed compasses, to carry in pocket for use with a tripod. The mpass costs from $25 to $75, including tripod.

4. Transit and its Adjustments

it (sectional view, Fig 5) has two spindles, one fitting in- h of which is fastened a horizontal circular plate; the outer lower plate, on which is graduated circle, and inner spindle to which the verniers and standards holding telescope are on of these horizontal plates is controlled by clamps and upper plate are two spirit-levels; the plates are leveled by ling screws, working between base and lower plate. Tran- bubble and vertical arc are called ENGINEER'S (OR SUR- a plain transit lacks these attachments.

compound objective, and an eye-piece composed of 4 lenses, recting transit, two lenses if inverting; an inverting instru- tter illuminated field. Between objective and eye piece is eld in its concentric position by four opposing screws tapped ds being outside telescope tube.

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Special attachments are: (*a*) Dark eye-piece for observing sun. nal or prismatic eye-piece for sighting high altitudes. (*c*) Reflector a telescope for illuminating cross-hairs when working in dark. (*d*) screw for measuring rates of grade directly. (*e*) Stadia hairs, which extra horizontal cross-hairs used in measuring distances by stadia. attachment for determining meridian by observation on the sun.

Precautions in care of transit. Neither tripod legs nor their shoes should to become loose. In taking instrument out of its box, always lift it by placing

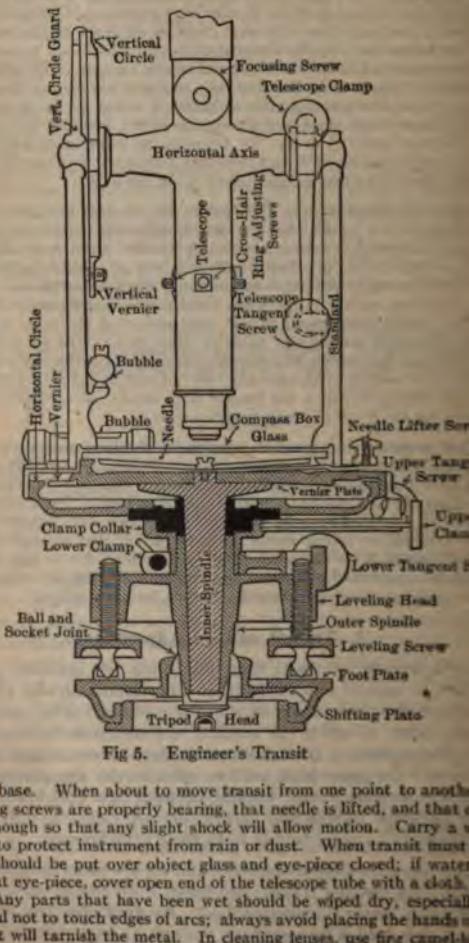

Fig 5. Engineer's Transit

beneath leveling base. When about to move transit from one point to anoth that the 4 leveling screws are properly bearing, that needle is lifted, and that set just firmly enough so that any slight shock will allow motion. Carry a bag at all times to protect instrument from rain or dust. When transit must in rain, the cap should be put over object glass and eye-piece closed; if wate telescope, take out eye-piece, cover open end of the telescope tube with a cloth it to dry out. Any parts that have been wet should be wiped dry, especial arc, but be careful not to touch edges of arcs; always avoid placing the hands graduations, as it will tarnish the metal. In cleaning lenses, use fine camel-h for screw threads, use stiff tooth brush and apply a very little watch oil afte

exposed screws. Carry transit in a box when traveling on a g, always enclose transit in its case, pack paper about it, and box packed with excelsior. Cover tripod threads with wooden ing.

ie transit are: (*a*) to make plane of plate bubbles per- axis of instrument; (*b*) to make line of sight perpendicular to make horizontal axis perpendicular to vertical axis of

Bring plate bubbles to centers of respective tubes by turning 80° and correct half of apparent error by turning capstan screws each bubble independently.

First adjust vertical cross-hair to lie in plane perpendicular to and level the instrument. Sight the vertical hair on a well- plates, and move telescope up and down. If the point does not ertical cross-hair, loosen screws holding cross-hair ring, and by rew, rotate ring until above condition is fulfilled. Then tighten second part of adjustment as follows: Sight telescope at point *A* clamp vertical axis; invert telescope on its horizontal axis and me elevation as *A*, in line of sight, and about same distance from n clamp, turn horizontally and sight *A* again; clamp horizontal on horizontal axis and if line of sight falls on *B* it is in adjust- *C* in line of sight beside point *B*. Mark or note a point *D* one- veen *C* and *B*, measured from *C*. To adjust, move cross-hair ed, by loosening screw on one side of telescope and tightening has an erecting eye-piece, move ring in direction *D* to *C*; if an e ring in direction *C* to *D*.

rds. Set up and level the transit. Sight on some high point is. Lower the telescope and set point *B* in line of sight about everse telescope, turn instrument 180° horizontally, sight again r telescope until point *B* is visible, and mark point *C* in line of as *B*. If *C* does not fall on *B*, raise or lower adjustable end of pstan-headed screw under it. Bring adjusting screw into posi- n; otherwise the block on which horizontal axis rests may stick . Then tighten capstan screws just enough to avoid looseness

adly out of adjustment, it is better to bring it gradually whole, rather than to attempt to adjust completely one the adjustment of one part will not disturb preceding e not subjected to strains, and instrument will remain

e "peg" adjustment, explained in Art 5).

al circle should read 0° when telescope bubble is in center of screws holding vernier and tap lightly until the zeros coincide. es what is called the INDEX ERROR in measuring vertical angles, in adjustment parallel to bubble tube.

move parallel to line of sight. Adjust line of sight as described very distant objects; then repeat same adjustment using points served in this last test, it indicates that objective slide does not sight. The adjustment is made by moving adjusting screws of to increase the error by one-quarter of observed error. Then sight again on distant points. The adjustments of objective tube, and centering the circles, are usually done by instrument

cts of errors in adjustment, so as to obtain accurate t must be used as follows. To avoid errors in plate izontally and bring bubbles half-way back to the middle

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by the leveling screws. Errors in line of sight and horizonta
sighting telescope first in its direct and then in its reversed
the mean of the results, when running lines or measurin
eccentricity are eliminated by taking mean of readings of t
and errors of graduation of the circle are nearly elimin
angle in different parts of circle or by measuring the angle
only one vernier is read in determining an angle, take
that was set at 0°.

Transits cost from $75 to $300. They weigh 5 to 15 lb, exc
weigh 7 to 12 lb. Accurate results can be obtained with the ligh
to 4½-in horizontal circle. It is advisable not to use too light a
instruments, on account of wind. Stiff-leg tripods cost $7 to $2
pods, $12 to $35.

5. The Level and its Adjustment

Engineer's level is a telescope to which a delicate sp
parallel to line of sight, so that, when bubble is in center
horizontal. The two common types are the Y- and th
both the telescope is mounted on a vertical axis, about
swing horizontally, and is leveled by 4 leveling screws.

Y-level has its spirit-level attached to telescope, which r
supports, which in turn are fastened to a horizontal bar
vertical axis. The telescope can be taken out of the Y's,
and replaced, when testing bubble for adjustment.

Dumpy-level has its vertical axis, horizontal bar, ar
scope, all cast in one piece, to which spirit-level is atta
level will stand much rougher usage than the Y-level, has
and permits fully as precise work. Practically the only a
has over dumpy-level is that the adjustment of line of sigh
conveniently tested.

Locke hand level is a metal tube with plain glass
small spirit-level on top. When looking through the t
bubble is seen in one-half of tube, being reflected by a pri
tube, the landscape is viewed. When bubble is in center,
where the horizontal line, which appears in center of bubbl
rod (Art 19).

Clinometer is similar to a hand level, except that bub
arm, free to move in a vertical plane. It is used for measuri
telescope along inclined line, moving the arc until bubble
reading angle of inclination.

Adjustments of the Y-level are: (*a*) To make horiz
horizontal and to make line of sight coincide with axis of t
allel to it; (*b*) to make line of sight and axis of bubble
make axis of bubble tube perpendicular to vertical axis.

(*a*) **Line of sight.** First make horizontal cross-hair horizont
level. This is tested after leveling, by sighting on a point and ne
hair appears to remain on the point, as telescope is slowly rotat
Adjustment is made, if necessary, by rotating the cross-hair ri
second adjustment of transit. When this is done, loosen clips
the Ys. Set a fine mark on a wall in the intersection of the cro
vertical axis. Rotate telescope 180° about its own axis. Correct
the screws of cross-hair ring, adjusting first one hair and then the

ect method. Bring the bubble to center of its tube and clamp te telescope in the Ys a few degrees on its horizontal axis; if entire error by the capstan screws at one end of bubble tube. ver a pair of leveling screws and bring bubble into center of tube; urn it end for end, and replace in Ys without jarring the instru- parent error by the vertical adjusting screw of bubble tube.

's. Level the instrument, then bring bubble exactly to middle, crews. Turn telescope 180° on its vertical axis and correct half sting screw of the Y support. Since the bubble is brought to eading of the rod, this last adjustment in no way affects accuracy venience.

lumpy-level are the same in purpose as for Y-level, but ction of instrument, done in a different order and in some cedure. They are: (*a*) to make horizontal cross-hair truly ument is level; (*b*) to make axis of bubble tube perpen- s; (*c*) to make line of sight parallel to axis of bubble.

djusted as described in paragraph *a* above.

evel the instrument and carefully center the bubble over a pair n telescope 180° horizontally and correct half apparent error in screws of level tube.

direct or "peg" method. Select points *A* and *B* 200 ft or el beside *A*, so that when a rod is held on *A* the eye-piece will k through the large end of telescope at the rod and take reading of sight. Point telescope toward *B* and read the rod in usual t bubble is in the middle of tube. Then set up level at *B* and . These observations give two independent determinations of between the two points; the true difference in elevation is the ces. (Same result can be obtained from a single setting of level, ay between *A* and *B*.) Leaving instrument at *B*, set rod at *A*, eight of instrument above *B*, plus or minus the true difference in d *B*. If line of sight is then brought to intersect the rod target, . The bubble being in center of its tube, the line of sight is now et by moving cross-hair ring by its adjusting screws. Example:

	Instrument at Sta *B*
a *A* = 4.243	Rod reading on Sta *B* = 4.618
a *B* = 5.724	Rod reading on Sta *A* = 3.177
d *B* = 1.481	Diff in elev of *B* and *A* = 1.441

es = ½ (1.481 + 1.441) = 1.461, the true difference in elevation. ow 4.618 above the station. Rod reading on Sta *A* should be o give a level sight.

may be applied to line of sight of the Y-level, except that in this as been set at correct elevation to define a level line, the line of arget by the leveling screws and then bubble is brought to mid- screws. This method is also used for adjusting the hand level. its, the adjustment may be made by moving either cross-hair tube. If cross-hair is moved, the adjustment of line of collima- bubble tube is moved, the vertical arc vernier must be adjusted.

of errors, in adjustment of line of sight of bubble tube, or of st be sure that bubble is in center of tube at instant rod is read, foresights are about equal in length.

ned for running long lines of bench levels, are usually equipped rs, an inverting telescope of high power, stadia hairs, very sensi- y a mirror simultaneously with the rod, and a slow-motion screw

s cost from $50 (architect's levels, with horizontal circle) to $150; $200 upward; hand levels about $10.

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6. Drafting Instruments

Only the usual forms of a few of the more uncommon are here described.

Planimeter is for determining area of an irregular plan is to follow perimeter of figure with the tracing point recording wheel at beginning and end of process; the gives area expressed in a certain unit.

Amsler polar planimeter, the most common form, has two and the other adjustable, the length of latter being set to give an ment, as indicated by scale marked on arm by maker. At end needle point, called the anchor point, which is pressed into the in a fixed position. The tracing point is at end of adjustab thus rests on three points, anchor point, tracer, and recordin area, press anchor point into the paper outside area to be meas that the tracer can follow around entire perimeter of area. run tracer around perimeter and note whether recording wheel over any creased or irregular part of drawing paper; if so, c to avoid it. Start tracer from a definite point in perimeter, pre the two arms will be about at right angles to each other. Read and vernier, obtaining four digits in the reading, the first fro from wheel, and the fourth from vernier. Move tracer careful starting point is reached, when the disk, wheel, and vernier sc difference of the two readings gives area in the unit at which le set. Check the area by another determination, with anchor poi so that recording wheel will travel over a different path on the there might have been some irregularity in paper, or dirt on slip and not accurately record all the path over which it trave ment recorded by planimeter is unknown, find it by runnin known area. If area is so large that anchor point can not be se it into segments, determining area of each separately; or ancho area, provided the area of a CORRECTION CIRCLE is added to re by maker. Results correct to within 1% may easily be obtai

Some planimeters are made with both arms of fixed length, t one unit. These cost about $15, whereas those with adjustable

Rolling planimeter has a tracing point at end of an adjus is fastened to a frame supported on two rollers. The whole ins and backward in a straight line, while the tracing point trav precision within 0.1% is easily reached with this instrument. made to allow for shrinkage or swelling of drawing paper w after drawing was made. These instruments cost $50 to $100.

Pantograph is an instrument for copying a plan to the same or cipally for enlarging or reducing a plan having many irregular li map with contours. It is a jointed framework of several pieces o to form a parallelogram. It rests upon three points, one of wh other two, *B* and *C*, are movable. There are other bearing point the instrument and are not essential to its principle. The two m in such positions that they will trace exactly similar figures and b points on their respective arms, the relative scales of the figures justed. The essential condition which the instrument must fulf is that *A*, *B*, and *C* shall lie in a straight line, and that each poi of three different sides (or sides produced) of the parallelogram. can be the fixed point. These instruments are usually provide indicating proper settings for various reductions or enlargement in joints, very accurate results can not, as a rule, be reached, graphs are sufficiently accurate for most topographical maps; in the use of the instrument for enlarging than for reducing. Co reliable work requires use of a pantograph costing $50 or more. ETDOGRAPH, costing about $125, is sometimes used for same pu

Drawing and Blueprint Papers

r working plans are of all grades, from manila detail
er yd, to well-seasoned, muslin-mounted paper, which is
y changes in humidity, costing $0.75 to $2.00 per yd, de-
quality. Mounted paper comes in 10-, 20-, and 30-yd
32, 58, 62, and 72 in. The best grades of drawing papers
sheets, either plain or mounted; common sizes are Demy,
, 17 by 22; Royal, 19 by 24; Super Royal, 19 by 27;
ouble Elephant, 27 by 40; Antiquarian, 31 by 53 in.

r is used largely for studies and for temporary copies
rmanent copies tracing cloth is used; there is a growing
permanent tracings. Tracing paper comes in sheets of
s mounted paper; also in 10-, 20-, and 50-yd rolls and
6, 40, 42, 48, and 54 in, costing 5 to 25¢ per yd.

uniform quality of finely woven cotton, coated with a
transparent. It is made in 24-yd rolls, in widths 30, 36,
costs 35 to 85¢ per yd. Most tracing cloth has to be
chalk or specially prepared pumice before it will take
and a rough side; most draftsmen prefer the rough side,
id pencil lines more readily, and will not show erasures
ss prints are made from it; but erasures can be more
oth side.

profile papers can be procured with colored lines, both
transparent paper or cloth; also mounted on muslin or
aper is printed in orange, green, red, and blue, in sheets
, and 40 by 50 cm, also on smaller sheets called "coordi-
ction papers are also made in 20- and 50-yd rolls with
de, and in metric units 50 cm and 75 cm wide. Profile
green, orange, and red, in three scales, called Plates
spaces horizontally to the inch and 20 vertically; *B*, 4
tically; *C*, 5 horizontally and 25 vertically. These are
20- and 50-yd rolls, with the engraving 9, 10, and 20 in
¢ per yd on paper, 40 to 60¢ per yd mounted on cloth,
tracing cloth.

most common of process papers. It is a white paper, coated
n which is sensitive to light. When fresh it has a yellowish
een, dark green, or blue color, it is old and will not give clear
rmanent blueprints are made on coated cloth. Good rag stock
han wood pulp paper. Following is a formula for the coating
rate of iron and ammonia 1 part (by weight) to 5 parts water;
of potash (recrystallized) 1 part (by weight) to 5 parts water.
in equal parts, in the dark or in subdued light, the mixture
paper by a flat brush or sponge, as uniformly as possible. The
n a dark room, and stored there until used. The above coating
drawing on tracing cloth or tracing paper for about 5 min in
x-printing paper use larger proportion of solution *a*. Slow-
prints.

llows: Dissolve ¼ oz ferrocyanide of potassium in 2 oz (½ gill)
¼ oz citrate of iron and ammonia in 2 oz water; mix as above.
per which will print through tracing cloth in about 4 min in

r and cloth is made in 10- and 50-yd rolls, in widths 24, 30,
osts 15 to 25¢ and cloth 30 to 60¢ per yd. according to width.
ose the tracing (with blueprint paper under it) in a printing
ending on sensitiveness of paper and brilliancy of light. Then

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wash paper thoroughly in water, to remove all chemicals. D wet, for wet spots cannot be eradicated from tracing cloth. I cerns engaged in making blueprints, supplying prints at from prints always shrink to a smaller size than the tracing; linea from 0.5 to 2%, depending upon quality of paper. For writing use a saturated solution of bicarbonate soda, applied with on

Brown process paper is a thin sensitized paper for makin tracing. The negative is then used to make prints having eit white ground. This double process yields prints which are sh original drawing. The process requires a bath in hyposulphite ing in water. Prints from brown process negatives can be m as black and perfect as the original drawings. Negatives cos

Direct black and white process prints give black lines on from this process do not shrink. Cost is about double the co also a direct process blackprint paper and cloth, of German m as blueprint paper. It requires washing in water only.

8. Plotting Traverses

By protractor and scale. The courses of a traver a more exact method than may be used for plotting det usually be plotted safely by protractor and scale. A necessarily close on the plan; any error of closure on error in scaling a distance or in laying off an angle. Th traverse can be computed from bearing of some other l rect, by means of the intermediate deflection angles; t

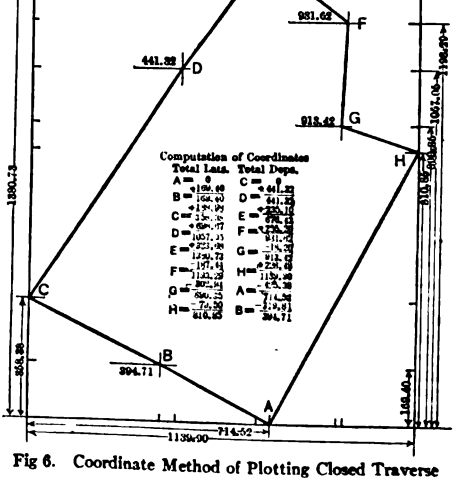

Fig 6. Coordinate Method of Plotting Closed Traverse

an east and west line through most southerly point be ch be no negative coordinates. The total latitude of any po

eceding courses, beginning with the most southerly point,
otal departure (or longitude), beginning with the most
case of a closed polygon, construct a rectangle of which
ence in latitude of most northerly and southerly points,
fference in longitude of most westerly and easterly points.
e plotted very accurately by straight-edge and reliable
by beam compass, and checked by scaling the diagonals.
off its total latitude on both the easterly and westerly
suring from southerly side; the point will lie on a line
oints. Along this line scale off from westerly side of
rture of the station, thus determining position of point.
ilarly plotted. To check a plot of this character, scale
ines. Lines running nearly parallel to the axes should be
nethod, preferably the tangent method, explained below.
tions and coordinate method of plotting the same trav-
is computed in Art 12; total latitudes and departures
mputed from balanced latitudes and departures in ex-
antages of this method are: each point is plotted inde-
plotting is done by scale only; and plotting can be readily

nate method of plotting to a traverse which does not
f coordinate lines accurately spaced and intersecting at
ese lines northward and eastward from origin of coordi-
hosen preferably at or beyond extreme southwest corner
es of the stations are computed from the unbalanced
departures, and plotted as described above.
rtant work an unclosed traverse may sometimes be
he tangent method, avoiding computation of coordinates.

ists in laying off the deflections of a traverse by plotting the
gles. First compute bearings of all the courses on the basis
n obtain algebraic sum of all the deflection angles, and add it
se; if result checks bearing of final course, the bearing compu-
plot the traverse roughly with scale and protractor, to deter-
; this is for purpose of starting first course on final drawing, at
hat the
roperly
ceed to
Draw
a line
in such
AB will
Con-
h *A* as
cb the
of *AB*,
Ab; *Ab*

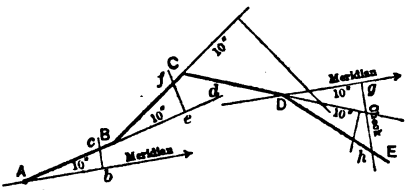

Fig 7. Tangent Method of Plotting Traverse

, say 10 in, because from this base line the entire traverse is
ugh *c* and scale off *AB*. From *Bd* erect another perpendicular
deflection angle at *B*, and then scale *BC*. Similarly plot courses
k every third or fourth course, as *DE*, by transferring the merid-
n construct a right triangle *Ddk*, scale *gh*, and compare it with
ing of *DE*. If error in bearing is 10 min or less, scale off on *gh*
earing, and draw *DE* through this new point, so that error in
e carried further along the traverse; if error found is greater
us plotting, beginning at last checked course, and find error.
e inch is convenient for constructing legs of right triangles.
h exceeds 45°, it is more accurate to plot the complement of

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deflection angle rather than angle itself, in which case a perpendi
point and used as base of right triangle.

Chord method consists in laying off deflection angles of a tr
of arcs which will subtend the angles. Fig 8 represents sam
plotted here by chord method. Draw *Ab* representing meri
radius, and scale from *b* the chord distance *bc*, which equals
half of the deflection angle. *Ae* is drawn through *c*, and ther
arc *ef*
deflect
chord
of half
f and
the a
every
DE,
throu
chor
sine
ing

Fig 8. Chord Method of Plotting Traverse

calculated bearing. If it is in error 10 min or less, correct it b
ing to calculated bearing, and draw *DE* in corrected position
10 min, re-plot and correct error. If deflection angle is gre
by a triangle and the rest by method of chords. In looking
angle, for plotting chords, the necessity for multiplying by
natural sine of half the angle directly as a chord, with a sc
the radius is measured with a 20-ft scale.

For maps of large areas, such as a state or a portio
ciently accurate to draw meridians and parallels of latitud
most common form of projection used is the POLYCONIC, in
representing the earth is developed on a series of cones t
In the U S Coast Survey Report for 1884, Appendix No
dinates of curvature for plotting parallels of latitude and
projection.

9. Finishing the Plan

Every plan should have a complete title writte
drawing is begun, so that it can readily be identifi
a meridian should be shown, and designated whet
names of abutting owners, and limits of their p
dimensions in their proper places. In case of a
expressed in square feet or acres and is lettered in
of sides are printed at middle of each line, and b
between them, should be given; also section cor
or other physical boundaries which may exist are
such as S B for stone bound, I P for iron pipe, s
It is the practice of many surveyors to omit fro
of lines, or angles; this custom can not be too se
practice to letter in some inconspicuous place, p
ence to the notebook containing survey from wh
initials of draftsman. If any leveling has bee
plan, the location and elevation of bench-marks
or plotted on map, preferably both.

On working drawings, and sometimes on
of the survey are drawn, usually as fine, colore
indicated either by very small circles, the ce
points, or by short lines drawn through the po
angles. Triangulation stations are represente

all squares, and other auxiliary stations by circles. The rks is frequently represented by a small cross and figures,

property and the physical features, such as streets or drawn as full black lines. Shore lines are drawn in black should, as a rule, be the heaviest lines on the plan, unless portant feature. Whenever colors are to be used, better obtained with water colors than with bottled inks. Red y unsatisfactory results; in a few years it spreads and leaving the tracing discolored. The following colors are rveyors: burnt sienna, raw sienna, yellow ocher, scarlet pia, cadmium yellow, chrome yellow, gamboge, Hooker's nd indigo. All of these, except gamboge, may be used on danger of running, but if process prints are to be made ssian blue and indigo will not print well. Some colors l prints may be made to do so by adding a little Chinese low to give them body, but not enough to change the ctory way to color a plan is to shade the line or areas by of tracing. Before using the color, rub places where it pencil eraser and after color has been applied rub it down e a uniform shading. On blueprints, for a red line, use for a yellow line, Schoenfeld & Co's light chrome yellow.

ing should, as a rule, read from bottom of plan, that is, hould read from left to right when the plan lies with its

ROMAN
GHIJKLMNOPQRSTU
VWXYZ&
ghijklmnopqrstuvwxyz
Reinhardt's Style
E F G H I J K L M N O P Q
S T U V W X Y Z &
fghijklmnopqrstuvwxyz
1234567890
F G H I J K L M N O P Q R S
T U V W X Y Z &
fghijklmnopqrstuvwxyz

Fig 9. Style Sheet for Lettering

ing running perpendicular to bottom should read from an or Gothic letters are appropriate on maps for pub- re a neatly finished appearance is desirable, but, since ed chiefly for utility, the Reinhardt style is applicable possibly main title of map.

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Reinhardt lettering is rapidly executed, presents a workman-like a used more than any other (Fig 9). For fine lettering, such as Roma pens are suitable. Some draftsmen prefer a little coarser pen, such as and a still coarser pen, Gillott's No 604 EF, will be found suitable for single-stroke work, as for numbering contours or for the smaller dim For still heavier lines, King's No 9 Nonpareil (similar to stub pen) is heavier lines than these, and particularly for executing the Reinhardt patent pens are good; they come in sets of graded size, costing about 9 ing holder. No 6 or 7 Paysant gives about right weight of line fo cloth, while No 4 pen will give a line heavy enough for the main line pens make lines which have somewhat rounded ends, so it is necessar when making the heavier letters, to square up the corners with an as Gillott's No 170. Paysant pens should not be cleaned with a clot taches to slot; they should be cleaned with a small piece of tough pap ball-point pens are used to some extent for single-stroke work, but they use as the Paysant pens.

Title should comprise: brief description of drawing, owner's name, lo scale, date of survey, surveyor's name and address. It should be so most important information is on the most prominent line. The lines centered, arranged in compact form, but not crowded. It is advisable most important or longest line. In other lines, when two words occur arated by a considerable space, first letter right-hand word and th word at same distance from center line of title as is the end right-han of title should conform to size of plan. Never waste time on a fancy read and as most draftsmen are not familiar with such alphabets, t to be poor. Put title in lower right-hand corner of a plan, so that turning up the corner of sheets that lie above it in a drawer. In som set up in type and printed on plan. A rolled plan or profile should b brief title lettered on back, at both ends, so that a title may be visib been rolled in either direction.

Notes are an essential part of most plans; they are usually lett style. Border lines are not necessary on construction drawings, but ings and maps a single firm line, drawn ½ to 1 in from edge of sheet, gi ance. A fine line just inside of a heavier one makes a good border line have a tendency to make border lines too heavy.

Meridian should be represented on all maps as a full arrow, if it b or as a half-arrow, if it represent the magnetic meridian, the half-arr same side as the declination. Make arrows of simple design, and no in size or breadth of lines. It is good practice to show both the true meridian, and to indicate on latter the declination existing at place a Do not let a meridian line cut any other lines of a drawing.

To clean a drawing use sponge eraser, art gum, or dry bread crum gasolene or benzene will remove pencil marks without affecting ink line life out of the tracing and makes it turn yellow with age; it also cause lect dirt more readily. A tracing upon which lettering has been sta cleaned with gasoline.

On working plans requiring accurate scaling, it is advisable to p two long scales at right angles to each other, by which to measure am or swelling of the paper, and to make allowance for it in subsequent from the plan.

Tracings from blueprints or from penciled working drawings can b them on a drawing table having a glass top, and placing table where p fall under it, or else placing electric lights under the table.

Conventional signs are used to represent topography and other A few of the most common are shown in Fig 10. They are used most maps and are produced either in black, as is custom with U S Coast vey, or in color, as on maps of U S Geol Surv. They are executed that straight lines, such as in conventional signs for marshes, railroa are usually ruled. The symbol for marshes should always be parallel

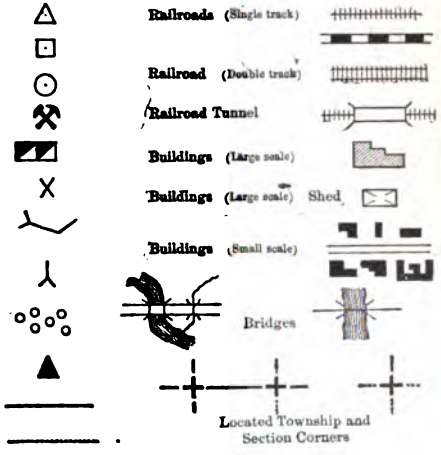
g" the first line outside shore line should be a full line about , drawn as close to the shore line as possible, and should follow larity. The next line is parallel to first, but a little lighter and nd so on; 5 to 10 lines are sufficient. Water-lining and fresh ally drawn in Prussian blue. In executing the symbol for ever-

Ledge — Evergreen Trees

Fresh Marsh

Depression Contours

Mine Dumps



ig 10. Conventional Topographic Signs

dividual symbol in lines of uniform weight, although the various es of different weights (Fig 10). To avoid tendency to make draw irregular groups of symbols in different parts of area to be etween these groups with smaller or lighter symbols. In the es it is intended to represent the plan of a tree in foliage, with a wer right-hand side. On the back of the U S Geol Surv maps nal signs used by that office.

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Contour lines are almost always drawn in burnt sienna water- Gillott No 303 or a contour pen. Every fifth or tenth contour is slightly wider and a little darker in color. In numbering contours, than are necessary to find the elevation of any contour without di be small figures in burnt sienna. A common mistake is to make co they subordinate the more important features on map.

Sub-aqueous contours are usually represented on hydrographi dash black lines, the shallowest contour having one dot between d tour in depth having two dots between dashes, and so on. In s representing fathoms of water are shown as single dots for first fa a space for second fathom, dots in groups of three for the third, and

LAND SURVEYING

10. Traverse with Compass and Tape

Surveys with compass and tape are adapted only to s sive land, in localities where there is little danger of local m Even serious local attraction may be circumvented if the surv of its occurrence, but when rocks containing magnetic iron ar erally throughout a district, it is difficult, if not impossible, t compass readings. At every station the reverse as well a should be read, to detect local attraction, and, if found, its determined and the erroneous bearings corrected (Art 3). usually read closer than ¼°.

If a compass survey were made of a square lot 1 000 ft on a side, a corresponds to reading bearings to nearest ¼°) in an angle would co of 2.2 ft on a side. In a compass survey, therefore, it is obvious measure the sides closer than to nearest half-foot or foot. Under compass surveys can not be relied upon for an accuracy exceeding 1 erly surveys were often made by compass and chain. In re-running o on transit can be used to aid in finding the physical features whic and a reliable transit and tape survey is then substituted for the ol pass survey (Art 11 and 15).

Magnetic declination is the angle which the needle makes with When necessary to determine this angle accurately, the true merid servation on the sun or on Polaris (Art 13), and then noting its c north end of needle points east of true meridian, it is called an east at a given place the declination changes continually by small amo are called VARIATIONS.

To set off the declination so that the compass will give true beari compass box toward the north. Then, if the declination is east, t to the right by the amount of the declination; if the declination is to the left. Needle readings will then be true bearings.

Isogonic lines are curves connecting points at which the magnet same. Isogonic charts are prepared by the U S Coast and Geodetic on a map the observed declination at numerous magnetic station declinations at intermediate places (see Fig 11). These charts do sults, but are useful for learning approximate declinations. Obse interpolated places are frequently quite different from those stated hence, whenever the magnetic declination is desired with precisio make observations on the true meridian (Art 13).

Changes in declination make it necessary, in re-running old lin bearings by an amount equal to the change in declination which h lines were first run. To determine declination for some past date, r Survey are useful, provided a magnetic station was situated near but it is not safe to rely on records of a magnetic station, even only 5 in a general way. It is always safer to determine the present decli

t time lines were first run, which may have been recorded with
ot then recorded, the best way to find the correction in bearing
ent magnetic bearing of any well-defined line of the old survey,
ntified stone heaps, and apply the difference between the pres-
to all original bearings (Art 15).

. Traverse with Transit and Tape

ch measurements should be made depends upon object of
y survey of valuable property, angles will be measured to
l distances to a hundredth of a foot; while in a farm sur-
minute and distances to a tenth or even to within a foot
t 22) are exact enough. Transits for general surveying read
utes. It is therefore necessary in precise surveys to measure
see end of this article), to obtain results consistent with
asurements. For better appreciation of the relation be-
ngles it should be borne in mind that 0.03 ft, at a distance
nin of angle. The degree of precision of transit and tape
in 1 500 to 1 in 40 000; to attain an error of less than 1
cial care, particularly in taping.

easurements are due to: (*a*) erroneous length of tape;
of pull on tape: (*c*) inaccurate plumbing; (*d*) incorrect
sag of tape; (*f*) changes in temperature; (*g*) effect of wind.

h can be corrected by comparing the tape with a standard;
oo-ft standard at some convenient place. To test a tape stretch
standard, and leave it until it acquires the same temperature,
ade. Test at intermediate points as well as for total length.
l should be noted; also whether it is suspended or supported.
e U S Bureau of Standards, Washington, will test a tape and
is too long, measurements made with it are recorded too short,
uld be added. It is good practice to have a new tape standard-
andards, and then keep it for the sole purpose of standardizing
A tape subjected to rough usage will soon have several repaired
errors in its length.

s a very appreciable effect upon the length of a tape; ordinary,
etch 0.01 to 0.02 ft with an increase of 10 lb above the ordinary
nt of stretch in any given tape can be readily determined by
a stout nail in the floor, and applying different tensions as de-
ance, tape lying throughout on floor. Spring-balance handles
ttached to tape in the field for measuring amount of pull.

ag is the most fruitful source of errors in taping; if plumbing is
e results can be obtained when measuring downhill than uphill.
much error is introduced by plumbing that for accurate results
inclined distance from horizontal axis of instrument to a nail
tion; read, on the vertical arc, the inclination of tape, and
ce from cosine of this angle: *hor dist* = *incl dist* × *cosine vert*
e computation may be simplified by using versed sine of angle,
— (*incl dist* × *versin vert angle*). The versed sine (1 − cos),
ber, may be taken from a table of natural functions and the
le with sufficient accuracy on a slide rule. This method re-
every other tape-length, but it is more expeditious and gives
s than plumbing. Another way is to tape directly from stake
; set up instrument at every alternate stake and measure ver-
ing stations, by sighting at a rod target set at same height
xis stands above the stake under transit; this gives inclina-
te horizontal distance by either method given above. Still
sure inclined distance from stake to stake, and obtain differ-

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ence in elevation by leveling; then compute horizontal distance by paragraph. When much inclined work is to be done, it is advisab 300-ft tape.

(*d*) **Alinement errors** are not likely to be large; lining in the enough for most measurements. If, in measuring 100-ft tape-len out of line, it will introduce an error of only 0.005 ft in that tape-l poor alinement may be computed by the formula: $c - a = h^2 \div$ taped distance, h the offset from the line, and a the correct length. the hypothenuse, a the long leg, and h the short leg of a right triang of a 100-ft tape is on line and the other 0.8 ft off line, the error in th $0.8^2 \div 200 = 0.0032$ ft. Of course there will be a similar error making a total error of 0.0064 ft in the 200 ft. The shorter the taped proportionately is the error due to faulty alinement. This form nearest 0.01 ft, if taped distance is less than 300 ft and offset does taped distance.

(*e*) **Sag.** Measurements frequently have to be taken with tape its ends. It is customary, in this case, to exert a stronger pull than act effect of sag. With a 12-lb pull, on an ordinary 100-ft ribbon at its ends, the increased tape reading due to sag is about 0.01 ft. for any particular tape by the formula: Lengthening due to sag = $($ where w = wt of tape in lb per ft of length, t = pull in lb, l = len supports in ft, and L = total length of tape in ft. The result will nary pull of 10 to 12 lb is given, the correction for sag may be a formula, though this is seldom done.

The most practical and accurate way of eliminating the error due by actual test the amount of pull necessary to apply to a suspended exactly 100 ft apart. Support plumb lines from points previously set 10 ardized tape when supported horizontally throughout, then note pu the suspended tape to be tested so that its end marks will coincide

(*f*) **Temperature correction.** A new tape is of standard leng pull of 12 lb when supported throughout its length. The average co of a steel tape is 0.0000065 per degree F. Thus, a change of 16° of a 100-ft steel tape produces about 0.01 ft error in its length. H are desired, the temperature of the tape must be noted and tempe plied. Small tape-thermometers are made for this purpose; the th in contact with tape, to obtain as closely as possible the temperature then, in sunlight, it is difficult to obtain correct temperature of tap

Cumulative and compensating errors. In all measurements portance to distinguish between errors which tend to balance and late, the latter being far more important. For those which tend to of errors which will probably remain uncompensated, according to me will be the square root of total number of opportunities for error 100-ft tape is 0.02 ft too long a cumulative error of about 1 ft will b a one-mile line. If, on other hand, tape-lengths are not marked cl total error made by this compensating error of 0.05 ft will be only $\sqrt{}$

Angle measured may be the deflection angle, the actual muth angle. The DEFLECTION angle is that between the last c the forward course; the ACTUAL ANGLE measured is usually the angles formed by two diverging lines, although, in case of many surveyors prefer to read consistently either interior The AZIMUTH angle is the angle a course makes with the me sumed) direction measured clockwise, usually from the south deflection angle the telescope has to be inverted, whence a collimation will be introduced into deflection angle; such err sated by turning the angle with the telescope first direct and taking half final reading as correct angle. Deflection R (right) or L (left), according to direction turned.

, set up the instrument at A with the vernier reading 0°, and
t telescope at magnetic south, true south, or other arbitrarily
amp the lower motion. Unclamp the upper plate, sight B, and
wise direction. This is azimuth of line AB. Take the instru-
not to let the upper plate be moved, invert telescope and back-
wer motion, reverse telescope to its direct position, loosen upper
reading is the azimuth of BC, referred to same meridian as AB.
in line of collimation becomes cumulative; the survey should
sible, by noting final azimuth of starting line AB. The differ-
d the final determination of azimuth of AB indicates at once
ular work.

e errors in azimuth readings, the magnetic bearing of each line
acticable, and compared with calculated bearing. The calcu-
is assumed to be its observed bearing; the calculated bearing
m calculated bearing of the line next preceding, combined with
een the two lines.* If a right deflection, add it to NE and SW
n NW and SE bearings; if a left deflection, reverse the process.
een 90° and 180°, take its supplement; if over 180°, subtract
d be exercised in noting quadrant in which the new line lies.
from the S direction, clockwise through 360°, azimuths between
earing of the same number of degrees; azimuths between 90°
ring equal to supplement of the azimuth; azimuths from 180°
ing equal to the azimuth minus 180°; azimuths between 270°
ng equal to 360° minus the azimuth.

tice of astronomers to assume the south point as zero azimuth;
been officially adopted by the U S Coast and Geodetic Survey,
rveyors. In general practice, some advantages are gained by
zimuths being read, as above, in clockwise direction. — ED.)

raverses. The algebraic sum of the deflection angles
t deflection being considered positive and left deflection
ior angles are read, or computed, the sum of all interior
erse should be $(n-2)\times 180°$, where n is number of lines
are read to nearest minute, each angle is probably cor-
a is the number of angles in a traverse, the allowable
$'\times\sqrt{a}$. Thus, if $a=9$, allowable error is 1 min 30 sec.
investigated before leaving the field; if angular error is
allowable, a mistake is indicated, which should be found
ethod for checking distances is explained in Art 12.

traverses. The only check on distances is re-measure-
erably in the direction opposite that first traversed. (A
can sometimes be closed and checked by the method of
e checked by determination of the true bearing (to near-
stellar observation (Art 13), of the first and any subse-
verse.

by repetition, set up transit at A, with circle reading 0°,
motion. Loosen upper clamp and sight C; read and record
plates clamped together, invert telescope, and sight B, with
p upper circle and sight C; read and record the angle. One-
k closely the first reading, but will be more exact than the first.
times, and observing final reading to nearest minute, an accu-
obtained. The method might be carried even further, starting
ther points on the circle, but ordinary transits are not reliable
sec. . The purpose of making half the repetitions with telescope
verted is to compensate errors of adjustment in line of collima-

this method can be applied as follows: The angle is first laid
set; this angle is then measured by repetition. If found to be

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in error, say 15 sec, the point is moved in proper direction a d
plying *tan* 15 sec by distance from the point to the instrumen

For determination of an area three general met
(*a*) Set transit at corners of the property and measu
measure distances along property lines. (*b*) If bounda
high fences or other obstacles which prevent setting u
but still permit distances to be measured along prope
obtained by measurement between lines staked out para
equal offsets. (*c*) If boundaries of property are such tha
to set up transit at corners, nor to measure distances dire
an independent traverse is run, regardless of boundaries.
of this traverse the property corners are connected by t
distances. In this case, a transit station should be set
erty corner to permit telescope to be focused on it (no
The property lines should also be measured, where poss

12. Computation of Areas

Adjustment of notes. First correct all slight er
they are not large enough to indicate mistakes, by alter
were taken from short sights, or those in which error is
the sum of all interior angles is $(n-2)\times 180°$. Next
course, starting from one of which the bearing is kn
correct tape measurements for erroneous length of tape
ture, if necessary. The field data are now in form for

Double meridian distance method. The data are conve
below. The LATITUDE of a course is its length times the
DEPARTURE is its length times the sine of its bearing. Latit
departures to the eastward are called positive, and southwa
departures are called negative. After these have been con
entered in their appropriate columns, find total of each colu

(**Note.** — The parallelism between other synonymous terms
which have been and are used by many surveyors, may be
westing = longitude difference = departure. Northing or
ence = latitude. In this article, the word latitude is used, n
as an abbreviation of latitude difference — ED.)

If the work were exact, the sums of the positive and of th
be equal; similarly with positive and negative departures. I
positive and negative totals are not large enough to indicate
ments or computations, errors may be distributed among th
according to following rule, provided angles have been read
to be applied to the latitude (departure) of any course is to
(departures) as the latitude (departure) of that course is to
(departures), without regard to algebraic signs. This rule a
likely to occur in tape measurements than in angles. Any k
in the field, which would lead surveyor to suspect that the err
take precedence over this rule. Furthermore, it is probabl
of tape and small obstacles on the line, recorded distances a
short; as a general rule, therefore, it is not good practice to
it would lengthen any of the distances. The algebraic sum
departures must be zero. In example given below latitudes
balanced so as not to increase length of any lines, and so as
line *HA*, which is known to have been measured under difficu

In the case of a COMPASS survey, the errors are as likely to be
hence, if nothing is suspected as to the probable locus of erro
to be proportional to lengths of courses, and the survey is bala
which alters not only the lengths of courses but also their di

departure) of any course is to total error in latitudes (depar-
re is to perimeter of traverse.
itudes and departures, next compute the double meridian
ourse. The D M D of first course equals departure of first
y other course equals the D M D of preceding course, plus
e, plus departure of course itself. The D M D of last course
l to departure of last course, but should have the opposite
signs must be carefully observed, not only in computing
computation of area. If D M D's are computed beginning
in the traverse, all D M D's will be positive. The last col-
e double areas is found by multiplying each D M D by its
lf the algebraic sum of the double areas gives area of the

Area by Double Meridian Distance Method

	Lat	Dep	Balanced Lat	Balanced Dep	D M D	Double area
2	+ 169.40	− 319.83	+169.40	−319.81	− 319.81	− 54 176
4	+ 188.98	− 394.73	+188.98	−394.71	−1034.33	−195 471
3	+ 698.67	+ 441.32	+698.67	+441.32	− 987.72	−690 088
5	+ 323.68	+ 235.10	+323.68	+235.10	− 311.30	−100 762
5	− 187.46	+ 255.20	−187.44	+255.20	+ 179.00	− 33 552
2	− 302.97	− 18.20	−302.94	− 18.20	+ 416.00	−126 023
3	− 79.50	+ 226.48	− 79.50	+226.48	+ 624.28	− 49 629
2	− 810.90	− 425.43	−810.85	−425.38	+ 425.38	−344 919
1	+1380.73	+1158.10	0.00	0.00	.....	2)1 594 620
	−1380.83	−1158.19				797 310 sq ft
	error	error				
	− 0.10	−0.09				

rk it is better to enter positive and negative latitudes and
ns, for convenience in adding.

lose agreement between the sums of plus and minus latitudes
s departures gives a check on the accuracy of multiplication
earings. The D M D's are checked as shown above, but
ible areas. The final area can be roughly checked by plani-
ng the plotted traverse into rectangles or triangles, scaling
ting their areas. A more exact check on the area is to com-
e double parallel distances (D P D), just as the D M D's
partures, and then obtain double areas by multiplying each

and *d* error in departures, the ERROR OF CLOSURE of survey
e error of closure of survey shown above is

$$\sqrt{0.09^2 + 0.10^2} = 0.135 \text{ ft},$$

sure is 0.135 ÷ 3 802, or 1 part per 28 300.
distances will be detected in the process of computation.
do not balance within reasonable limits, the error in depar-
itudes equals tangent of bearing of that line which would
on the plan, and if only one mistake has been made in
ably occurred on a line having approximately this bearing.
l minus latitudes, or departures, should be seen to be about
arture, of any one course, it is probable that the bearing of
orded in wrong quadrant.

he work of computations. One of the best is that by R. L.
ecimal places, for every minute of angle and up to 100 units

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The same table can be used for computing horizontal an
sponding to inclined tapings. Nearly every textbook of surv
pocketbook contains a traverse table computed to 15 min of
a traverse table computed to single minutes, a table of log sin

When a traverse does not follow property lines (Fig 12), areas may be computed as simple geometric figures, as in following example:

In Fig 12, ABC is property line and XYZ, 76° 24′ traverse line.

$aA = 25.2 \times \sin 76° 24' = 24.5 \quad bB = 12.7 \times \sin 56° 12' =$

$aX = 25.2 \times \cos 76° 24' = 5.9 \quad bY = 12.7 \times \cos 56° 12' =$

$ab = 5.9 + 207.6 + 7.1 = 220.6 \quad AB = 220.6 + \frac{(24.5 - 10.6)}{2 \times 220.6}$

$$\text{Area } AaBb = 220.6 \times \frac{24.5 + 10.6}{2} = 3$$

$$\text{Area } AXa = \frac{24.5 \times 5.9}{2} =$$

$$\text{Area } BYb = \frac{10.6 \times 7.1}{2} =$$

$$\text{Area } AXYB = 3$$

Another method of wider application, especially in properties, is to compute area from the COORDINATES (see Art 8, this Sec, and Art 9, Sec 18, on *Underground* nates of the corners are computed in ordinary manne distance of each from nearest, or most accessible, traver coordinates are ascertained by method previously descrit consecutively around the tract. Multiply each latitude ence between the following and the preceding longitud tracting the preceding from the following. One-half is the area.

Meandering boundaries, such as brooks, roads, et measuring perpendicular offsets from a traverse line w veniently near the boundary. If curvature of boun offsets are taken at regular intervals, but if it be errati at each point where direction changes. In either ca trapezoids are computed, and their sum added to or sub traverse lines. If offsets be taken at equal intervals, made by one of the following methods.

Trapezoidal rule. Area = $\frac{1}{2} d (h_e + 2 \Sigma h + h_e')$, w val between offsets, h_e and h_e' = end offsets of the Σh = sum of intermediate offsets. This rule assumes straight line between adjacent offsets.

Simpson's one-third rule. Area = $\frac{1}{3} d (h_e + 2 \Sigma h_o$ d = common interval between offsets; h_e and h_e' = first twice the sum of the odd offsets (3d, 5th, 7th, etc); 4 sum of the even offsets (2d, 4th, 6th, etc). To apply an odd number of offsets; if the number is even, comp trapezoid separately. This rule assumes that betwe boundary is a parabola.

In Fig 12 the area of $efZd$ by the trapezoidal rule = $\frac{1}{2} \times$ $+ 11.4 + 122) + 1.6.7] = 1\,293$ sq ft. By Simpson's rule

$+ 11.4) + 12.2] + {}^1/_2 \times 20\,(12.2 + 16.7) = 1\,286$ sq ft. Area
ng area of trapezoid $Bgfe$ and subtracting triangle BgY, giving
is computed by finding area of trapezoid $dZJC$ and subtracting
ı ft. The length of side BC can be found by computing the
y method of Art 10. For example

$$ck = 20 + \frac{(16.7 - 12.2)^2}{2 \times 20} = 20.5 \text{ ft.}$$

Determining the True Meridian

best adapted to the needs of an engineer are: (*a*) by
y transit, the bearing of sun, found from its altitude;
attachment for transit; (*c*) by observing the bearing of
t elongation. The observations consist in measuring
angle between sun or star and the survey line of which
termined; preliminary observations on other factors re-
roblem, such as the altitude of sun or star, the latitude
e usually also necessary.

ns certain times of day are more favorable than others,
ie observations should not be made at all. During sum-
sphere) observations may be made at almost any time
nd between 2 and 5 p m. Observations before 7 a m or
unsatisfactory, because the sun is so near horizon that
introduces a large error into the measured altitude of
ts, sun should be at least 10° above horizon. In mid-
be necessary to wait until nearly 9 a m before making
een 10 a m and 2 p m astronomical conditions are not
ate solution of the spherical triangle that must be em-
un's azimuth. In general, when sun is well to the east or
orable; when it is near meridian they are unfavorable.

's bearing. The only additional equipment necessary
r's transit is a colored glass over the eye-piece. Even
with in the following manner: Draw out the eye-piece
old a card behind it; if telescope be then pointed at sun,
the cross-hairs can be seen on card. The image may
ard to or from eye-piece; the cross-hairs can not usually
rojected on to the sun's disc. In computing sun's azi-
know the time within 5 min or less. If possible the
gulated to standard (railroad) time.

sts in measuring simultaneously the altitude of sun's center, and
some reference line. For a comparatively rough result, say,
in is not serious, it will be sufficient to bisect the sun's disc,
ertical and horizontal cross-hair; for more precise results, it is
er and then lower edge of sun, for altitude, and right and
ngle. The means of these pairs may be taken as altitude and
nter, as though they were both measured directly at same in-
ope is then pointed along reference line and horizontal angle
double observations on the sun with least effort, the edges to
ected that only one tangent screw will have to be turned at a
northern hemisphere) the sun is rising and moving to right.
sun appears in upper left-hand quadrant of field, the sun will
recede from the horizontal cross-hair. If transit has an invert-
will appear reversed. By setting the horizontal hair a little
disc, the sun's motion itself will bring cross-hair into contact
c; it is therefore necessary only to keep vertical cross-ha[illegible]

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contact with right edge, by means of the upper tangent scre
instant both cross-hairs are simultaneously in contact with th
position is reached the watch time should be recorded, the ve
read, and the index error of vertical circle determined. The
tion consists in bringing the disc into the lower right-hand qu
hair a little to right of left edge, and keeping the horizontal h
upper edge of disc, by the vertical tangent screw, until both
The watch time is recorded, both circles are read again, and
determined. The mean of the two vertical angles is assumed
of the two horizontal angles, and to the average of the two ti
should give sun's bearing to nearest minute of angle; for gre
may be repeated, 3 readings being made in each position of
has a complete vertical circle, so that altitudes may be read
is advisable to take the second half-set of readings with tele
thus compensating errors of adjustment (Sec 4). It is undesir
extend over more than 10 or 12 min of time, since the curv
troduce error into result. If observation be made in afternoo
in upper right-hand and in lower left-hand quadrants.

To calculate the sun's bearing, take the mean of measur
it the refraction correction in following table, and call this h i

Refraction Correction to the Altitu

(Minutes to be Subtracted)

Altitude	Refraction	Altitude	Refraction
5°	9′.9	13°	4′.1
6	8 .5	14	3 .8
7	7 .4	15	3 .6
8	6 .6	16	3 .3
9	5 .9	17	3 .1
10	5 .3	18	3 .0
11	4 .9	19	2 .8
12	4 .5	20	2 .6

Next compute sun's "declination" for the time of observatio
from the Nautical Almanac, published each year by Navy D
from the handbook of any instrument maker who publishes a
"apparent declination" of sun is given in the Almanac (Part I)
under heading "Greenwich Mean Noon"; the difference for 1
to right. By means of this hourly change, the declination for
may be computed. The number of hours since Greenwich
follows: In the Eastern time belt the instant of 7 a m is sam
Mean Noon, hence the Almanac may be considered as comp
Eastern time. For example, if observation be made at 10:20
must be increased or decreased by 3½ times the change for 1 h
sive values of declination will show whether declination is inc
Central time belt, 6 a m corresponds to Greenwich Mean
5 a m, and in Pacific belt 4 a m. If Standard time is known
will generally be close enough for purpose. The corrected dec

The latitude of the place must be obtained either from a ma
If taken from a map it must be obtained within about ½ min
is found as follows: The maximum altitude of the sun should b
transit, by setting horizontal cross-hair on lower edge of sun
it continues to rise. When sun drops below cross-hair, the alt
tical arc. This reading must be corrected for index error, refr
The semi-diameter may be taken from the Nautical Almana
the declination. Call the corrected altitude h. The latitud
equation, $L = 90° - (h - D)$. South declination must be consi

$4^h.74$ west, the observed altitude of sun's lower limb was 01′.0; refraction correction, −02′.0; sun's semi-diameter, = 25° 21′.3. On this day, the sun's declination at Green- 3″; change in 1 hr, +19″.59; hence increase in declination local noon (4.74 hr later) was 1′ 33″; and true declination la then gives latitude = 42° 20′.7.

center at instant of observation is now found by the formula:

$$\cos Z = \frac{\sin D - \sin h \sin L}{\cos h \cos L}$$

north point (either east or west).

ervation on sun, April 12, 1911.

oo′

Horiz angle	= 84° 28′	Vert angle	= 41° 30′
Horiz angle	= 86° 11′	Vert angle	= 41° 16′
Mean	= 85° 19′.5	Mean	= 41° 23′
		Refraction	= 1′.2
			h = 41° 21′.8

Standard) 55″ × 2.1^{hr} = N 8° 22′.4

= 1′.9

D = N 8° 24′.3

s	*Nat Sin*	*Log Sin*	*Log Cos*
4′.3	.14631		
1 .8		9.82009	9.87538
1 .0		9.82844	9.86867
oduct	−.44517	9.64853	9.74405
merator	−.29886		9.47547 n
4′.0			9.73142 n
6 .0			
9 .5			
6 .5 E			
6′.5			

nt to use the natural functions, Z may be found by

$$\cos \frac{Z}{2} = \sqrt{\left(\frac{\cos s \cos (s - p)}{\cos L \cos h}\right)}$$

= ½ $(L + h + p)$.

out error of the watch on standard time, local time may be

$$\sin \tfrac{1}{2} P = \sqrt{\frac{\cos s \sin (s - h)}{\cos L \sin p}}$$

n, and sec), added to the west longitude of place gives Green- sing value of declination given in Nautical Almanac, under ent Noon," the declination may be found by same process as d Time.

a small appliance, which may be attached to an en- for finding the meridian by an observation on the sun. nt forms, but the general principle is that the attach- d about a polar axis (parallel to earth's axis), so as to tion, while the whole instrument can be rotated about swing transit telescope into plane of meridian. By a motions the telescope of the attachment (or its equiv- sun. If proper settings be made for observer's lati- on, this pointing at sun can be accomplished only by into plane of meridian.

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The three steps required in the process are as follows: Fir
in such manner that the solar telescope may be pointed at
parallel to earth's axis. The method of doing this depends
ment. In the Burt solar attachment the declination is set
provided for this purpose. In the Saegmuller pattern it is
arc of transit and an attached level, the small telescope bei
telescope when declination is laid off. In other forms of atta
and Smith's, which contain mirrors, this setting is made by
upon construction of instrument. After the declination sett
of the attachment is given an inclination to horizon equal to
tion. In the Burt and Saegmuller attachments this is done

Fig 13. Half Celestial Sphere

the co-latitude of the place. In the Shattuck and Smith at
scope is placed in position by setting vertical circle to read la
these two settings have been made the solar telescope is sighte
attachments having mirrors, the mirrors are turned so that s
When sun's disc is bisected exactly by cross-hairs, the plane
coincident with plane of meridian.

The declination referred to in preceding paragraph may
Almanac for the given date and for the instant of Greenwi
change in the declination per hour is given in the next colum
cerns making solar attachments publish handbooks each y
and hourly change. To obtain correct declination take nu
Greenwich mean noon and multiply this by the "difference
added or subtracted according as declination is seen from t
creasing. The number of hours since Greenwich noon is re
Greenwich noon is the same instant of time as 7 a m Eas
Central time, 5 a m Mountain time, or 4 a m Pacific tim
Central time belt, the number of hours since 6 a m is the nu
wich noon. Declination corrected in this manner has still to
The proper correction for refraction in declination may be fo
by instrument makers, or it may be observed by Comstoc
vertical hair on one edge of sun and note the time; set vernie
note time when same edge of sun is again on the cross-hair.
of time between observations and h be the altitude in degree
equals $2\,000 \div h \times n$. Add this correction to north decli

titude used in solar observation may be taken from a reliable
y method already described.
lar attachments is that the adjustments must be frequently
r results. Direct solar observations made with any engineer's
nd consume very little more time than observations with the

construction and use of the various attachments, see cata-
rt solar attachment is manufactured by W & L E Gurley,
Sons, Denver, Col, manufacture the Shattuck attachment.
., Pa, make the Smith attachment. Bausch & Lomb Optical
e the Saegmuller Solar; and nearly all of the instrument
ments similar to the Saegmuller. Another instrument, the
Smith attachment.

is an instrument for determining direction of true me-
the sun. It consists of a horizontal plate upon which
al sights, like those of the ordinary needle compass,
which is fixed in same vertical plane as sights; the
es about a vertical axis. An arm, carrying a lens and a
the polar axis that it may be set at any inclination to
bout it.
dian, set the arm at proper inclination to polar axis by
tion on the declination arc. Next give polar axis an
tude of the place, by setting off this latitude on the
is now turned about the polar axis and the whole in-
ical axis, so that the lens throws an image of sun on
the sights are in plane of the meridian.

pole-star are not quite so conveniently made as those
ulations are simpler and accuracy is somewhat greater.
ay be identified as follows:
consisting of 7 fairly bright
with bent handle (Fig 14).
bowl farthest from handle
because a line through them,
distance between the stars,
ery near Polaris. The posi-
by noticing that Cassiopeia
le of pole-star from the dip-
small circle about the (invis-
al day, with a radius (in 1913)
ach day the star is on merid-
bove the pole, called its up-
in when below the pole, at its
extreme eastern position is
and its extreme western posi-
Observations are best made
g 14 shows constellations as
s is at its upper culmination;
nverted they are seen as they
tion; and by looking at the
margins of page, the positions
astern elongations of Polaris.

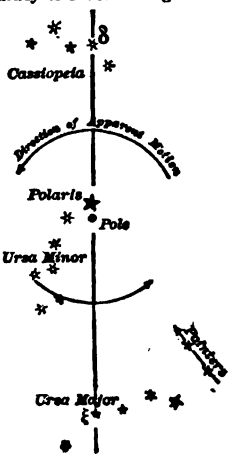


Fig 14. Circumpolar Stars

ary, in addition to the transit, consists of a lantern and
t into field of telescope and make cross-hairs visible.
w accurately the instant at which elongation will occur,
now the time within a few minutes, so as to have ample

time to prepare for the observation. The position o
give a fair idea of how long one must wait before Pola
elongation. A still closer estimate of time may be mad

Time of Elongation of Polaris

Date	Eastern elongation	Western elongation	Date	Easte g
Jan 1	$12^h\ 50^m$ p m	$12^h\ 40^m$ a m	July 1	12^h 5
Feb 1	10 48 a m	10 38 p m	Aug 1	10 5
Mch 1	8 57 "	8 47 "	Sept 1	8 5
Apr 1	6 55 "	6 45 "	Oct 1	6 5
May 1	4 57 "	4 47 "	Nov 1	4 5
June 1	2 56 "	2 46 "	Dec 1	2 5

Note. — The figures given are LOCAL TIME, which does not v
for different parts of United States. To change to standar
longitude between standard meridian and observer's meridian,
units; 1 hr corresponds to 15° of longitude. Add this to loc
standard meridian, and *vice versa*. The standard meridians

EXAMPLE. Mean local time, Feb 1 = $10^h\ 38^m$ p m, at a
The Central standard time is therefore $10^h\ 42^m$ p m. To fin
than first of month, interpolate in table. The daily change i

A half hour or so before time of elongation, transit should
by vertical cross-hair. As star slowly moves right or left tow
it is followed by the lower tangent screw. At eastern elonga
western elongation it will be setting. When extreme positio
be transferred to a stake on the ground, or an angle may be m
For greater accuracy, a second observation may be made in
telescope in reversed position, leveling plate bubble again b
of the two results will compensate errors of adjustment. It
bearing of line marked out, or the bearing of star at elongati
formula: *sin* azimuth = *sin* polar distance × *sec* latitude.
line, this angle must be laid off to right for western elon
elongation. This may be done by transit, or by a perpendi
tape.

The LATITUDE required in formula need not be known with g
taken from a map if it can be found within 1 or 2 min of true
served directly, following method may be used. If the altitu
when it is above the pole (upper culmination), and its maxim
this will be meridian altitude. The latitude is then compute
The measured value of altitude h must be corrected for refra
error of vertical arc. The POLAR DISTANCE p is found by lookin
in Nautical Almanac, in table headed "Circumpolar Stars"
it from 90°. If the star be observed at lower culmination, t
added to altitude.

Example of observation for latitude. Observed altitud
mination = 43° 37′ 00″, index error = + 30″; refraction corr
altitude = 43° 36′ 29″. Since declination of star was 88°
1° 15′ 25″; hence latitude = 42° 21′ 04″.

Example of observation for meridian. The direction of Po
ern elongation Apr 6, 1904, in latitude 42° 21′ N. Declination
The azimuth is calculated as follows:

$$\log \sin p = 8.32267$$
$$\log \sec L = 0.13133$$
$$\log \sin Z = \overline{8}.45400 \qquad Z = 1° 37′ 47$$

f to right, to fix a point on meridian 310 ft N of transit. The
ws:

log 310.0 = 3.49136
° 37′ 47″.9 = 8.45417
0.94553 Offset = 8.821 ft.

otained with greater accuracy by this method than by observa-
t instrument is used in both direct and reversed positions and
ach pointing on star.

bstacles and Inaccessible Distances

n necessary when obstacles occur on a traverse line. If
e AB (Fig 15), neither end being visible from the other,
ediate point
are visible, a
run. ‖ Meas-
nce CB and
B by trigo-
of Art 10, if

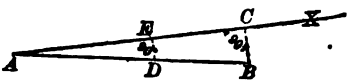


Fig 15. Running Random Line

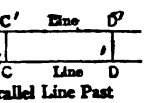
g), and set any desired point D on AB as follows: $AE = ED = BC \times (AD + AB)$. By this method points on AB ning out and actually measuring line AB. *Tan* $CAB =$

X can not be run, a traverse from A to B can be run in which ing side; its direction and length can then be computed by . Any point D can be set on AB by computing its latitude ing them with latitude and longitude of nearest transit point H s follows: the difference in their longitudes divided by differ- s tangent of bearing of line HD, and length

$$HD = \sqrt{\text{diff lat}^2 + \text{diff dep}^2}.$$

pointing in direction of calculated bearing, and laying off dis- e accurately set on AB. Since this traverse method involves will not lend itself so readily to accurate results as the case lom line is straight. In running any straight line, when neces- by reversing telescope, the mean result of a double reversal he telescope being first erect and then inverted, to compensate

ne past obstacle. One of the most exact methods of ne past an obstacle of limited size is to run a parallel tacle. The instrument is set at B (Fig 16) and BB' is BB' is made any convenient distance which will bring nd obstacle. Similarly point A' is set opposite point A; rt the transit need not be used for laying off right angle at A. Points A and B must have been accurately set on line. The instrument is then set at B', backsighted on A', the telescope inverted, and points C' and D' set on line. Leaving telescope inverted, another backsight is taken on



allel Line Past
cle

ed to compensate instrumental errors; points C' and D' these two lines if any errors are evident. Transit is then ngle turned off, and point C set, the distance $C'C$ being y then setting at C and sighting ahead on D ($DD' = CC$)′, in run forward in its original direction. The distance

$B'C'$ is measured carefully, to give distance BC; this i
lines BB' and $C'C$ shall be laid off at right angles by t

The other offsets AA' and DD' are not in any way connect
the line; they simply define its direction; hence, if convenien
to show these distances as swing offsets for transitman to
DD' should be taken 200 ft or more from obstacle, if practic
Should obstacle be in a hollow, so that it is possible to se
A, the point D, or a foresight of some sort, should be set on
used as foresight when transit returns to original line. The
as above, by an offset line around obstacle. Sometimes it i
actly on line on ridgepole of a building (if that is the obstac
backsight for extending line on other side.

Equilateral triangle traverse. In Fig 17, set inst
set stake D a sufficient distance away, measure CD.
60°, and set point E, making $DE = CD$. Set up at E,
line EB. Evidently $CE = CD = DE$.

To pass obstructed portions of main traverse an aux
deflection angles may be run which will lie near the for

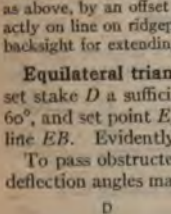


Fig 17. Triangular Traverse Past an Obstacle

angles are small the distanc
traverse can be applied as th
on main traverse, from point o
main traverse is again reache
where an isosceles triangle is f
as the base. Suppose at A on
of 0° 24′ R is taken to B on
421.7 ft); at B deflection 0° 4
and BC made 421.7 ft. The
and by laying off at C deflectio
erse line is resumed. This method can be extended to c
les triangles, the bases of which are either main traverse

To measure an inaccessible distance, where line
a pond, any of following methods may be employed:

(*a*) Lay off from transit line AC (Fig 18) a line AB,
and can be taped, and set stake B; then lay off ABC
intersecting main transit line. Measure angle CAB
$AC = AB \div \cos CAB$; or better, $AC = AB + AB$ ex
distance CB can be computed, and checked against th

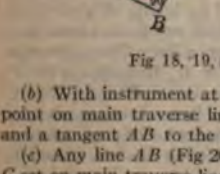


Fig 18, 19, and 20. Measuring Inaccessibl

(*b*) With instrument at A (Fig 19) and a swing offs
point on main traverse line on other side of pond) m
and a tangent AB to the swing offset. $AC = 100 \div \sin$

(*c*) Any line AB (Fig 20) may be laid off along sho
C set on main traverse line across the river. Measure
C as a check, and from AB as a base compute AC by
made perpendicular to CA and some multiple of 100
simplified.

between two inaccessible points *A* and *B* (Fig 21), two accessible points *C* and *D*, measure *DC* and angles d *BCD*. Compute *CB* in triangle *ACD*, and then in triangle *ACB*

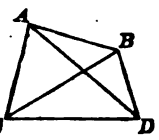


Fig 21

accessible distance *AB* (Fig 21) le points, by observation on two nd *D* when distance *CD* is known, *DAB*, *CBD*, and *ABC*. Assume te *CD* by process described in above a ratio between *AB* and *CD*, and since *CD* is known, B may be computed.

15. Re-running Old Lines

property lines, surveyor must first determine where property lie, and then survey those boundary lines. He correct original lines, even though sure that errors exist t of all look for physical evidence of monuments or other existed on the boundary lines; failing in this, he should to their location on such evidence as occupancy, dimen- r the testimony of competent witnesses. It must not be ry is missing because not at once visible. Stone bounds r three feet deep; the top of a stake soon rots off, but re often found many years after the top has disappeared, ion should be carefully dug over to find traces of the old

deed it is assumed that the document was intended to oundaries of which form a closed traverse. Therefore, e omission of a whole chain-length, or the reversing of will cause a deed description to close, this change may here record of original survey does not close, deeds of often of assistance. Where artificial features are men- these always take precedence over recorded measure- ich marks must be mentioned in deed to acquire force or ts. When area does not agree with boundaries, as de- undaries control. All distances, unless otherwise speci- as straight lines; but distances stated as so many feet y, are supposed to follow these lines even if not straight. a plan, the dimensions on this plan become a part of de-

Where property is bounded on or by a highway, the abutters ne, but where it is an accepted street each abutter yields his ic use. If, however, the street be abandoned, the land reverts that the property is bounded *on* the street; but if it read that t he owns no fee in the street. If a street has been opened and years, bounded by walls or fences, and there has been no pro- e lines usually hold as legal boundaries; but if street lines have officials, and evidence of these lines is such that they can be lly requires a much longer period of occupancy before a private from the public in the street. In the case of a line between ence in location of the boundary will, in general, make it the a mistake in its location, and it has not been brought to atten- ties, nor any question raised as to position, then occupancy for e it a legal line.

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Property bounded by a non-navigable stream extends to the meaning of which is a line midway between shore lines, no time of drought. If property be described as running to a riv to low-water mark, unless otherwise stated; if described as it means to edge of the uplands; if to shore line, it means to has its average regimen of flow. Where original ownership stream, and the water has subsequently receded, the prope gives to each owner along shore his proportional share of rive lines will therefore run perpendicular to channel of stream fr of division and shore lines. A more complete statement o above, particularly with reference to U S Public Land Su address on "The Judicial Functions of Surveyors," by Chief- Supreme Court. (*Proc* Mich Assoc of Eng and Surveyors, 1

Should all evidence of artificial boundaries of a property be to accept deed description as best evidence. If directions of bearings, it is necessary first to determine declination of nee declination should be stated in deed or on original plan, b neither. If the date can be established, the declination for obtained from records of local surveyors or from past U S one line can be identified as a boundary, the difference be bearing and its original bearing gives difference in declination deed lines can be run out by correcting all bearings by this original survey may have been of different length from the readily determined by measuring length of any well-defined difference thus found applied proportionally in measurement

16. United States Public Lan

The system. The United States system of survey rated in 1784 and since modified by various acts of public lands "shall be divided by north and south li true meridian, and by others crossing them at right a ships six miles square," and that corners of townships marked with progressive numbers from the beginning shall be subdivided into 36 sections, each of which sh nearly as may be, by a system of two sets of parall true meridians and the other by parallels of latitude, former at right angles, at intervals of a mile. Since evident that the requirement that the lines shall confor also that townships shall be six miles square, is mathe

A surveyor whose work lies in a district formerly lands should procure from General Land Office, at Was ual of Surveying Instructions for Survey of the Pub States." In this will be found all general and many garding the system, only the general scheme of which

Subdivision work is carried on as follows: *First*, INITIAL POINT, a PRINCIPAL MERIDIAN, by astronomica true meridian through initial point; and a BASE-LINE of latitude through initial point (Fig 22).

These operations are performed in different localities as regions. The principal meridian is a straight line and the every point at right angles to the meridian through that poi by first running a straight line and measuring offsets from it allel of latitude. Two methods are used for this, called the gent method, complete explanations of which, with tables, issued by the General Land Office.

Second, the division of the area into tracts approxim establishment of STANDARD PARALLELS, sometimes c

of latitude extending east and west through 24-mile meridian; and also establishment of GUIDE MERIDIANS, s through 24-mile points on base-line and on standard north to intersection of next standard parallel or base-line.

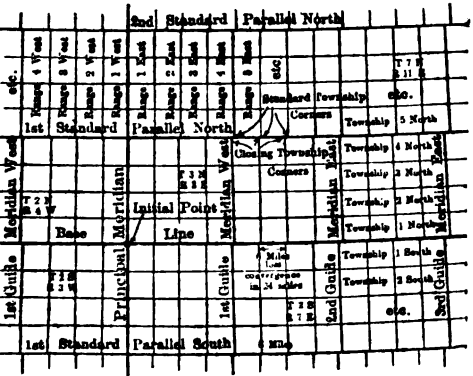


Fig 22. Subdivision of 24-mile Tract into Townships

RIDIONAL LINES, or RANGE LINES, which are true merid- township corners, established at intervals of 6 miles on standard parallels, and extending north to an intersec- dard parallel or base-line; also establishment of LATI- OWNSHIP LINES, joining township corners previously of 6 miles on principal meridian, guide meridians, and

s in fieldwork, the east and west boundaries of all townships out the north and south will vary in length, being a maximum rallel or base-line forming southern limit of a 24-mile tract, rming its northern limit. Townships are designated by num- h and south from base-line and east and west from principal contiguous townships or sections situated north and south of NGE, while such a series in an east and west direction consti- nship 2 North, range 4 west of the sixth principal meridian" ly abbreviated to "T 2 N, R 4 W, 6th P M" (Fig 22). The e lines are made full 40 chains for the entire 24 miles, except ile, into which all excess or deficiency due to irregularities of Similarly, corners on east and west lines are so marked that easurements is thrown into the most westerly half-mile of a

f each township into SECTIONS (Fig 23), each approxi- 40 acres), by establishing SECTION LINES, both meridional l to and at intervals of 1 mile from eastern and south- hip. The sections in all townships are numbered.

rners, surveyor sets up instrument at southeast corner of town- n, and retraces range line northward 1 mile, and township line e. This is for comparing his own meridian and needle obser- chain with those of surveyor who laid off township exteriors. er of section 36 he runs north on a line parallel with east ing a quarter-section corner at 40 chains and a section corner section corner just set he runs east on a random line, parallel

to south boundary of section, setting a temporary quarter-sectio
intersecting the range line he notes the falling of his random lin
runs or falls short of length of south boundary of section. If
50 links (33 ft, representing angular deviation of 21 min), and
short of length of southern boundary of section 36 by not more t
course which will join the two section corners is calculated; this
the west, the permanent quarter-section corner being set at its

From the section corner just regained, the survey is now co
tions 25 and 26, the direction being changed slightly to east or
latitudinal section line just completed exceeded or fell short
and 80 chains on this line quarter-section and section corners,
the section corner a random line is run across to the range line
culated and run as before. This process is continued until 5 of

36 31 32 33 34 35 36 31
Standard Corner
Line
Section Corners
Standard Township Corner
Closing Section Corners
Closing Township Corner
1 6 5 4 3 2 1
(b)
Township Line
Range Line or Guide Meridian
6 5 4 3 2 1
7 8 9 10 11 12
18 17 16 15 14 13
19 20 21 22 23 24
30 29 28 27 26 25
31 32 33 34 35 36
Range Line or Guide Meridian
Township Line
(a)

Fig 23. Subdivision of Township into Sections and Method of Marking Corners

are enclosed.
township is no
section corner
is run north t
from data thu
culated and r
township line
north boundar
tion line, howe
random inters
lished as a cl
tance from ne
measured and
the permanen
established at
tial corner, the
thrown into th

In a simila
ranges of secti
being run east
viously establi
rected back.
of section cor
dom lines are
and are closed
corners in range line forming western boundary of township.
back, however, the permanent quarter-section corners are est
from initial corners of the randoms, thereby throwing all fr
the most westerly half-miles. Reference to Fig 23 will a
method of subdivision.

Subdivision of sections. When public lands were pa
quarter-section was usually the unit area granted as a "hom
tablishment of the quarter-section corner at center of a sect
division of original "quarters" into "eighties," "forties," or
has necessitated location of numerous corners in addition to
by government.

The interior quarter-section corner of a section is always lo
straight lines joining quarter corners on opposite sides of sec
the section may be located within township; that is, it applies
sides as well as to other sections of township. A modification
in west and north sides of a township; in case of sections lyi
ional line from the quarter corner on north or south boundary
east line of section; similarly, in case of sections lying in nort
rior line initiated at the quarter corner on east or west bound
line of section. The reasons for this procedure in these spec
consideration of methods previously described for establis
section corners in such sections. For subdivisions smaller tha
general methods are employed.

n the ground by various kinds of monument, depending upon
of the corner to be perpetuated, the soil, the materials avail-
nd special conditions. Where stone is plentiful, stone monu-
imbered districts, where suitable stones are difficult to obtain.
e points. In prairie regions, where neither stone nor timber
rth may be raised over the corner, a small marked stone. a
harcoal, or some other permanent and distinguishable mark
. Occasionally, in timber lands, the corner falls on a spot
n case the tree itself may stand as the monument.
ed with horizontal notches, to indicate their respective posi-
ction corners on range lines, including under this term prin-
are marked with notches on their north and south faces, the
qual to number of miles to next adjacent township corner
section corners on township lines, including base-lines and
hed on their east and west faces. Township corners, being
ownship lines, are marked with six notches on each of the four
ed, corners on correction lines are marked SC on their north-
faces, depending upon whether they are standard or closing
interior of a township are given notches on their east and
number of miles to east and south boundaries of township.
sections 20, 21, 28, and 29 would have two notches on south
dicated in Fig 21. Quarter-section corners are marked with
meridional lines on their west and those on latitudinal lines
erever possible, a monument set at a corner is witnessed by
h may easily be found, are not readily moved or obliterated.
manent. In timbered country the stone or post is usually
," situated near the corner.

survey of public lands are required to be returned to General
tive form complete data of alinement and of all characteristic
sed or near the lines. The original notes of public land sur-
ve been transferred to the state authorities, to whom applica-
copies of original plats and field notes as are desired.

ecretary of State, Montgomery.
ommissioner of State Lands, Little Rock.
ommissioner of Agriculture, Tallahassee.
uditor of State, Springfield.
uditor of State, Indianapolis.
ecretary of State, Des Moines.
uditor of State and Register of State Lands, Topeka.
egister of the State Land Office, Baton Rouge.
ommissioner of State Land Office, Lansing.
ecretary of State, St. Paul.
ommissioner of State Lands, Jackson.
ecretary of State, Jefferson City.
ommissioner of Public Lands and Buildings, Lincoln.
ate Engineer, Bismarck.
uditor of State, Columbus.
ommissioners of Public Lands, Madison.

e states named either the original records, or copies of them,
ng the various counties of the state, and are kept for reference
County Register of Deeds, County Surveyor, or other official
s of township plats and field notes of surveys of area covered
above states may also be obtained from General Land Office.
rices. In other public-land states, copies of records can be
Surveyor General at State Capitol, except in California and
General are at San Francisco and Portland respectively.

s. An act of Congress specifically provides that corners
d shall be established as proper corners of sections or quarter-
ntended to designate, irrespective of whether or not they were
t place. A further provision is that "the boundary lines actu-

line. In absence of engineer, all this work can be done by foreman. to set grade stakes and offset-line stakes even though engineer h man will then be able to check lines and grades at any time.

To lay off any angle with a tape only, first lay off 100 ft, from vertex *A* of angle, by the 3-4-5 method, using a binations of dimensions for the two legs and hypothenu *CAB*: (3-4-5), (6-8-10), (9-12-15), (12-16-20), (15-20-25), (1 etc. On the perpendicular, measure distance *BC* equal t tangent of angle; angle *CAB* is the required angle.

LEVELING AND CONTOU

18. Profile Leveling

Profile leveling consists in determining elevation of points on a line so that its profile can be plotted and comput The line is first marked off in 100-ft stations, or at shorter The level is set up and a reading, called a backsight (B S or leveling rod held on a BENCH-MARK (B M), which is a point of above mean sea level or any other assumed datum plane determined. The B S is added to elevation of B M, givi ment (H I), or elevation of line of sight above datum pl called foresights (F S or − S), are then read on as many st as can conveniently be seen from instrument, and elevati are found by subtracting F S readings from H I. Rod-re all distinct changes in slope which occur on the line, wh points or not; such intermediate points are located by tape ing) and are recorded as plus stations, as shown in sample

Profile Field Notes

Profile of proposed drain line from Shaft *A* to Pine Brook. Aug. 18, 1913

Sta	+S	H I	−S	Surface elev	B M & T P elev	D
B M1	1.76	963.28	....	.....	961.52	Top of N E c hoisting en 1, p 2
0	....	....	4.6	958.7	.....	.....
1	....	....	5.2	958.1	.....	.....
2	....	....	5.5	957.8	.....	.....
3	....	....	5.8	957.5	.....	.....
3+36	....	....	4.8	958.5	.....	.....
4	....	....	1.1	962.2	.....	.....
T P	3.21	965.51	0.98	.....	962.30	Top of bould
4+77	....	....	0.5	965.0	.....	.....
5	....	....	1.7	963.8	.....	.....
6	....	....	4.9	960.0	.....	.....
6+75	....	....	6.2	959.3	.....	W bank Pin
6+92	....	....	13.7	951.8	.....	Channel of
....	....	....	8.9	956.6	.....	Approx high
B M2	....	....	7.66	.....	957.85	Top N W cor house No Elev 957.8

e level to a new position along line, a TURNING POINT levation determined, to be used at the next set-up of v H I. When new H I is found, by a backsight, F S its ahead on the line, after which the elevation of a d so on. The turning point may be a regular station,

ablished before profile levels are run, in which case on those B M's which are passed during profile level- ould if possible be used as T P's. If B M's have been closing check is made on B M nearest end of line of n sample notes. Sometimes the line of levels is made r returning to original B M. By comparing its final t beginning of the work, the error of closure is ascer- ' P readings, although a mistake may still occur in an tion point.

chosen with regard to permanence, and should be nd found. The rod readings on B M's and T P's o one more decimal place than those read for profile. djustment, the distances between level and rod, when ould be about equal for backsight and foresight. The depend upon distance at which rod can be read dis- sion required. Generally, sights should not exceed 300 quired to nearest 0.01 ft, and even at a much shorter air may prevent such precision.

tes may be checked by computing the sum of the s on the first and last bench-mark and on intermediate ence of these sums should check the difference in ele- final bench-marks. It is well to apply this check at level notes. If station points have been marked in , two experienced men can run a line of profile levels ugh to permit a set-up every 600 ft, at rate of about et-ups per hr.

frequently adopted when it is impracticable to complete a ugh new country, where no B M's have been established. single T P, readings are taken on two different T P's, near tion by at least 1 ft. From next set-up, a backsight is taken are computed, which should agree within whatever limit of f work in hand. This is an excellent method; it requires a line of levels by ordinary method described above, and ry new set-up, by detecting mistakes as soon as made.

hing B M's are sometimes run by a precise level (Art 5), or ial precautions with ordinary level. The observations are eliminate the recognized common errors in leveling, which oft ground; (*b*) unequal expansion and contraction of parts s of temperature; (*c*) variable refraction of air near ground; ight and foresight; (*e*) selecting unstable turning points; bubble not in center of tube at instant of reading.

ripod are eliminated by making two readings on backsight set-up, in following manner; first, B S is read, then F S; lastly B S once more; two rodmen are employed. Correct from average B S and F S readings. Errors due to changes y prevented by shortening time between reading of B S and ent should be shielded from sun's rays, and from wind, by . Rapidity is of great advantage in eliminating errors due s due to refraction of air may be partly avoided by using the level high above ground, and by making observations

in the middle of the day, rather than in early morning or late
and foresights are kept nearly equal in length by reading these
hairs; notes should be so kept that they will show at a glance
and B S distances are equal or not. A foot-plate or pin may be
to use for a T P, when a satisfactory point can not be found
instrument. To aid in holding rod plumb various types of spir

In work of precise character, allowance must be made for cur
refraction of atmosphere. These corrections are usually combi
following: for sights of 300 ft, combined correction is 0.002 ft;
1 000 ft, 0.020 ft. The correction is applied by subtracting it fro
but if rod be held equally distant from instrument on F S and
ture and refraction is compensated.

Allowable errors in precise leveling of various surveys are
Survey, $4^{mm} \times \sqrt{\text{kilometers}}$; U S Lake Survey, $10^{mm} \times \sqrt{}$
River Survey, $5^{mm} \times \sqrt{\text{kilometers}}$; U S Geological Survey, 0.
actually reached fall well within these limits. A high grade o
on the Barge Canal Survey of New York with an ordinary Y-le
allowable error of closure was only $0.02^{ft} \times \sqrt{\text{miles}}$. An accura
by a precise leveling instrument can be reached with ordinar
level, but not so economically.

19. Cross-section Leveling

Leveling for cross-sections is required for determi
rial, whether excavated from a cutting or a borrow-pit, or
mining, or occurring in a mine dump. The entire upp
divided by transit and tape into 10- to 100-ft squares (or
to nearest tenth are taken at all corners of squares, and at
points as necessary, in case of irregular ground. Lines ru
are usually lettered, numbers being used for the other s
point F, 8 lies at intersection of line F and line 8; and
(not at a corner) is designated thus $(H + 6, 7 + 2)$, which
from line H toward line I and 2 ft from line 7 toward line
leveling over the same system of cross-section lines, befor
the differences between original and final elevations at
vertical edges of a series of vertical truncated rectang
edges of a borrow-pit, for example, will probably occur
trapezoidal truncated prisms. Let A = area of horizonta
prism, h_1, h_2, h_3, etc = lengths of vertical edges of prism, a

For truncated triangular prism, $V = A \times (h_1 + h_2 + h_3)$

For truncated rectangular prism, $V = A \times (h_1 + h_2 + h_3 +$

When additional elevations are taken at intermediate
prism should be divided into triangular prisms; the addi
forms the vertical edge at one corner of each triangular
rectangular prisms occur, having same area of cross-secti
of volume may be simplified thus: Volume of assembly of
prisms $= A\ (p_1 + 2\ p_2 + 3\ p_3 + 4\ p_4) \div 4$, in which p_1 = sum
one prism, p_2 = sum of heights common to two prisms, p_3 =
to three, p_4 = sum of those common to four prisms.

If several intermediate elevations have been ascertained, it is
from original surface down to some plane below final surface.
putations for volume between final surface and the same plane.
these two volumes will give required volume.

Road cross-sections, as a basis for estimating quan
railroad or highway construction, are taken by a met
from the above. From plan of proposed road its aline

ter line (Art 18), and is subsequently plotted. On is drawn, which corresponds either to sub-grade or to Roads are usually first finished to sub-grade, which is ce by an amount equal to thickness of road covering, r ballast of a railroad. The width of base of road and are known. For ordinary earth the slope is usually ertical, called "a slope of 1.5 to 1."

he engineer sets grade stakes at every full station, or d slope stakes at both sides, where finished slope will d. All three stakes are marked, stating amount of de at center point, or, in case of slope stakes, stating base of the road to surface of ground at these points. not only at every full station, but at every distinct er line, and also where surface on either side of center ate section properly to represent the volume included Cross-sections are taken perpendicular to center line rves.

he center, set up the level and find the H I to nearest 0.01 ion or bench-mark of which the elevation is known. From vation at given station. The H I minus grade elevation e, which is computed to nearest 0.1 ft. Take a rod-reading he difference between rod-reading for grade and surface rod- at point. It is customary to record cuts as + and fills −. ned by adding the cut to grade elevation, or by subtracting, e is driven on center line and marked with cut or fill, thus:

he points where side slopes meet surface. These stakes are e the ground lies above or below base of section; it is called , but, strictly speaking, there is no cut or fill at slope stakes. is found by trial as follows: In the case of a cut, estimate ermined, and from slope of surface, what the probable side rom center stake to a point on side slope having this cut pe). Make this computation roughly, measure out this nd take a rod-reading at that point. The rod-reading for base of section) minus this surface rod-reading gives cut at act distance out from center stake to a point on side slope mputed distance equals measured distance from center to ct; if not, a second trial must be made by holding rod on operation; the difference between measured and calculated re to hold the rod for second trial. When correct point is d distance equals calculated distance, the stake is marked distance this point lies above the base. The depth of cut, slope stakes, are entered in notes. The same process is re- er side of center. Rod-readings are taken at intermediate needed to define shape of surface; their positions are located center stake, while the cuts at these points are the differ- r grade and surface rod-readings. The field notes of a sec- 30-ft base and slopes 1.5 to 1, which required 4 readings to would be entered thus: $\frac{21.3}{-4.2}$ $\frac{14.7}{-6.1}$ -3.2 $\frac{17.1}{-1.4}$.

ion may be computed by dividing it into triangles, or cross- their areas determined by planimeter (Art 6). By the end tween sections A_1 and $A_2 = \frac{1}{2}(A_1 + A_2) \times$ distance be-

ns, canals, and other engineering structures are begun t right angles to line of structure, at intervals of 10 to nditions, and extending far enough on either side to

include the possible location of structure. These cro
ing the surface, are plotted for each station, and on
cross-sections of proposed completed structure at the
gives a complete record of conditions before work is st
a continuous graphic record of work may be kept fro
of different colored lines on the various cross-sections.

20. Barometric Leveling

The barometer is based upon the principle that
function of elevation above sea level, a change of 1 in
sponding to a change of about 900 ft in elevation. A
also with changes in temperature and humidity, whe
measuring differences of altitude with a barometer, to
and make proper allowance for them. Both mercuri
are used in surveying, but the latter, on account of i
generally employed.

Barometric leveling is used for reconnoissance, or
for a small-scale map. With an aneroid, results as clos
but not without repeating observations and using in
The error in elevations determined by barometer i
whence the percentage error is smaller for large dif
for small differences. The barometer does not give
difference between two readings will be a function of
the two points, provided atmospheric conditions ha
affect readings.

The aneroid barometer has two scales, the inner o
of mercury and the outer one to feet of altitude, the
being, in most instruments, at 31 in on mercury scale.
be movable with respect to inner scale, for the numbe
sponding to 1 in of mercury varies in different parts o
"compensated" are supposed to be so adjusted that
the instrument will not affect readings. The instr
carefully, to avoid disturbing its delicate mechanism
tap the case lightly to make sure that the instrument
changed pressure; it should then stand a few minutes
true reading. It should not be heated by the sun
be held in either vertical or horizontal position wh
readings in these two positions are different it shoul
at all stations. As accurate results may be obtained
aneroids.

The best method of determining elevations by baro
ments, one being kept at a fixed station of known ele
intervals to indicate changes in atmospheric pressure,
to the various points of which elevations are to be de
in elevation shown by the traveling barometer, corr
cated by fixed barometer, will be the true difference
should be noted whenever the field barometer is read
correction as indicated by fixed barometer. The air
be noted whenever barometer is read. If only one
should be read at initial station and again at same st
other observations, atmospheric changes during the i
from difference between first and last readings.

Determining Difference in Elevation by Barometer

Barom Inches	Feet	Barom Inches	Feet	Barom Inches	Feet	Barom Inches	Feet
20.00	18 110	22.75	21 466	25.50	24 457	28.25	27 133
20.05	18 175	22.80	21 533	25.55	24 578	28.30	27 179
20.10	18 240	22.85	21 590	25.60	24 559	28.35	27 225
20.15	18 305	22.90	21 647	25.65	24 610	28.40	27 271
20.20	18 370	22.95	21 704	25.70	24 661	28.45	27 317
20.25	18 434	23.00	21 761	25.75	24 712	28.50	27 362
20.30	18 499	23.05	21 818	25.80	24 762	28.55	27 409
20.35	18 563	23.10	21 874	25.85	24 813	28.60	27 454
20.40	18 627	23.15	21 931	25.90	24 864	28.65	27 500
20.45	18 691	23.20	21 987	25.95	24 914	28.70	27 545
20.50	18 755	23.25	22 044	26.00	24 964	28.75	27 591
20.55	18 818	23.30	22 100	26.05	25 014	28.80	27 637
20.60	18 882	23.35	22 156	26.10	25 065	28.85	27 682
20.65	18 945	23.40	22 212	26.15	25 115	28.90	27 727
20.70	19 008	23.45	22 267	26.20	25 164	28.95	27 772
20.75	19 071	23.50	22 323	26.25	25 214	29.00	27 817
20.80	19 134	23.55	22 378	26.30	25 264	29.05	27 862
20.85	19 197	23.60	22 434	26.35	25 314	29.10	27 907
20.90	19 260	23.65	22 489	26.40	25 363	29.15	27 952
20.95	19 322	23.70	22 544	26.45	25 412	29.20	27 997
21.00	19 384	23.75	22 599	26.50	25 462	29.25	28 041
21.05	19 446	23.80	22 654	26.55	25 511	29.30	28 086
21.10	19 508	23.85	22 709	26.60	25 560	29.35	28 131
21.15	19 570	23.90	22 764	26.65	25 609	29.40	28 175
21.20	19 632	23.95	22 818	26.70	25 658	29.45	28 220
21.25	19 694	24.00	22 873	26.75	25 707	29.50	28 264
21.30	19 755	24.05	22 927	26.80	25 755	29.55	28 308
21.35	19 816	24.10	22 982	26.85	25 805	29.60	28 352
21.40	19 877	24.15	23 036	26.90	25 853	29.65	28 396
21.45	19 938	24.20	23 090	26.95	25 902	29.70	28 440
21.50	19 999	24.25	23 144	27.00	25 950	29.75	28 484
21.55	20 060	24.30	23 198	27.05	25 999	29.80	28 528
21.60	20 120	24.35	23 251	27.10	26 047	29.85	28 572
21.65	20 181	24.40	23 305	27.15	26 095	29.90	28 616
21.70	20 241	24.45	23 358	27.20	26 143	29.95	28 659
21.75	20 301	24.50	23 412	27.25	26 191	30.00	28 703
21.80	20 361	24.55	23 465	27.30	26 239	30.05	28 746
21.85	20 421	24.60	23 518	27.35	26 287	30.10	28 790
21.90	20 481	24.65	23 571	27.40	26 334	30.15	28 833
21.95	20 540	24.70	23 624	27.45	26 382	30.20	28 877
22.00	20 600	24.75	23 677	27.50	26 430	30.25	28 920
22.05	20 659	24.80	23 730	27.55	26 477	30.30	28 963
22.10	20 718	24.85	23 782	27.60	26 524	30.35	29 006
22.15	20 777	24.90	23 835	27.65	26 572	30.40	29 049
22.20	20 836	24.95	23 887	27.70	26 619	30.45	29 092
22.25	20 894	25.00	23 940	27.75	26 666	30.50	29 135
22.30	20 954	25.05	23 992	27.80	26 713	30.55	29 178
22.35	21 012	25.10	24 044	27.85	26 760	30.60	29 220
22.40	21 071	25.15	24 096	27.90	26 807	30.65	29 263
22.45	21 129	25.20	24 148	27.95	26 854	30.70	29 306
22.50	21 187	25.25	24 199	28.00	26 900	30.75	29 348
22.55	21 245	25.30	24 251	28.05	26 947	30.80	29 391
22.60	21 303	25.35	24 303	28.10	26 994	30.85	29 433
22.65	21 360	25.40	24 354	28.15	27 040	30.90	29 475
22.70	21 418	25.45	24 406	28.20	27 086	30.95	29 518

permission, from the "American Civil Engineers' Pocket Book."

Calculating the altitude. The Laplace formula for calculati
tion (neglecting small corrections) is:

$$D = 60\,158.6\,(\log h - \log H)\left\{1 + (t_a' + t_a - 64^\circ)\right.$$

where D = difference in elevation in feet; h = height of mercu
inches; H = height of mercury at upper station in inches; t_a' a
temperatures, Fah. The temperature correction is the only one n
aneroid; it is often large. In case a mercurial barometer is used,
at upper station reduced to temperature of mercury at lower sta

$$H = h'\left\{1 + 0.00008967\,(t_m - t_m')\right\}.$$

where t_m = temperature of mercury at lower station and t_m' = t
at upper station. Table on preceding page, condensed from that
the Laplace formula; it gives the preceding values of log height × 60 158.6

Example using one aneroid barometer to find difference in ele
set River and Blue Hill:

Station	Time	Inches, b
Neponset River	1.00 p m	30.39
Blue Hill	1.45 "	27.73
Neponset River	2.25 "	30.39

Time going up = 40 min; total time = 85 min. Difference in
lower station = 0.005. Probable reading at Neponset River
− (45 ÷ 85 × 0.005) = 30.392 in.

From barometric table	30.392 in	= 29 042 f
	29.730 "	= 28 466
	Difference	= 576 f
Temp corr = $576\left[\left\{\frac{1}{2}(53+42)+47\right\}-64\right]\div 900$		= + 20
Difference in elevation		= 596 f

Boiling-point thermometer. Great differences in elevation
roughly by means of the boiling-point thermometer. Such meas
the fact that the boiling point of water diminishes as atmosphe
and therefore the temperature of boiling point is a function of th
ment. The observation consists in noting temperature of boiling
eter placed in a boiler made especially for the purpose. The app
sea level may then be found by referring to the following table.

Boiling point (Fah)	Alt in feet above sea level	Boiling point (Fah)	Al abov
190	11 720	208	
195	8 950	209	
200	6 250	210	
202	5 185	211	
204	4 130	212	
206	3 085	213	

This table is taken from Wilson's *Topographic Su*

21. Contours

A contour line is the intersection of a level surface
ground. The shore line of a lake is a contour, and if lake
the new shore line would be of different shape and would

. Contours, therefore, connecting points of same ele-
epresent the slope of ground, and provide a means for
nap the approximate elevation of any point. They are
mber of feet above the datum and at regular intervals,
; the number marked on contour is its elevation above

ontours are: (*a*) All points on any contour have same
ontour closes on itself, either within or beyond limits
which closes within limits of map encloses either a
In depressions will usually be found a pond or a lake;
ater the contours are usually marked in some way to
d) Contours can never cross one another, except where
cliff, in which case there must be two intersections.
contours are spaced equally. (*f*) On a plane surface
arallel. (*g*) In crossing a valley, contours run up the
turning at the stream, run back on other side. Since
ight angles to the lines of steepest slope they will gen-
to the thread of stream at point of crossing. (*h*) Con-
ivides) at right angles.
oss a contour map, the profile of that line may be con-
ions of points at which contours are cut by the line are
istances between these points can be scaled or projected
ely, if profiles of a sufficient number of lines are given,
se lines on a map, mark the elevations, and sketch the

it should be borne in mind that they can be interpo-
and that the data necessary to obtain in the field are
haracteristic points on profiles, such as tops of knolls,
rofiles of ridges and of streams. Contours are usually
d bottom of a natural eroded slope (where no wave
n middle Where contour interval is small, and much
und may be cross-sectioned (Art 19) and from elevations
s may be sketched; or contours may be actually traced
ld by stadia; in this case the rodman moves up or down
eading shows that the foot of his rod is on a contour.
ien located by distance and azimuth (Art 23).

is often used to determine elevation of additional points,
on transit line as B M's, as follows: First measure distance
eye, which may be, say, 5.2 ft. Then, from a reading of the
vn elevation the height of levelman's eye is determined. The
n uphill or down until rod-reading is such as will correspond
ntour; the point thus found is located by tape or by pacing.
l on a side-hill, the levelman then moves uphill past the located
nds, until he reads the contour interval plus 5.2 on the rod;
ontour, which is then located by taping. The levelman re-
rodman travels uphill until levelman reads on rod 5.2 minus
s now resting on the next higher contour. In going downhill,
r while levelman backs down the hill until he reads 5.2 minus
he will be standing on contour next below rodman; the rod-
and backs down the hill until the rod-reading is 5.2 plus the
next lower contour. Hand-leveling is not accurate enough
bout 400 ft away from a B M. If several profiles be run along
dotted on map, and then a few necessary side shots be located
an then be sketched. The best method of locating contours
methods (Art 22 to 25).

TOPOGRAPHIC AND RAILROAD

22. Stadia Method

The stadia method measures distances by observing, through instrument, the space on a graduated rod intercepted betwe called STADIA HAIRS, which are spaced equal distances above an zontal cross-hair. When rod is held at different distances from spaces on the rod will be included between the stadia hairs. T measuring distances while filling in details of topographic and h has the great advantages that the intervening country does not inaccessible distances, as across water surfaces, can be measure ment are compensating rather than cumulative, while its accura kinds of work, even for determination of area in some cases s part in 500 may be attained with the stadia. STADIA RODS ar illustrations indicating method of reading them.

Fundamental principle of the stadia is the geometric theorem homologous sides are proportional. The common vertex of the tri the center of the transit, but a point in front of telescope obje

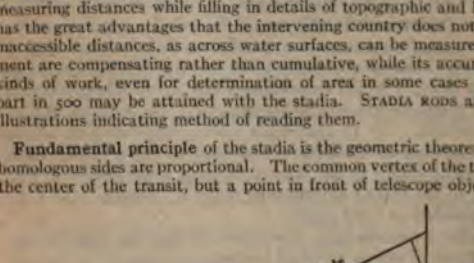


Fig 25 *a*

and 2 ft for plane-table alidades. The other function $(F \div i)$ is customary to space stadia hairs so that $F \div i = 100$, the eq ments reduces to: *distance* $= 100\ s + 1$. Every 0.01 ft on rod t distance of 1 ft.

Inclined sights. It is necessary always to hold rod verti being taken. When line of sight is inclined, the distance inte represents a distance greater even than the inclined distance t tance is greater also than horizontal distance. All inclined re reduced to the horizontal, and if difference in elevation betwee rod station is desired the vertical distance also must be compute By trigonometry, making one slight assumption, it may be sh

$$\textit{Vertical Distance} = (F \div i)\, s \times \tfrac{1}{2} \sin 2\alpha + (F$$
$$\textit{Horizontal Distance} = (F \div i)\, s \times \cos^2 \alpha + (F + c)$$

For the ordinary conditions (when $F \div i = 100$, and $F + c =$ proximations that $\sin \alpha = \frac{1}{2} \sin 2\alpha$, and that $\cos \alpha = \cos^2 \alpha$, formulas, which assumptions are nearly true for small angles,

$$\textit{Vertical Distance} = (100 \times \textit{rod interval} + 1)\, \tfrac{1}{2} s$$
$$\textit{Horizontal Distance} = (100 \times \textit{rod interval} + 1) \cos^2$$

These are the formulas used for ordinary stadia work. The red be best accomplished, however, by use of tables, diagrams, or which are based upon these formulas.

23. Stadia Fieldwork

by (*a*) azimuth, (*b*) distance, and (*c*) angle of elevation
ations are not required, the vertical angle is read only to
hat is close enough for determining horizontal distance;
equired, vertical angles should be read to nearest minute,
ute, and vernier index correction noted for each important
ace and azimuth angle are read with whatever refinement
n hand; as a rule, distances are recorded to nearest foot,
t minute, although for side-shots, which are to be plotted
nuth to nearest 5 min is close enough. Traverse lines
ey are measured by tape, or by stadia, as accuracy of sur-
me surveyors prefer to run traverse lines by transit and
vel over traverse to determine elevation of certain transit
ther bench-marks, thus establishing an accurate skeleton
hich follows over same ground for filling in details. For
y be advisable to tape the courses of control survey, but
he stadia method is sufficiently accurate. In latter case,
d on both foresight and backsight.

g a survey, if the true azimuth of any course be known, all
refer to the true meridian; but if no meridian has been estab-
e case, the azimuths may be referred to magnetic meridian, or to
tion. Azimuth angles are most commonly read from the south
horizontal vernier is set on 0°, the telescope turned to point
ower clamp tightened. The upper clamp is next loosened and
station; the horizontal arc is then read clockwise to determine
referred to the magnetic meridian through first station. The
t is obtained by sighting telescope on it and reading the vernier,
ing clamped. On moving to the next instrument station, the
nverting and backsighting, as explained in Art 11. Magnetic
e noted as a rough check on azimuths.

setting one of stadia hairs on a whole foot-mark, by vertical
the intercept between the stadia hairs to nearest 0.01 ft. Care
take middle horizontal cross-hair for a stadia hair; this can be
g a mental estimate of the distance, or by adding oblique hairs
sionally a half-interval may be read when something obstructs
l; it should be recorded as twice the interval read (or, better,
es" taken separately).

d by sighting middle cross-hair on the point on rod correspond-
ontal axis of telescope to station beneath transit. This distance
r (H I); it is not the same as H I in leveling, which is the ele-
datum. The line of sight is thus made parallel to a line from
tical angle read is the inclination of this line, whence difference
tations can be directly computed. Same vertical angle is used
distance.

ight on H I point of rod, owing to obstacles in line of sight, the
ing at any convenient foot-mark on rod, this reading of middle
tes, with the vertical angle.

on as follows: When transit has been set up and properly ori-
h angles, the transitman loosens the upper motion and sights
be located. He first reads the stadia intercept from that part
int, and records it; then sets middle cross-hair on H I point of
r bisecting rod. He then signals rodman to go to next point,
azimuth, vertical angle, and index error, if necessary. If eleva-
e does not set on the H I, nor read vertical arc unless angle is
eds 3°, it is recorded only to the nearest 10 min, since it is used
tions..

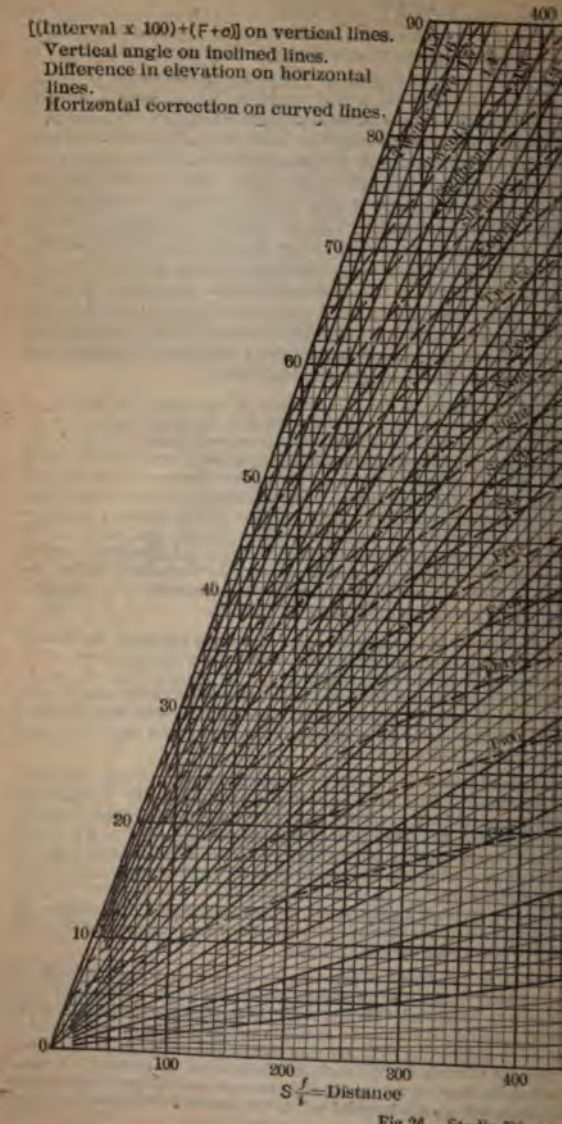
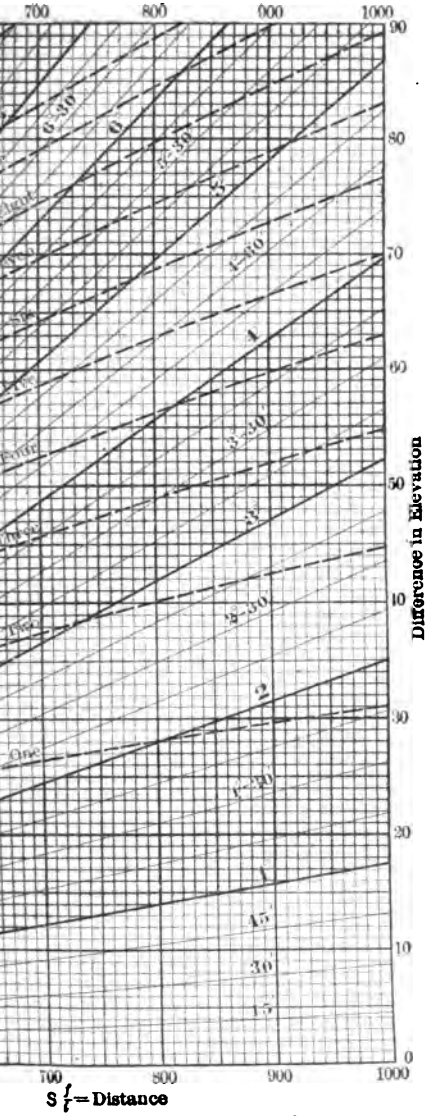


Fig 26. Stadia Diagram


700
800
900
1000
90
80
70
60
50
40
30
20
10
0
0-30
0
2-30
3
4-30
4
3-30
3
2-30
2
1-30
1
45'
30'
15'
Difference in Elevation
S $\frac{f}{f}$ = Distance

Stadia notes are kept on left-hand page in columns headed, fro
follows: Station, Distance, Vertical Angle, Difference in Elevation
last two columns, as a rule, are computed in the office (Art 24). On t
should be drawn such sketches and details as are required to convey
tion to enable a draftsman unfamiliar with locality to plot the notes.
are usually designated by letters, and side shots by numbers. These
recorded on sketch within small circles, or in some other manner, so th
mistaken for measured distances, which will also appear in sketch.
survey is to produce a contour map, the accuracy of map depends in l
the completeness of description given in notes regarding slope of th
located points. The elevation of points on ridges, valleys, knolls, an
be determined in the field, but without a full description of intervenin
interpolation of contours is often almost impossible. Notes like the
the required information: "Straight slopes from pt 16 to 17, 17
Supplementary sketches illustrating shape of the ground are often
interpretation of field-notes, and their expression on map, are the
of topographical surveys made by stadia; whence the importance of
tive notes can not be overestimated. In open rolling country, where
terval is required, a party of three men should cover about 5 acres pe

24. Stadia Computations

Reduction of stadia notes includes computation of DIFFER
TION (vertical heights) and HORIZONTAL DISTANCES (or horizon
Computations almost always made by tables, diagrams, or stad
latter being the most convenient means for reductions both in fi
Horizontal distances can be quickly computed also by use of R
RECTIONS, which are the distances to be subtracted from incli
obtain horizontal distances; to the latter must be added the inst
$(F + c)$, provided character of work in hand requires taking a
or 2 ft. Since the rod intercepts are usually read to nearest 0.01
recorded only to nearest foot; hence, in most topographic wo
$(F + c)$ may be neglected, and there is no need of applying horiz
for vertical angles under 3°.

The stadia diagram, Fig 26 (by J. E. Finch, C. E., of Columbia
100 × rod interval, on vertical lines; vertical angles, on inclined lin
elevation, on horizontal lines; and horizontal corrections, on curved lin
is used as follows: Rod interval × 100, as recorded in field-notes und
427; vertical angle 3° 51′; instrument constant $(F + c) = 1$ ft. Follo
428 to inclined line 3° 51′; horizontal line reads 28.7, which is entered
in the column headed Diff in Elev. It is then applied to the elevation
to give elevation of the point desired; if vertical angle is upward (+)
elevation of transit point and if downward it is subtracted. The hor
is read from the curved line which lies nearest the intersection of ver
inclined line 3° 51′; in this case it is a correction of 2 ft, to be subtract
of $(F + c)$ has been added, as follows: $427 + 1 - 2 = 426$ ft.

This diagram will be found of great assistance in applying rough che
tions, especially in the use of plane table, if it be remembered that on
of 34′, the vertical height is 1% of inclined distance; on a vertical angle
and on an angle of 5° 46′, it is 10% of inclined distance.

Even without this diagram, differences in elevation may be comp
table of horizontal corrections is available. After horizontal distance
multiply it by tangent of vertical angle, to get vertical distance; this ca
performed on the tangent scale of an ordinary slide-rule.

When vertical angle is not read to the H I mark on rod, this fact m
account in computing differences in elevation, as follows: Suppose the
135; vertical angle, −8° 50′ on 2.5; H I, 4.5. From diagram, diff e
which add +2.0, giving −18.6. The 2 ft is positive because vertical
ured to a point on rod 2 ft lower than H I point, which made observ
greater than it would have been if sight had been taken at H I point

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s an attachment to the vertical arc, giving readings which ables, diagrams or slide-rule. It requires more manipulation usual method, but is of advantage particularly when eleva- the field as work progresses.

25. The Plane Table

ed for constructing a map directly in the field. The ard, like a drafting board, supported on a tripod, with clamping so that it can be oriented in any desired azi- table top is 24 by 30 in, and the tripod most commonly ch has an ingenious ball-and-socket leveling device and ALIDADE is used to define directions and measure dis- ope equipped with stadia hairs and is mounted on a n Y supports; these are connected to a metal column f a flat piece of metal about 18 in long, of which both arallel to plane of telescope. On this base are two the table. The telescope has only a vertical motion e entire alidade is moved about on top of map after

ble has generally been limited to topographical maps drawn imate field is much broader than that. A system of triangu- nd tape traverse may first be carefully surveyed and plotted run to determine elevation of triangulation or traverse points, a control for plane-table survey. In this case the plane details, and accurate maps on a scale as small as 40 ft to the The most important advantage of the plane-table over other all the sketching is done in the field, where topographer can is mapping. He can sketch details at once in their proper his memory and without making elaborate notes. For this ately sketched from a much smaller number of located points instance, by transit and stadia method.

ut a system of triangulation, or a transit and tape traverse, for making such a survey graphically, as explained below. ccuracy of those points located by plane table is limited by er than of the measurements. In the field the map must be y moisture, so as to preserve accuracy of plot. Celluloid e used even in rainy weather.

sually, though not always, requires more time for fieldwork thods; it is also more dependent upon favorable weather. h the field and the office work, the plane table has proved ansit and stadia for work in open country, while the results, e. In open rolling country, a contour interval of 5 ft being hould survey about 8 acres per day.

rsection. To begin a plane-table survey it will generally map two plotted points corresponding to two points on the hich is known, and of which one, at least, can be occupied ethod of locating points, without measuring any distance, is is plotted on plane-table sheet, representing the measured e. The table is first leveled and so set that a on the map is round (a special plumbing appliance is used for this adjust- is placed along base-line ab drawn on map, the table is turned ghts signal B, and horizontal motion is clamped. Line ab is table is said to be oriented. The alidade is now placed with ough a, the telescope is sighted to some signal C, and an in- d C; the point c on map is somewhere on this line. Now vertically over B) and repeating the process of orienting the C, point c is located at intersection of lines ac and bc. In of points may be located.

Locating points by direction and distance. The commonest w
by plane table is to obtain direction with alidade and measure distan
is then laid off to scale along the straight-edge. If a control travers
table can be set up over any of the plotted points and oriented as ex

Locating points by resection. It is sometimes desired to lo
station from a base of which only one end can be occupied by the ta
the points on ground at ends of base-line; C is the point to be located;
plotted on plane-table sheet. Set up at A, the accessible end of th
table by sighting B with alidade pointing along ab. Then, centering
an indefinite line toward C, for full length of the alidade. The table
and oriented by the indefinite line just drawn. Since position of c on
is not known, its position on map must be estimated for setting table
alidade is now sighted toward B, with the edge of its base on b and
drawn, which cuts the first indefinite line, thus locating point c desire
found is not over point C on ground, reset over C and repeat process.
found by this method should be checked, if possible, by resection line
of which the positions are known to be correct.

26. Triangulation and Trigonometric Level

Triangulation consists in locating certain points by obser
starting from a base-line which has been accurately measured
often used as a control for a topographic or hydrographic surv
tion system comprises a series of triangles, the corners of whic
ing stations or points which it is desired to locate. In this sy
the length of one side of some triangle must be measured; the
in each triangle be observed, lengths of all other lines in the sy
culated by trigonometry. Three types of triangulation syste
nized: First, a series of approximately equilateral triangles;
central polygons, for example, a row of hexagons each having a
third, a series of quadrilaterals with both diagonals drawn.
is the cheapest when it is desired to extend survey along a
surveying a river. It has the disadvantage of having but few
few geometric conditions which must be satisfied by the mea
second system is adapted to surveying an area of greater wid
The third is the most accurate, because it has the greatest n
No attempt will here be made to describe the precise work
U S Coast Survey in mapping large areas, the following note
such work as the mapping of a few square miles.

Fieldwork consists in first making a reconnoissance of the
suitable triangulation points with due regard to shape of the
should be as nearly equilateral as possible, choosing a favorabl
base-line to be measured by tape, and ascertaining that the
visible each from the others.

Base-line should be located on fairly even ground, for conve
ment, and its position should be such as to admit of accurate connect
gulation system. After clearing line of obstructions, stakes are line
tape length apart. The grade of each tape length is determined by
the actual measurement of each tape length the intermediate support
vertically and horizontally, proper tension is applied with a spring
perature is read on two or more thermometers attached to the tape.
the ends of tape length are scratched on strips of metal tacked to
least two complete measurements of the entire base should be made.
For complete description of methods used in base measurement, see
intendent of U S Coast and Geodetic Survey for 1907 (p 105) and 191

Angle measurements. The transit is set up at each end of base-
carefully measured by repetition (Art 11), to the signals erected at

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tended to be connected directly with the base. The transit is then
s points of triangulation system, and the necessary angles are meas-
of every line in system can be computed, and as many as desired
omputation through different triangles.

ngulation points are frequently made of a 4 by 4-in pole supported by
e are tacked black and white cloth bands of definite width, or flags.
supported 7 or 8 ft above the ground, so that theodolite may be set
hen measuring angles. Tripod signals may also be made of young
smaller ones the pole usually rests on the station point; this neces-
ie entire signal when angles are measured at that station. Where it
a high signal, on account of woods, a tall mast may be made of 2 or
ther, and braced by wire guys. In all cases it is advisable to tack
and to that part of the pole which is *vertically over the station*, so
ay always be made on that particular point.

leveling consists in computing difference in elevation of two
s of horizontal distance and vertical angle between them; it
l with triangulation work, the vertical angles being measured
rizontal angles. A vertical angle is measured to some definite
e height of which above station was determined when signal
height of instrument above its own station should also be
orded. In very precise work, angles are measured with a
cle instrument. In less precise work, an ordinary theodo-
: of which reads to 30 sec or to 20 sec, may be used, but with
nly single readings can be made; in which case the best re-
by averaging several independent readings, half of which are
pe direct and the other half with telescope inverted. In
x correction, or reading of vertical arc when telescope is level,

ty in obtaining accurate results by trigonometric leveling is
the coefficient of refraction of the air. This varies with
re, and atmospheric pressure, and the only way its effect
s by taking simultaneous observations between two stations.

ne station. Multiply horizontal distance by tangent of vertical
gle correction for curvature and refraction by the formula: $h = K^2$
distance between stations in miles, and h is correction in feet. If
ft, then $h = 0.02\ K^2$ (nearly). This correction is applied to increase
if vertical angle is positive, or to decrease it if angle is negative.

neous observations can be made, $h_1 - h_2 = K \tan \frac{1}{2}(\alpha_2 - \alpha_1)$
$+ K^2 + 12 R^2\}$, where $h_1 - h_2$ = difference in elevation of stations
e earth subtended between plumb lines through the two stations
to horizontal distance between the two stations) in feet; α_1 and
observed vertical angles; R = radius of earth at latitude of sta-
t is close enough to take log R (in feet) = 7.32068.

27. Photographic Surveying

of locating topographic details for construction of small-
- 3 miles to 1 in) is afforded by photographic surveying.
ained where country has characteristic shapes, and is not
o afford good positions for taking the views. To locate a
obed from two stations, the positions of which are known.
ow the direction in which camera was pointed when each
n, and the focal length f of lens.

or this work sometimes consists of a transit, so constructed
elescope can be removed and a camera mounted in their

place, so that azimuth angles can be measured, or laid off, circle. The instrument is leveled and horizontal angles be defined points are measured while telescope is on transit; w substituted, the same angles are laid off on the circle and made. Such instruments cost about $400.

Notches on the 4 sides of the camera opening yield corres edges of every picture; lines are then drawn across photograp 4 points. The vertical line is known as the PRINCIPAL an HORIZONTAL line; their intersection is called the PRINCIPAL P

If photograph be held at distance f in front of the eye, and normal a, b, and c on the picture will appear to cover corresponding points A, scape. The photograph, when held in this position, thus gives a n between various points in the view; a fact utilized in plotting.

To plot map, first locate positions of camera stations X and Y station draw a circle of radius f, and a radial line representing direc was pointed when photograph was taken, as Xm and Yn. Line op photograph, tangent to line, and qr represents Y, its principal line pa plot point a, first meas the distance from a to plot that distance on l larly, the distance na' other photograph and The plotted position of tion of Xa' and Ya'', o ner are plotted as ma

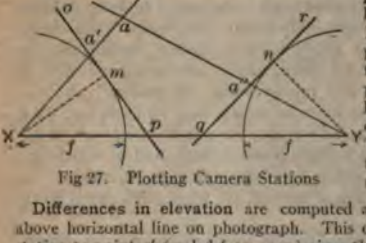

Fig 27. Plotting Camera Stations

Differences in elevation are computed as follows: Scale the above horizontal line on photograph. This distance divided by d station to point a' (scaled from map) gives the natural tangent of depression. The actual horizontal distance Xa is then scaled from has been plotted, and is multiplied by tangent of vertical angle, to vation between camera station and point a; or it can be found gra map are now sketched from a study of the photograph, after chara been plotted.

28. Hydrographic Surveying

Hydrographic surveying is usually applied to the deter of bottom of a body of water, sometimes including a determi of material composing the bottom. It is customary first to shore, by triangulation or traverse, to which the hydrograp referred, then to measure (usually from a boat) the depth o points, and determine position of these points. Besides or tape outfit, a sounding-pole (or lead-line in deep water) and sary equipment will be required. In tidal waters, and in level changes rapidly, a tide gage should be installed. Soundi to self-reading rods, graduated to 0.1 ft; a shoe is sometimes end, provided with a cup-shaped cavity which, when smea soap, brings up samples of material from the bottom. The l or a hemp or cotton line, to which is attached a lead weig chain is satisfactory, having cloth tags of various colors for f there is little current, a 6- to 10-lb weight suffices for depths

Locating soundings. There are 6 general methods: (*a*) on a range line, at uniform speed, and soundings are located (*b*) Boat is rowed on a range line, while positions of soundin a transit reading from shore, or by an angle taken with

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on a range line, or not, while its position is located by usly by two transits on shore, or by angles taken simul- ns from boat by two sextants. (*d*) Soundings are located Soundings are defined by the intersection of fixed ranges. a wire or cord is stretched from shore to shore, and sound- red points along this line.

l angle. In still water, where the boat is easily held on any are conveniently located by proceeding along a range line of r two objects on shore, such as range poles, and then "cutting ng pole by a transit angle observed from a previously deter- ant the sounding is made. The ranges may be fan-shaped if steeple or some other object of known position situated far that the lines will not diverge too rapidly. The recorder writes s as called off by leadsman, and also the times of sounding. by calling out "Sound" about 5 sec before each sounding is kes soundings as quickly as possible, usually within 2 or 3 sec ydrographic work of Corps of Engineers, U S Army, in which cated by two angles taken with transits on shore, soundings , or 30-sec intervals, and an observation is made each minute the instant signal is given from boat. The chief of party directing work in the boat, and seeing that boat is kept on the The signal is given by holding up a flag for about 10 sec and stant the sounding is taken, at which moment the transitmen leadsman, or to the pole, if visible. In work of U S Engineers, black flags are used for signaling. Both the recorder's and es should report color of signal, as well as the time for each ing two means of identifying angles with their corresponding heck is of particular value in taking long lines of soundings. evel, the time record is required also, to reduce soundings to er datum. The tide-gage readings are recorded at regular ay.

is to locate every slight change in shape of bottom, it is desir- requently as leadsman can handle the pole, instead of at timed e boat is usually rowed on a range and the instrument men on soundings that are designated by signalman in boat, the time second. As a rule every sixth or eighth sounding is located. e soundings being quite close together, the points that were ap and intermediate readings are interpolated, the soundings e equally spaced between the located ones, unless time record

29. Mine Surveying

Underground Surveying."

30. Railroad Location

RECONNOISSANCE is an examination of the country railroad is to run, for purpose of choosing the general worthy of further investigation. PRELIMINARY SURVEY route chosen as the result of reconnoissance. LOCATION g in staking out the railroad as straight lines and curves. en called upon to lay out highways, or branch railroad he treatment here given will, therefore, refer to branch- same principles apply to main lines. If reconnoissance equent surveys will not rectify such defect.

in best available map of the country. For this purpose the articularly valuable, because they give elevations as well as be obtained for 10¢ each from office of U S Geological Sur- on these maps, a paper reconnoissance can be made in the

office, probable grades and alinement can be studied, and a very good gener
without going into the field. Such paper reconnoissance ought always to be
examination of the ground, to determine character of excavation, bridge cros
important details. Large portions of the country have not yet been surve
Geol Surv, and it is not uncommon for an engineer to find himself without
him. In this case he should travel over the most favorable routes, obtaini
pacing or stadia, and elevations by aneroid barometer and hand-level; these
porated into a rough reconnoissance map. He should take into consideratio
cost of road, but also that of maintaining and operating it, noting, therefore,
ture, length of line, earthwork, character of soil, bridging, tunneling, and c

A ridge location usually has advantage of affording good drainage. A
usually permits better grades, but almost always requires more bridges an
is more subject to washouts. A sidehill location, where slope is not too irr
has advantage that proposed line can be placed at any desired grade. O
tainous countries, operative grades can not be obtained, even in bott
valleys; the road must then be purposely lengthened by entering lateral
looping upon itself. In routes which cross drainage systems, it is impor
mine the lowest passes and use these points as governing the location; in
is advisable either to tunnel at the pass or to operate for a short distanc
grade, with a helper engine or by breaking the train in halves. In such l
is almost always of much greater importance than length of line or amount
although due regard must be given to sharpness of curvature, for reasons

Preliminary survey is a topographic survey of the belt in wh
location of the railroad will lie; it will first be run as a transit and
to be used as a back-bone for topographic details. An accurate
survey aids greatly in the location survey which is to follow. I
measured usually to 0.1 ft, and deflection angles to nearest minute;
distances are read by stadia. This map is usually plotted on a sc
400 ft to 1 in. A preliminary party in open country will cover
miles per day. The level party following the transit party establ
marks and determines the profile along preliminary line.

Topographic details are sketched to scale on cross-section paper in th
elevations given by level party as bench-marks; a stadia survey or a plane
adapted to this work in open country; in wooded country a hand-level is
A 5-ft contour interval is customary. The preliminary map and topogr
should be plotted at close of each day, from which the locating engineer w
day to day whether to push ahead on the line adopted, or to retract and
different ground. Not infrequently preliminary surveys of more than one l
The kinds of material encountered along the route are noted and the cha
bridge crossings. Upon the preliminary plan the locating engineer can dra
line, composed of straight lines and circular curves, and from the contour
proposed location line can be plotted. From a study of the different line
the engineer will determine a final line, which will be drafted upon preli
and run out by the party as center line of the location.

Location line, as laid out on the preliminary map, will be so choser
sect controlling points, such as bridge sites and passes, and to minimize
of earthwork. Due regard is given to grades and characteristics of th
On curves, the rate of grade should be flattened about 0.04% per deg
ture, to neutralize additional train resistance; called CURVE COMPE

It is good practice to lay out grade line for a branch connection as nearly
ticable for about a train length from its junction with main line. Similarly
should be given for short distance in vicinity of a mine; this permits locom
its train before climbing up-grade, or, if grade is down, it prevents loaded c
ning away. These flat grades are advisable, from an operating standpoin
obtain them, it is necessary to steepen other grades. Wherever the line i
more than 500 ft long, it is advisable to let this occur on grades as steep as
vide drainage. In passing through low flat land, the line should be built a f
meadows or swamps, with ample culverts for brooks, and the grade should b

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ruction of such culverts. It is bad judgment to economize on the ngs. The width of cuts, at the base, is usually made about 6 ft ovide for side ditches.

line has been finally drawn on map, it will probably be found to cross ninary traverse several times. Scaled distances along the traverse location line, which will be used by the field party in laying out the line. Sometimes when the possible location is constricted, as in a ion is worked out by the field party and the preliminary survey is tion line is actually being run out in the field, minor changes will ountering physical conditions not recorded on preliminary map.

center line of location, each tangent is first defined and run to an cent tangents; this locates vertices of the curves. The curves are EFLECTION ANGLES. Measurements are taken to nearest 0.1 ft, and ate; a stake is driven at every full station point and at the beginning ve. The beginning of the curve is called the "T C" (tangent to e "C T" (curve to tangent). A level party, immediately following ains data for a profile, reading surface elevations to 0.1 ft, and T P's

When final alinement has been staked out, the level party, using profile as a basis, takes cross-sections at every station (Art 19).

are used to connect tangents of a branch track, the refine- urves not being necessary. The point at which two tangents P I" (point of intersection). The intersection of tangents is called le, and is equal to the angle sub- dii drawn to the T C and C T. e is distance from T C to C T, chords (a sub-chord may occur at f a curve). The degree of curva- at center subtended by a chord less than 100 ft is called a sub- ral angle is a sub-angle. The and D is expressed by formula, $\frac{1}{2} R = 50 \div D$.

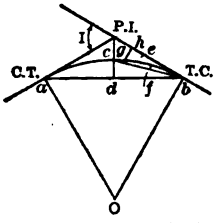

Fig 28. Layout of Circular Curve

tem, the degree of curvature has ways: (*a*) As the angle at center subtended by a chord of eflection angle of a chord of 20 m.

simple circular curve. Following terms apply to its elements.

$s = R$	cd = Middle ordinate = M
$l = C$	cPI = External distance = E
ent distance = T	hg = Offset from tangent = t

ons between these elements are true when I is less than 180°:

$= 50 \div r$

$730 \div R_n°$ (approx) $= T_1° + T_2°$ (approx) $= E_1° + E_n°$ (approx)

$o \div D$ (approx) $= \frac{1}{2} C \div \sin \frac{1}{2} I = T \cot \frac{1}{2} I$

$\sin \frac{1}{2} I = 2 T \cos \frac{1}{2} I = 2 M \cot \frac{1}{2} I$

$I \div D$ (length of curve in railroad practice only)

$n \frac{1}{2} I = \frac{1}{2} C \div \cos \frac{1}{2} I = E \cot \frac{1}{4} I$

$\sec \frac{1}{2} I = T \tan \frac{1}{4} I = M \div \cos \frac{1}{2} I$

$rs \frac{1}{2} I = R - \sqrt{(R + \frac{1}{2} C)(R - \frac{1}{2} C)} = C^2 \div 8 R$ (approx)

$d bg^2 \div 2 R$

a the length of a curve is usually considered as its actual arc, e of curvature D is not used. The central angles of any part of ortional to the arcs they subtend. In computing such cases, the , E, M, and t apply.

$o° = R \times$ (length of arc for radius 1) $= C \div (C^3 \div 24 R^2)$ approx

$\overline{bca}^3 \div 24 R^2$) approx

Deflection angle is the most common method for locating simple curve a curve by this method, set transit at the T C, the vernier reading 0°, a (Fig 29). Lay off deflection angles AB 1 and measure chord B 1, thus lo Angle AB 1 = ½ α_1, and $\alpha_1 = D \times B$ 1 ÷ 100. Leaving lower plate of tr loosen upper motion, lay off AB measure chord 1–2, and so on, sett and checking at end of curve on The total deflection angle to any is evidently half the central angle and that point; where stations ar the deflections increase by ½ D fo sive station.

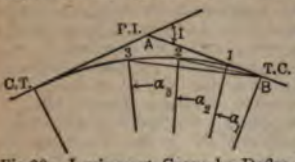

Fig 29. Laying out Curve by Deflection Angles

Deflection angle for any sub-c 200 (in degrees) = $c \times 0.3 \times D$ (in D is in degrees).

Example. Find deflection angles of a 6° curve connecting T C 8 + 42 + 73-4, the curve being to right.

$58.3 \times 0.3 \times 6 = 105' =$ 1° 45′ to Sta 9
1° 45′ + 3° = 4° 45′ " " 10
4° 45′ + 3° = 7° 45′ " " 11
7° 45′ + 3° = 10° 45′ " " 12
$73\text{-}4 \times 0.3 \times 6 = 132' =$ 2° 12′
12° 57′ to Sta 12 + 73-4

C T 12 + 73-4 − T C 8 + 41.7 = 431.7 = Length of cu
4.317 × 6° = 25°.902 = 25°54′ = I; and 12°57′ = ½

This checks deflection angle from T C to C T. As a rule deflection ang computed to nearest half-minute.

Main-line locomotives, as a rule, are limited to curves of 16° to 20 upon their length of wheel base and number of wheels having flanges senger coaches will not go around a 20° curve unless the truck swivel-c connected. A 4-wheel switching engine will, as a rule, go safely arou 75-ft radius, and a 6-wheel switching engine around one of 90- to 1 depending upon its design. A single box car can be pulled around a radius, and two box cars around curves of 80 to 100-ft radius if a spe coupler be used between cars, and between tender and locomotive. about 140-ft radius the corners of freight cars are liable to collide if th has the ordinary M C B coupler. On all especially sharp curves the ga properly widened, the rails braced, guard rails set, and curvature ma form. STANDARD GAGE is 4 ft 8.5 in. The widening of gage may be ¾ in on sharpest curves. It is advisable to lay guard rails on all curves 14°. It is of importance to lay out a branch line so that a main-lin can run over it; this can be done if the curves are of not less than 3

Cost of branch R R lines varies with amount of earthwork a required. Under ordinary conditions, where cuts and fills average total cost per mile of single track branch line is about $10 000. tracks in place will cost from $1.20 to $2.00 per linear ft of track estimate assumes ties at 65¢ each, that gravel ballast is obtaina few miles, that ballast is 12 in deep under ties, and that relayer ra costing $20 per ton (that is, rails previously used on main-line track but still serviceable for branch track):

	Cost pe
Gravel ballast @ $0.65 per cu yd in place	
Ties, 24 in c-c, @ 65¢ each	
Relayer 75-lb rails, @ $20 per ton	
Rail splices and bolts, 5¢; spikes, 4¢	
Laying and surfacing track, without tie plates	
Total	

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ncluding rails of both tracks, turnout ties, frogs, switches
te, is from $400 to $600 each, for lengths of 70 to 100 ft
point of frog.

be constructed, where excavation and filling is earth,
ostantially the ground surface, and few cuts are greater
$1 000 per mile. These costs assume that some material
aced upon subgrade, to make a suitable surface for heavy
nes a road with travel way 10 ft wide and occasionally
to pass. To surface a mile of road 10 ft wide with 12 in
hin one mile, will cost from $1 000 to $2 000.

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SECTION 18

UNDERGROUND SURVEYIN

BY

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Note. — Numbers in parentheses in text refer to Bibliography at end of

Introductory. An underground survey is based on precisely the same one on the surface. The methods of reading angles and measuring distan puting azimuths, bearings, latitudes and departures, coördinates, and elev plotting the results, are alike in all essentials; for these principles, refere made to the preceding section. In their practical details, however, a num ences are introduced by certain peculiar conditions under which a mine sur conducted. These conditions are: 1. Working in the dark requires that b to be sighted and the cross-hairs of a telescope must be illuminated. 2. constricted places, and in passages through which traffic must not be inter requires the use of special devices for mounting the instrument. 3. Stat placed in the roof, when possible, to assure their permanence and visibility inclined sights are more prevalent, requiring that a transit shall be adjustment; with very steep sights, an auxiliary telescope is required, inv rection in the reading of angles. 5. A higher degree of accuracy is usually in ordinary land surveying; in this respect, mine surveying is comparable wit of bridge piers or the boundaries of city real estate. This is because, in m portant problems in mine engineering, aside from mere mapping, an error time and money in the subsequent mining operations. Furthermore, happens that a complete circuit can not be made, whereby the error of cl determined and distributed, or a mistake located and corrected; hence ca refined methods is the only assurance of accuracy. 6. Shaft-plumbing problem, peculiar to underground surveying,

1. Stations

ion of the points to be occupied as stations will facilitate
ns, and extra time devoted to this end is well spent. In
where the workings are regular and long sights possible,
he head chain man to select and put in the stations as the
in most metal mines, especially if an entire new survey is
visable to select the points throughout the whole workings
instrumental work. Three men can do this to best ad-
ections of workings the point should allow sights to be
ns; intermediate points should allow the sights to be as
rdinarily, little advantage is gained by attempting to set
ine, with the idea of reducing office calculations; an excep-
nt may be made in the case of a series of chutes along a
ch of which it is proposed to run short, independent sur-
for this purpose can be set in line with sufficient accuracy
drift. In dodging corners in a crooked passage, a longer
be obtained by making an abnormally low or high set-up,
le temporary discomfort to the instrument man; this is
inclined workings, where projections in roof and floor,
ls, have to be dodged. In passages where traffic is heavy,
so low that a projecting spad would be in the way, stations
vall. In setting a station at which the side auxiliary tele-
(Art 4), an additional allowance of space must be left
he line of sight.

In important surveys, and where permanency is desired,
ixed in the rock roof of the workings. For temporary
be fastened to timbers, or may be set on the ties or rails
floor; but in such cases, permanent roof stations should be
ntervals. This is best done by drilling a hole in the roof,
plug, and driving a spad into the plug. To save labor of
uld be as small as possible; there is no advantage in having
If the roof is high, and the rock so hard as to require
a hole of 1 to 2 in diameter may be unavoidable. A special
diameter, drawn out small at one end and sharpened to a
in wide, used with a 2-lb hammer, is good for this work.
ome coal mines a twist drill will bore rapidly. In any case,
ot less than 1.5 in deep.

f soft wood, preferably turned in a lathe, from dry stock,
from wet wood; the former plug is less likely to drop out.
plugs in creosote, to prevent decay. Still greater perma-
by using plugs made of soft lead. Plugs should be so pro-
of the holes that they will drive in snugly without mashing,
e roof.

and composition of spads in general depends upon the importance
survey. In a wet mine, especially if sulphur be present, a steel
e than a few weeks; a soft-iron spad will last a little longer, but
mate corrosion but copper, brass, or zinc. A spad should be so
at a cord of any size, at any time, will be obliged to hang in exactly
spad in the form of an open hook is better than one having an eye,
e hanging and unhanging of the plumb-bob.

eties of spads are the following (Fig 1): (*a*) Screw-eyes of
ron screw-eye, with a large eye, is objectionable b

rust may prevent the cord from hanging always from
one having a small eye is generally too short on the scre
the plug. (*b*) Staples and screw hooks; open to the sa
eyes. (*c*) Finishing nail, with the head bent to form a
while it lasts. (*d*) Flat-headed wire nail with a notch i
accurate position, but it is not easy to adjust quickly the
cord for setting up a transit. This drawback can be o
short piece of cord with a loop in its lower end, through
be passed; the upper end is wound around the nail
with head flattened, and a hole punched or bored throug
but subject to corrosion. This spad is improved by mak
point down, either by filing, or by using a triangular p

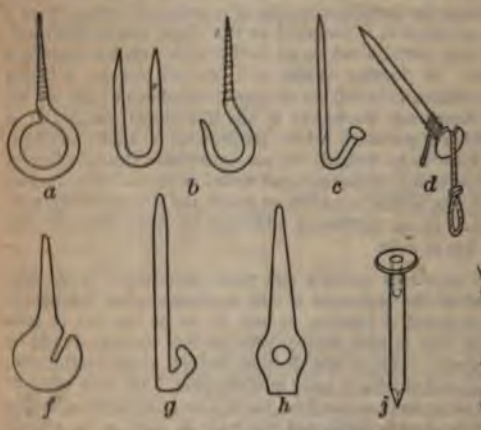

Fig 1. Varieties of Spads

The long point of the nail should be cut off, leaving a
(*f*) Flattened horse-shoe nail with a notch sawed or fil
sawed, the slit should be made wide enough to allow f
gomery's spad, sold by survey supply houses; has the
centering and easy manipulation of cord; could be imp
brass instead of soft steel. (*h*) Specially designed spads,
cite fields, cast from copper, brass, or zinc; are extra st
in acid water. Tinned iron spads of the same design are
of fine copper wire (No 26 B & S) inserted into a $\frac{1}{16}$-i
drill, and held up by inserting a shoe-peg: limited to sol
able nor easily manipulated. (*j*) Floor spad, as used in th
copper mines, is made from a 3-in wire nail, by drilling
the head, large enough to stick a match into. The nai
tie, a match inserted into the hole, illuminated by two
a sighting point; subject to the drawbacks of all floor st
unreliable when placed in a track-tie. (*k*) A boiler ri
and length, split part way by a saw, makes an excellent flo
so high that roof stations are impossible. A hole is dr

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with a steel wedge to spread it, hammered snug, and the head with a center-punch.

rked most frequently by red or white paint, applied with A small paint container, holding only enough for one day's ied, as the paint quickly becomes dirty. The 1-lb carbide rlene lamp, having a tight screw cover, is excellent for this ud, or dust should be cleaned from the rock before apply- his, the flat bristle brush used for cleaning sinks is service- marks will prevent confusion of spads; thus, circles for es for benchmarks, crosses for line points, etc. The num- may be painted near it, or it may be stamped with steel of sheet copper, lead, brass, or zinc. When a tag is used, riven through a hole in it, rather than to hang the tag from

bering has many advantages, though in some important umbered serially in order of setting, and a reference index fy them. The advantages are: (*a*) Knowing where to go to a given number. (*b*) Knowing where you are by the num- also simplifies the notation of samples and assays. (*c*) Being ation even if its tag or painted number has vanished. be adapted to the particular mine. In general, it is a good or 1000 numbers for each level; if the workings extend in ns, odd numbers may be given to the stations on one side ven numbers on the other. The numbering of stations has been suggested (1), with a letter or Roman the level. This involves delay until the co- computed, and the use of unwieldy numbers

2. Illumination

nt. (*a*) By candle or lamp held in back of a behind the plumb-bob string. A screen of in a dry mine; in a wet mine oiled paper or fabric can most conveniently be stretched embroidery hoops, of not more than 4 in diam- ene lamp, having a 3-in round reflector, or a a white enameled face made for this purpose. ht, no translucent screen is required, as the ight background. (*c*) Special lamp targets, oil or acetylene, used only in connection with od of transversing, described later. (*d*) Plum- sperm or lard oil. These are so constructed int of the bob, and the wick, hang accurately The hook may be hung directly in the spad, cord, by which its elevation can be adjusted. a clear, still atmosphere, the flame affords an oint; with shorter sights, the wick tube is ying in gassy coal mines, a plummet safety- le on the same principles. (*e*) Specially de- of various forms are in use in many mines, to economize the services of a man to hold the e most satisfactory mechanical backsight is a light tripod, lene lamp, which can be quickly set behind the cord of an

Fig 2. Plummet Lamp

ordinary plumb-bob. Instead of a lamp a candle sconce havi screen is often used.

Of cross-hairs. With short sights against an illuminated with long sights against an acetylene lamp, no special illuminat hairs is necessary. When illumination is required, the oblique supplied by all instrument makers, for attachment to the end is entirely satisfactory. A similar reflector can be improvis visiting card and putting it into the front of the ordinary tran

Of verniers, etc. Instrument readings are generally mad form of light the surveyor carries. Magnetic readings must cand'e, or a copper lamp. Electric storage-battery lamps ar reading verniers. Instrument makers sell a combination el magnifying lens for this purpose. If a plain magnifying lens i have a metal case, not celluloid. For all-round use nothing exc lamp, with parabolic reflector. The makers supply a specia which points the flame upwards, so that it is less likely to injur held close to it. The objection of some surveyors that an especially of the larger sizes, does not leave the hands free to op ment, can be overcome by hooking the lamp on a tripod leg, or or collar of the coat, or on a strap around the neck.

3. Transit Mountings, Plumb-bobs, Etc.

Tripods. For underground work tripods having extens necessity. The clamps and rings should be extra loose, to al of the extension piece in the mine dampness. The steel po should be sharpened from tin working on inclined rock floors. allowing the center of a transit fortably within 30 in of the flo convenient under low roof. set-ups, or in awkward place bered stope or a shaft botto pod (Fig 3) can be set on a plank.

Fig 3. Trivet

Bracket. In timbered drifts, where a tripod would be in th and in constricted places where no tripod can be set, the bracket is useful. A sound and firm timber must be selected; a hole is bored horizontally at the proper spot, and the bracket is screwed snugly into place with a steel bar fitting through the hole provided for that purpose. The screw-plate is then leveled roughly by the clamp joint, the transit is screwed on, leveled accurately, and brought exactly under the point by the sliding head.

Fig 4. Transit

Plumb-bobs. In draughty workings heavy bobs, weighing 14 to 18 oz, should be used. The instrument bob should preferably shown at *a*, Fig 5, with a fine point. For sighting bobs t

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e upper end gives a more definite point to observe on
Bobs with interior reels are unreliable in underground
soon gets wet and dirty; neither are

should be braided, not twisted, to
bob, and raveling at the end; for the
ving a tubular weave with a core is
is better than silk for underground
knot is easier to untie. When a cord
knot, discard it. Adjusting devices
buttons, tend towards inaccuracy and
se; a sliding bow knot can be tied in
a single pull on the loose end, and
one hand.

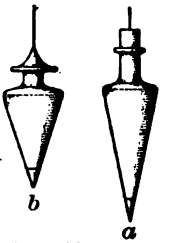
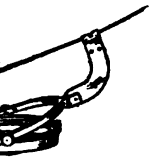


Fig 5. Plumb-bobs

4. Measuring Horizontal Angles

to the omnipresence of steel rails, pipes and cables, trolley wires
d signaling circuits, in modern mines, the compass can not be
ct observation of bearings. For accurate work, also, readings to
ee are not close enough. For less important work the compass
n angle by noting the bearings of both the lines forming the angle,
l attraction will not affect the accuracy of an angle reading made
ct bearings are then computed from the angles, on any desired
nount and direction of magnetic declination is immaterial, and
nderground surveying.

ass (Fig 6) has great utility for traversing through tortuous
topes. A strong linen cord is stretched from station to
station, which may be merely of temporary character, like a stick braced from wall to wall. Special brass arms are provided for screwing into timbers, but a heavy nail answers as well. The compass is then hung at any convenient point on the cord; if the cord is inclined, clips are provided to keep the compass from sliding. For reading angles, where local attraction is suspected, a special


ng Compass

provided, permitting the compass to be brought under the
he bearings of both lines from the same position.
e of compass the circle is graduated from 0° to 360°, with
rdinal points.

ocket transit" (Fig 7) is a magnetic compass designed for
ld in the hands while taking observations. The cover is
y to the sighting axis, and has a mirror on its inner side,
is scratched in continuation of the *N–S* axis; the cover and
ith two small holes on this line, one near the hinge, the
ite edge. On the side opposite the hinge is a folding vane
In use, the compass is held vertically below the eye, and
e cover opened about 135°, the level tube turned at right
ght, and the sight slot lifted to the vertical position. Look-
or, the operator can then see the reflection of the forward
time observe that the instrument is level: the compass
beam of light from the other station is seen to pass through

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(*a*) The construction should be substantial, and not to
less th
ments
servic
groun
sit is
the sm
its la
winds,
horizo
large c

Fig 7. Brunton "Pocket Transit"

should preferably be graduated and numbered from 0°
only. If graduated in both directions, the figures on
inclined to right and left, to indicate in which direction
be read. Graduations should be marked with extra plai
cise work, two verniers are desirable on the horizontal circ
minutes; finer graduations are confusing in undergrou
should be offset not more than 30° from the *N*–*S* axis, t
ing around the instrument to read it. (*d*) Compass ma
transit intended solely for underground work; it is us
read it, though many surveyors rely upon it to check
readings, and the transit can be made stronger with
should be a complete circle, of same size as the horizon
0° to 90° and back to 0° in both directions. Two verni
most accurate work. The whole vertical circle shou
level tube under the telescope should be extra long and
that both ends of the bubble are easily visible when
on the horizontal axis is recommended for accurate wor
be inverting, to secure better vision where poor illum
prismatic eye-piece is necessary for reading plus vertic
(*f*) Stadia hairs are objectionable in a transit intended

imuth method. For methods of adjusting transits see Sec 17, d work the most important adjustments are: (*a*) Plate bubbles. to collimation. (*c*) Vertical verniers to read 0° when telescope axis truly horizontal; this is particularly important because of ghts. A sensitive striding level attached to this axis is a valuable es.

The practice of transferring a roof point to the floor, this point, should be condemned. It introduces inaccur- is no easier than setting under the roof point, as a little e with the latter method will prove. Hang the plumb- ills under the center of the spad, not tangent to one side verniers at 0°. Move the tripod under the plumb-bob by the legs, keeping the head of the tripod fairly level and under the point, regardless of the transit itself. Level ljust the plumb-bob until its point almost touches the elescope, and observe the direction and distance by which Loosen all leveling screws by the same amount, and tripod head, without turning it, until the points coincide. level accurately by means of the long bubble and striding sary to repeat the shifting operation once more.

sing. The manipulation of the transit, and the several have been fully described in the preceding section, nty can be avoided by adopting an invariable rule for the of all angles, whether exterior, interior, acute, or obtuse. t angles be read always clockwise, sighting first at the bly recording in the same manner as read. The record 178° can then have but one meaning, namely, that the on 9, backsighted on station 8, and turned 178° to the deflections or azimuths are read, instead of angles, it is orded reading refers to the course indicated by the last rce of circumstances it is more convenient to read an , in the direction opposite to that in which the traverse e angle should nevertheless be recorded as read; the ould be designated plus or minus, as read, leaving it to pplementary angles and transpose signs into conformity se.

This is the method most commonly employed in under- ly a medium degree of accuracy. For the actual process s *advantages* are: (*a*) More rapid than other methods. lculation and plotting, especially when numerous side points off the traverse. (*c*) An error of closure in the scovered as soon as the final course is read. Its *draw-* uracy of any one angle reading is limited to 30 min. kept in perfect adjustment by daily inspection, because umulative. (This objection can be overcome by taking lescope alternately direct and inverted, although, by so e distorted by the amount of the instrumental error.) error is discovered, there is no sure way to ascertain or occurred at one set-up; nor, if the latter, at which hecking a single error at a station is the only argument ings, but under *modern conditions* the compass is more *the observer.* *Extreme carefulness* on his part is the *y.*

Deflection traverse. The vernier being set at 0°, at the back station with the lower motion; then ere motion to point at the station ahead. Record the defl ing the lower motion, turn the instrument back to t Invert telescope, and turn off the deflection again. divide by 2, and call this the correct deflection angle mental errors are compensated, and the possible error is reduced to 15 sec. The mistake most likely to occ method is to confuse right and left on very small d graduated in clockwise direction only, and deflections read, the mistake is less likely to happen.

Angle traverse. By the system of repetition angle reading can be reduced to almost any desired traversing two sets of four repetitions will give the cor greater than can be attained in the taping of distanc Method of repeating angles is described in Sec 17, Art on preceding page for recording angles. The only di regular and extensive work is that it adds to the bur which, however, is often warranted by the higher deg

Fig 8. Lamp Targets

Three-tripod method requires the use of specially instruments, a transit, and two lamps or targets. The ver inclosed by a thimble which fits into a socket in the levelin clamp and a spring plug engaging a groove on the thimble fit into the same socket, and the length of the lamp standa

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of the wick tube, is at the same height above the leveling head as sit. The lamp standard carries a pair of level bubbles. The best lard oil should be burned if the bull's-eye lantern supplied by the small-sized acetylene lamp with a special surveyor's tip (Art 2) is s bulky, and more satisfactory in every way. It can be held in ut threaded on a short stud soldered to the bottom of the carbide The method of traversing is as follows: While the transit is being sets one lamp under the forward station and one under the back the target, or of the wick, being set exactly under the point of the

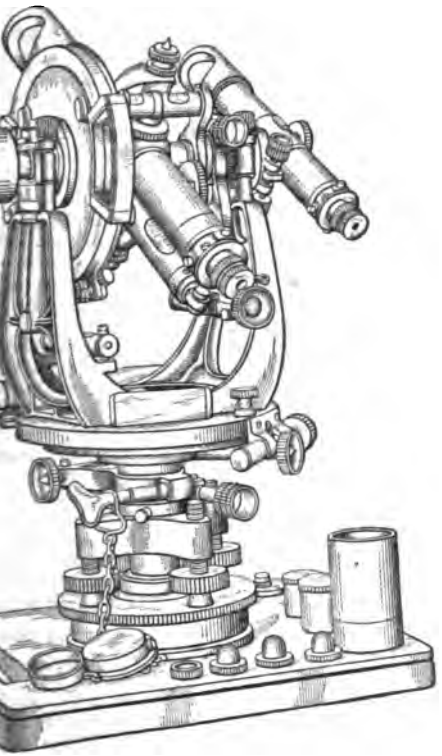

Fig 9. Fully-equipped Mine Transit

eveling head being set level by the bubbles. For centering a target, ontal and the sighting point brought accurately under the bob; the e target turned until it squarely faces the transit point, and the light ument man then reads the angles with as much repetition as he ers are otherwise occupied. Tapings are made from the side center e side center of the target, or to the wick tube. The transit and the en interchanged, above the leveling heads, and the rear lamp and he next station ahead. After the interchange, it is advisable to test e bobs, and for level. The *advantages* of the system are: (*a*) Appli- igh chambers where roof stations are not feasible. (*b*) Save

transit man's time in setting-up. (c) Allows assistants to be
note-taking, or otherwise, while the instrument man is read
degree of accuracy in angle reading limited only by the m
instrument and the personal error of the observer. (e) It
liability of error in setting-up, in measuring inclined distance
in reading vertical angle, since all these measurements can
sight operations, when a failure to check will be noted. Its
of the extra apparatus, and the labor of transporting it. For
mines the refinements of this method are unnecessary, and it
ities in this country.

Steep sights. For sighting at an angle exceeding
when the line of sight of an ordinary transit would be
zontal plate, three expedients are available: Auxilia
telescope; eccentric telescope.

Auxiliary telescope may be attached either to th
horizontal axis, or to a lug on top of the main telesc
attached to the opposite point to prevent distortion an
The following ADJUSTMENTS must be applied to the a
To bring the two telescopes into the same plane: Aft
iary snugly, sight the main telescope at some sharp poi
Then, with its capstan tangent screws bring the auxi
point, with its vertical hair if above the main telescop
hair, if at the side. This adjustment must be made ev
used.

To make the two lines of sight parallel: Determi
centers of the telescopes, by direct measuring, or by
sight
away.
rod,
space
200 o
cross-
til th
cepte
sight.
sary
excep

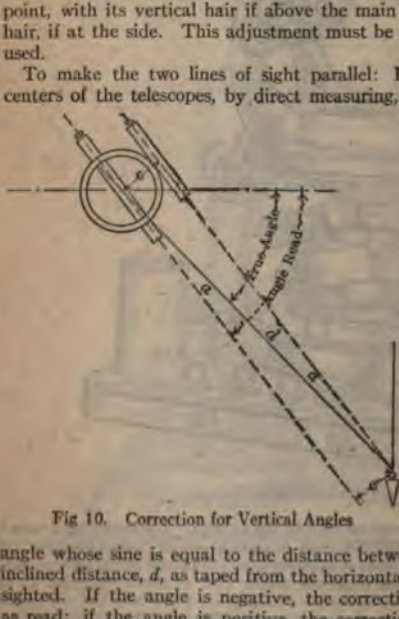

Fig 10. Correction for Vertical Angles

To
auxili
the m
angles
direct
Vertic
rected
Fig 1
correc

angle whose sine is equal to the distance between tele
inclined distance, *d*, as taped from the horizontal axis o
sighted. If the angle is negative, the correction is su
as read; if the angle is positive, the correction is ad
correction thus have the same sign.

Side telescope. When using the side telescope,
directly, as usual. In the single reading of a horizon

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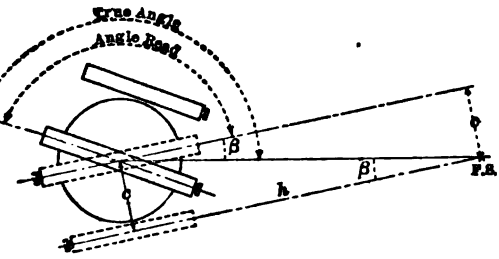
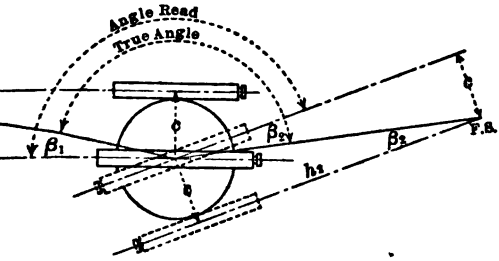
by the main telescope, by reference to Fig 11 it is obvious is that angle whose tangent is c divided by the horizontal t sighted; this horizontal distance, however, is equal to the ltiplied by the cosine of the vertical angle. That is, *tan e × cos vert angle*). If the auxiliary is on the right of nd if the angle is read clockwise, the correction must be



ion for Horizontal Angles; Backsight with Main Telescope

read. If the side telescope must be used for both backsight rrection for each sight must be computed separately; from g 12) first subtract the correction β_1 for the backsight, and on β_2 for the foresight, the auxiliary being on the right, and ise.

zontal angles with the side telescope can be much simplified of corrections eliminated in the following manner: Bisect



tion for Horizontal Angles; Backsight with Side Telescope

ither the main or the side telescope, turn the angle and vith the side telescope; invert the telescopes and turn the ne manner as before, without resetting the vernier at zero. ading will be the true angle. Referring to Fig 13, the angle $+ \alpha - \beta$; the second reading = true angle $- \alpha + \beta$; second reading = 2 × true angle. Instead of making the ely, they are added automatically on the circle of the

erse is being run, and the use of the auxiliary telescope is uired, the angles at such set-ups can be quickly determined

with the side telescope by the above method, the azimuth coe
traverse continued, after setting the vernier at the computed a

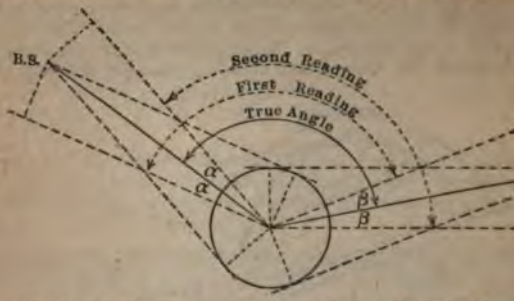


Fig 13. Reading Horizontal Angles with Side Telescope, withou

Prismatic telescope. By placing a mirror or prism, which will
sight by 90°, on the objective end of the telescope of an ordinary transi
of steep sights may conveniently be read directly on the horizontal circ
vertical angles by this means involves two difficulties: (*a*) The refl
exactly 90°; (*b*) the movement of the objective, for focussing, alters the
city and thereby the amount of correction to apply to the reading. T
prismatic telescope surmounts both of these difficulties; first by its
constructed as to cause an invariable reflection of 90°, and second by it
device, which renders unnecessary the moving of the objective. The
tached to the main objective, or to a side objective mounted at the rig
horizontal axis, which is hollow. In the latter case the line of sight is
time by a smaller prism which can be inserted, when desired, into the
telescope by turning a milled head on the left-hand end of the axis.

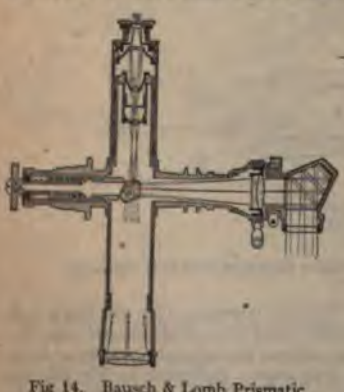


Fig 14. Bausch & Lomb Prismatic Transit

Fig 15. Correction for
by Prism on Main

Adjustments and corrections. Prism on main objective. Be
prism, sight the telescope on a sharp point at as great an angle as
below the transit point, and bisect accurately with the vertical hair;
the transit remaining clamped, attach the prism to the end of the t

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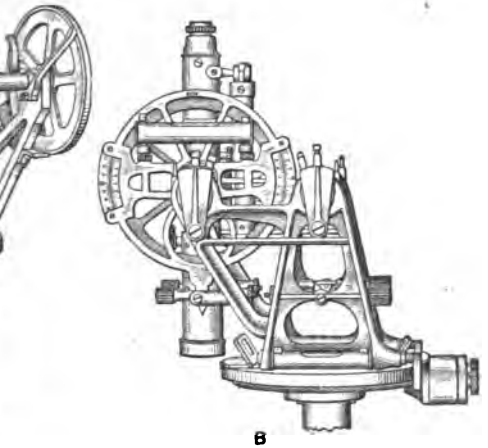
l by means of its tangent screw turn the prism until the point ertical hair. Horizontal angles can then be read directly. The le by this method involves two computations (see Fig 15). has the sign opposite to that of the angle β as actually read. $-\delta$, and $\sin\delta = e + d$. If the backsight, for example, is steeply e instrument station, and the foresight downward, it is more telescope for the backsight and read the horizontal deflection, ngle, which would involve adjustment of the prism for each

ve. Sight the telescope on a sharp point, and bisect accurately Attach the prism to the side objective, and bring the interior the former by its tangent screw until the point is again bisected. e read directly. Horizontal angles are corrected, or corrections ely the same manner as with an auxiliary telescope.

Several manufacturers make a transit on which the main ently, or may be temporarily, offset from its usual position, to vertically, if need be. One of the former type, made by Young nd one of the latter type, made by C. L. Berger & Sons, at *B*,



ig 16. Two Types of Eccentric Transits

transferable telescope precautions must be taken to assure the ontal axis, by means of a striding level; the coincidence of the e with the central position of the long bubble should also be l angles are to be read by duplication. Both horizontal and directly. The instrument should be centered under the station, should be taped to the eccentric axis; vertical distances are then computation, and the only correction to be applied relates to which must be increased by the amount of eccentricity of the city.

5. Measuring Vertical Distances

In most metal mines, and in many coal mines, vertical distances, a obtained by computation on the basis of vertical angles; the leveling m used, but even then the leveling is commonly done with a transit. Wh has to be done, it is advisable to use a Y-level, because in important respective elevations of various points should be known with no less a horizontal coordinates.

Computation from vertical angles. Angles below the hori negative, and those above horizontal are called positive. In F taped distance from the center of instrument at A, to the point α is the vertical angle, with its appropriate sign carefully noted the vertical distance is: $v = l \sin \alpha$, and takes the same sign a of instrument, $H.I$, or i, is the vertical distance from the cent up to the bottom of the spad, and the height of sight, $H.S$, o sponding distance from the point sighted at up to its spad. T elevation between the two spads, d, is thus the algebraic sum: That is, when the vertical angle is positive, add v to s and s

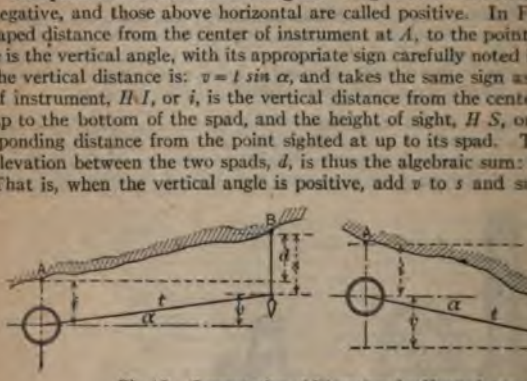


Fig 17. Computation of Elevations by Vertical Angl

the angle is negative, add v to i, and subtract s. Sometimes will be read, while the actual difference in elevation is positive, but if the algebraic signs are kept firmly in mind, no confusion wil ing the elevation of A, the elevation of B is computed by the El B = El $A - i \pm v + s$. In practice it is usually more accura but little more office labor, to read vertical angles to a defini plumb-bob string (say, the top of the plumb-bob) just as they c to attempt to set the sighting point at the same distance below instrument is below its point. The $H.I$ and $H.S$ are read with a pocket steel tape graduated to feet and hundredths. In me of a transit it is customary to tape obliquely from the bottom o center of the end of the horizontal axis; this does not introd unless the distance is short. The following table gives the corre tracted from the oblique distance as measured on a transit of w the axis is 0.25 ft from the center of the telescope:

Corrections to Subtract from Oblique Measurements

Oblique distance	Correction	Oblique distance	Correction	Oblique distanc
0.6	0.091	1.1	0.026	1.6
0.7	0.066	1.2	0.022	1.7
0.8	0.050	1.3	0.019	1.8
0.9	0.039	1.4	0.016	1.9
1.0	0.032	1.5	0.014	2.0

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rn in Fig 18 is a companion instrument to the hanging ung on the same cord before or after the compass has ttached to a fine fastened through er of the arc. Ow- the cord, and the itself, it is neces- lings, equidistant se thereto as pos- ge. If the slope °, a single reading will give the cor-

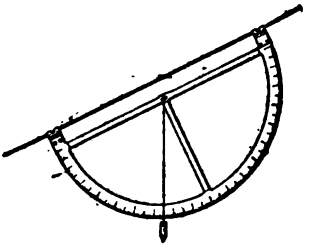

Fig 18. Hanging Clinometer

sit (Art 4), when l angles, is oper- tted sighting arm to its fullest extent, turning up its e sighting hole; also open the lid about 45°. Standing e body of the instrument vertically and sight through slotted arm, and through the round hole in the edge of to a light at the next station. At the same time move g the level vial until the bubble, as seen in the mirror, en read the vernier on the graduated end of the arm. d clinometers are just as satisfactory for underground

transits. The use of these instruments for reading vertical 4, in connection with horizontal angles, since the correction s the reading of the other.

rtical angles by transit. Since vertical angles can not e manner as horizontal, they must be duplicated and al angle of a traverse of any importance should be read the telescope normal and once with it inverted, and der to eliminate zero error of the vertical vernier. For isable to read both verniers, in both positions, and take the average of the four readings; this further compensates any eccentricity error in the vertical circle. While traversing by repetition angles, or by double deflections, no time is lost by reading vertical angles with the telescope in both positions; on an azimuth traverse, the only check is to read the vertical angle on the backsight as well as on the foresight, making two complete sets of readings of the taped distance, the *H I*, and the *H S*. The auxiliary telescope, when attached to the horizontal axis, gives vertical angles direct; when on top of the main telescope, a correction must be applied (Art 4).

Underground leveling is performed in a manner the reverse of that employed on the surface. The zero end of the rod is pressed upward against the bottoms of the spads, or other desired points; backsight readings are thus negative in sign, and foresight readings Sec 17, Art 18.) Elevation of a point on the floor is od right end up and recording the reading as negative

The form of notes and the method of computing elevati
like those used on surface work. A self-reading rod
ployed, unless the air is clear, the sights short, and a
The target for underground service should have a ¼-
cut on the horizontal line, behind which a light is held.
extending to 5 ft, is a convenient length for mine work

6. Tapes and Taping

Underground conditions require a tape to be light in
ments are usually made with the tape unsupported
strong and durable, to withstand unavoidably rough t
acid water; and its numerals to be as large and disti
mistakes of reading in a poor light. If the graduatio
or brass sleeves, these should be soldered in place, as wel
water out. Riveted sleeves are not recommended.
tapes are most resistant, but expensive. Tapes shou
constant pull, say 20 lb, and for a constant temperature
necessary to apply corrections for temperature, except
accurate work, because mine temperatures are fairly
necessarily corresponding to the standard temperature o
and temperature corrections see Sec 17, Art 11. It i
surveying to tape a distance exceeding 300 ft, and for
of 100 or 200 ft will be found most satisfactory. A c
the tape anywhere along its length adds to convenienc
acy of the work, besides avoiding the sharp bending o

Graduations are most commonly at 5-ft intervals, m
sleeves. Exact distances must then be ascertained by
a pencil, or by holding the thumb-nail on the point, and
5-ft mark with a pocket tape. This is a troublesome
be inaccurate. The difficulty can be diminished by ha
tape 10 ft from the end, the outer length being gradu
foresight man then holds the nearest 10-ft mark at his
measures the exact distance with a 6-in rule graduated
0.50 ft on one side, and from 0.50 to 1.0 ft on the other,
in either direction to the nearest foot-mark. Inaccu
best overcome by using a steel ribbon tape graduate
throughout. Such tapes are more expensive than fla
more cautious handling.

Taping practice. Distances should ordinarily be ta
from the center of the horizontal axis to the point sigh
vertical angle, usually the top of the plumb-bob, on t
exception to this rule is in the case of nearly horizonta
are subsequently to be obtained by leveling; the tap
on the floor. For distances up to 100 ft a steady pull
a spring balance being used until the operators becor
pull. For greater distances the necessity for applyi
can be avoided by taping "on the swing." The tape h
clamp handle adjusted to a comfortable position; it is
brought taut by a steady but rapid pull. This bring
into a straight line, above its normal sagging curve, at w
distance can be noted. This method requires conside
ings should always be repeated; it is also open to the

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ard pull. For long distances it is more accurate to use a
a catenary correction.

rground work should be larger and stronger than for
e shown at *a*, Fig 20, be employed, for a ribbon tape 100
have a capacity of 150 or 200 ft, to prevent clogging by
omination for mining work. Type *c* is as simple and
as been devised. It is made of sheet metal, preferably
the shoulders on the dividers, plenty of metal should be

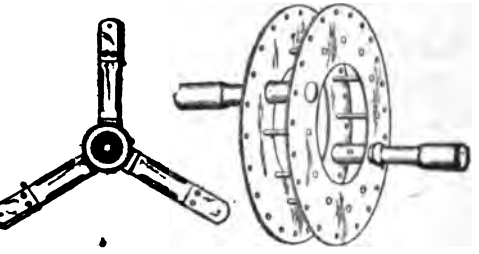

Fig 20. Tape Reels

rm the riveted heads. The two wooden handles should
prevent them from turning inside the ferrules and wear-
p out. A reel of this type, of 10 in diameter and 2 in
flat-wire tape. So much of the tape as is not needed for
he reel by cord or leather thongs.

es. Tapes used underground should be cleaned, dried, and
tape breaks, it is not necessary to send it to the maker for
es of soft sheet brass, about 3/8 in long and of a width sufficient
ith a double lap on one side (1/2 in is about right for a flat-
are tinned on one side, and then folded as shown in Fig 21.
roken ends of the tape, apply zinc
with a soldering iron for a distance of
break. Fit the broken ends closely
brass sleeves, close the latter with
apply more flux, hold a hot soldering
solder flows, add a little more solder
d finally wash off carefully any excess
t rusting. A neater job can be made by cutting out the 5-ft
occurred, and inserting in its place a 5-ft length cut from an
e figures on the new sleeve. This plan avoids multiplicity of

Fig 21. Brass Sleeve for Repairing Tapes

7. Underground topography

thod is best suited to rooms, entries, drifts, and narrow
is stretched between stations, either before or after the
ired, lay it on the floor in a straight line approximately
th the zero end preferably at the instrument. At every
e the distance at right angles from the tape to the walls
d, a rule, or a linen tape. If the object of this work is
olumes, as a basis for payment of wages, for example,
d to the nearest tenth of a foot, and considerable care
neasure at right angles to the line of the traverse. For
easurements need be only to nearest foot. If all offsets

are measured at about waist height, errors due to irregu
pensate themselves in the long run. The readings m
to scale on the right-hand page of the note-book, v
ordinately, with a center line to serve as the line of si
be recorded to the right and left, opposite their appro
line. Another way is to record the readings thus:
50-ft point the right wall was 3 ft and the left wall 5 ft
to roof and floor are taken at instrument points by d
intermediate points by a leveling rod sighted from th
bisected at each point by the line of sight. Roof and fl
in the form of fractions, as above, but should be distin
drawing a circle around them.

By angles and distances. This method is appli
bers, and lends itself particularly to the azimuth meth
set up at a station and oriented the lower plate by a
light held at as many points as desired, recording
angle, and the distance to each. As these notes are
tractor, duplication of angles is not necessary. Afte
taken, observe the azimuth and distance to the nex
the traverse is being run by deflections, or by angles
side shots is the same, but in plotting the notes, the
each station with its zero on the line of backsight.

By tape alone. The method is suitable only fo
obstructed by pillars or timbers, in which all of the poi
From each of two traverse stations measure the dista
and plot by swinging intersecting arcs from the pl
the situation of the points referred simply to their ow
horizontal plane, an observation must be made on th

By angles alone. Two traverse stations are re
all the desired points can be observed. With the ve
station and sight the other station with the lower n
both horizontal and vertical, successively to each
up on the other station and read angles to the sam
directed to the first station. Time can be saved
sighting simultaneously at the same point. Plot with
lines. If elevations are required, they can be con
horizontal distances, scaled from the map, by the tan

If the points in the chamber are inaccessible for the man
expedient has been found successful. Two transits are re
stations commanding as much as possible of the excavatio
oriented. Take the ground-glass plate out of the back of
place an acetylene lamp. This will throw a narrow beam o
can be projected against the roof and walls of the chamber,
simultaneously by the two transits. Plot and compute
another station may insure a more complete capture of all t

The Tonnesen method (2) was devised for estimating
gold mines, where the slope is often regular for long dista
area excavated. It aims to reduce the office labor involv
distances to the horizontal basis, and then reprojecting to th
calculated. Tonnesen's instrument, the "stereometer," is
the nautical sextant, and reads angles in the plane of th
accurately located points are set in the stope, in convenient
the faces of the stope are taken either by angles and dist
intersecting angles from both stations. Or, a traverse m

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e, offsets being measured to the solid ore. If the uniformity aults or folds, the area is divided into panels, each having a eparately measured.

8. Shaft Plumbing

plumbing are of iron, steel, copper, brass, or phosphor he size of the wire should be as small as will safely ob; B & S No 20 or 24 is convenient. Iron wire is easily lity permits the bob to stretch out its kinks without bing has to be done, at intervals, iron wire is liable to Steel wire (piano wire) is much stronger, but is not must be handled cautiously to prevent kinking, and is wire is easy to get, is ductile and non-corrodible, but as carry much weight, and therefore can not be stretched aft is more subject to disturbance by air currents and e is stronger than copper, but is subject to kinking, sion its springiness makes it troublesome to untangle. strong, durable, and not too springy; it is not quickly ve. In general, iron and copper wires are more suitable om rapid air currents and falling water, while steel e better adapted for deep shafts and troublesome

er from flat-irons and sash weights to carefully turned , and in weight from 10 to 125 lb. Irregular-shaped because in twirl- ng motion to the difficulty in de- ition. Cast-iron of being suscep- nces, tending to r true position. n a case where a magnetic merid- e to an iron bob. ence on iron bobs Lead bobs, used wires, are wholly The Handley (3) shell, accurately

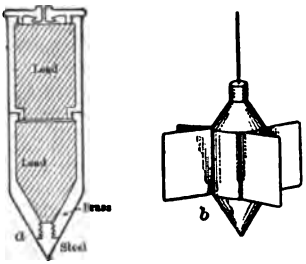

Fig 22. Shaft-plumbing Bobs

threaded connections, permitting a bob of any desired . (Fig 22, *a*.) The brass shell is poured full of lead, lb. An iron bob fitted with fins to reduce twisting nerged in a bucket of liquid, is shown at *b*. The fins jacket closely fitting the bob. An excellent bob can be eel shafting.

top. If the shaft survey is only an isolated case, the ce for fixing the position of the wires at the surface. a notch in the edge of a board or in the head of a nail, be kept on a spool. If the plumbing operation is uently, it is well to provide better designed and more s convenient to support wires in such manner that dily shifted to bring them into line, or to vary their ep the excess wire on a reel, where it can not be injured.

The apparatus used at Butte (a) is shown in Fig 23, bracket is shown in Fig 24. Another type, and the end a

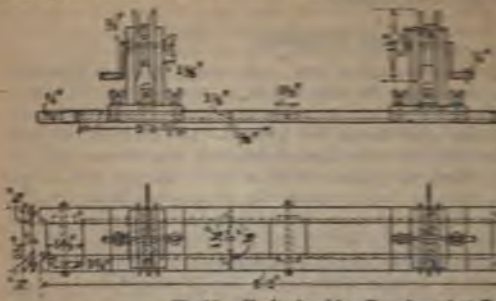

Fig 23. Shaft-plumbing Board used at B

it, are shown in Fig 25. Other still more elaborate forms are not necessary for ordinary cases.

When two vertical shafts with connecting workings single wire in each shaft is needed, preferably hung near determined the coordinates of the two wires by a closed s underground at a station as far from wire No 1 as it can be the mean position of the wire. This operation need not be performed with such extreme precision as when only a single shaft is available, since the base-line for the underground survey may be several hundred or thousand feet long. Assume any bearing for the line from the wire to

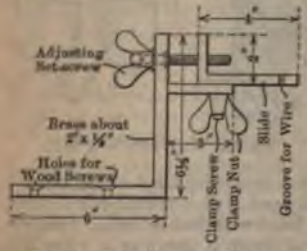

Fig 24. Shaft-plumbing Bracket

Fig 25. Shaft-plu

the first station, and carry the traverse on that basis through No 2. Compute the coordinates of wire No 2 with respe underground notes, and from these calculate the length and line connecting them (see Art 9); the length should cor similarly computed length on the surface, as a check on t Compare the computed underground bearing of this line w determined on the surface, and correct all underground be

Hanging two wires in one shaft. If only one verti for connecting an underground survey with the surface, tw

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e in this shaft, and their connecting line used as a base for
ey. Extreme care is therefore necessary in every detail of

experiment (5) that wires are liable to deflection if an air
er than 100 ft per minute is present in the shaft. The
is proportional to the velocity of the air current, but
s up or down the shaft is immaterial. A wire hung in the
shaft is more strongly deflected towards the long side of
wo wires be hung in opposite corners, their deflections
ctions and the bearing of the line connecting them at the
displaced. This is a more serious source of error than
he wires. It is therefore recommended that in a shaft
nt of air the wires should be hung at about the middle
ather than in the corners. If possible, the ventilating
ced or stopped while shaft plumbing is going on. If this
ortant work, board flues can be built from the top to near
ft, and the wires lowered inside of them; these will afford
lling water. A light bob should be used for lowering the
obs attached below. Freedom of a wire from obstructions
) By watching, from above, a light slowly rotated around
aring the swing period of the bob with the correct period
nown length of the wire; (*c*) by wrapping a little ring of
e plumb wire at the top, and letting it drop; (*d*) by caus-
ort to be moved a definite distance and direction at pre-
time, noting whether corresponding movement occurs at

As a means of detecting angular displacement of the line con-
in the bottom of a shaft, it has been suggested (6) that three bobs
les be read until the three wires at bottom are precisely the same
top. If this be the case, since it is highly improbable that all
been displaced by exactly the same amount, and in the same
sitions of the bobs can be accepted as correct.

op of shaft. It is usually convenient to bring the transit
es; if not, their respective positions may be triangulated
ed on next page. Having hung the wires approximately
nsit on a point from which both wires can be seen. If the
is adjust-
ought into
y sighting
the other;
the transit
line with
ead. The
n in Fig 26

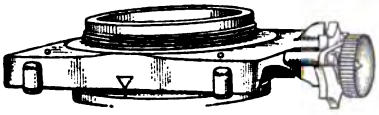

Fig 26. Micrometer Head

ation. All being set, measure the distances from wire to
wire to the transit point, and from the transit point to
on. Also measure, by repetition, the angle from the last
the wires. From these data compute bearing of the line
dinates of either or both wires.

ottom of shaft. If a suitable instrument station can be
ot, the transit may be lined in on the wires; if not, their
ngulated. If the shaft is not deep, and is free from air
ater, sights may be taken on the free wires; otherwise,

the mean position of each wire must be ascertained, and the position, or the point itself be used for sighting. In any c screen must be fixed behind each wire.

Alinement is obtained by a series of approximations, as at micrometer head is particularly helpful for this purpose. It i the nearer wire at a point about level with the transit and in light iron chain, through the links of which the farther wire c

Triangulation. First measure the distance between wires a divergence of the wires, it is highly probable that the lin is also deflected in azim sary means must be e the interference of air c (See above.) Set the as nearly in line with venient. Measure the transit to both wires, underground station, a each wire to that station ments are not all essent to check one another. *B* are the wires and point, and *a*, *b*, *c*, are these angles, the kno triangle are *C*, *a*, *b*, *c*. then be computed in se

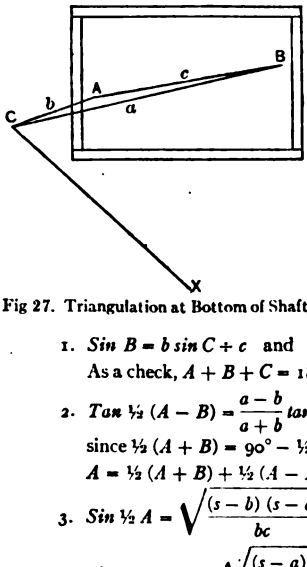

Fig 27. Triangulation at Bottom of Shaft

1. $Sin\ B = b\ sin\ C + c$ and $sin\ A = a\ sin\ C + c$
   As a check, $A + B + C = 180°$

2. $Tan\ ½\ (A - B) = \frac{a - b}{a + b}\ tan\ ½\ (A + B) = \frac{a - b}{a + b}\ tan$ (
   since $½\ (A + B) = 90° - ½\ C$.
   $A = ½\ (A + B) + ½\ (A - B)$ and $B = ½\ (A + B)$ -

3. $Sin\ ½\ A = \sqrt{\frac{(s - b)\ (s - c)}{bc}}$, where $s = ½\ (a + b +$
   Also, $sin\ ½\ B = \sqrt{\frac{(s - a)\ (s - c)}{ac}}$ and $sin\ ½\ C =$

Formula 1 gives more precise results for small values of advantage of setting the transit closely in line with the wires; a is that the error introduced by inaccurate measuring of the dis

Formula 2 should be used when it is not possible to set up in line with the wires. Formula 3 is independent of the angl as the transit must be set at *C* to read the angle to the next st be read as well. Knowing the bearing of *A*–*B* from the surfac the bearing, first of $A - C$ or $B - C$, and then of $C - X$, and nates of *C* and *X*.

Sighting on the free wires. The amplitude of oscillation and vibration can be reduced by immersing the plumb-bob in a pail of w At Anaconda (7) they use a pail 18 in deep and 10 in diameter, having at the bottom with its outlet 2 in below the top of the pail. The pail this level, and ½ in of engine oil is poured on top to prevent the splas Further protection is given by a split conical lid with a 2-in hole. not to allow the bob to touch sides or bottom of the pail.

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position of wire. (*a*) If the period of swing gives
ration, the wire may be followed with the upper tangent
and the extreme right and left position noted on the
of a series of such readings will give the mean position.
olique cross-hairs, these, in connection with either of the
as a guide in bisecting the amount of maximum dis-
ging wire, after which the mean position can be marked
Fix a strip of clean wood behind the wire, approximately
e of sight of the transit, and mark a succession of extreme
and set a fine nail at the mean position. (*d*) Instead of a
graduated rule may be used, the extreme positions being
it and the mean calculated. (*e*) Draw a graduated scale
placed behind the wire, and read deflections as above.
or inserting in the eye-piece of the telescope, as in micro-
suggested. *Note.* — In all the above methods involving
lculate the mean position, it is important to take a large
oo or more are none too many in case of a deep shaft,
falling water. On account of local vibration the bob
te swing, and it is difficult to ascertain the exact point of
rof H. S. Munroe suggests that observations at regularly
give the mean position with greater accuracy.

mean point for purposes of taping, the above methods
observations to be made on each wire, from positions
nearly as practicable at right angles. One set of readings
ire from a first position of the transit, which will then be
t for the second set; or, any kind of a telescope can be
nt, and the readings made simultaneously.

) obviates the use of scales and telescopes altogether.
n a board, and fix it horizontally so that the wire passes
ter of the hole. Cause the bob to swing in various planes
f contact with the circumference of the hole, indicating
d. The usual pail of liquid should be omitted. Remove
h in the hole, draw lines to connect corresponding points,
e majority of these lines intersect.

It is generally advisable, and, if sights are to be made on
f the mine, it is essential to fix the wires in their mean
ethod, under favorable conditions, is to bring the wire
st the edge of a board so set in the timbering as to be
ly against the
en nailed fast,
. If the mean
n observations
imple combina-
and strings will
The Schmidt
eful when much
to be done. It
having a 1½-in
timbers. The
es is calculated

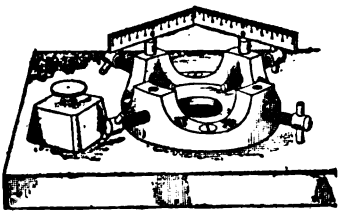

Fig 28. Schmidt Shaft-plumbing Apparatus

two scales. The bob is then disconnected, and the wire
ne hole in the square block; this is then placed between
nd shifted by them until the wire is seen to occupy its
having been attached again. All tape readings can then

be made horizontally, from the transit to both wires, a
thereby eliminated.

Instrumental methods of plumbing. The side or t
transit may be used to project the meridian of a survey
down a vertical shaft; special eccentric transits (Art 4) a
for this purpose. A plumb-bob must also be dropped, t
at bottom, but as a slight error in coördinates is not so
bearing of an underground base-line, the combination of
tion and a single plumb-bob has great advantages. Obse
from either top or bottom.

Transit at top of shaft. Set the transit on heavy t
that it can see to the bottom of the shaft. Sight along a
bearing; then sight down the shaft. Stretch a wire ac
bottom level, as nearly as can be estimated under the tr
plane as the telescope. The wire should have a weight
and at each end should be laid in the thread of a long screw
on a heavy trestle; by turning these screws manipulat
bisected by the transit throughout all its visible length.
bearing on which to start the underground traverse, an
gives the coordinates of a starting point, which may be sit
shaft. Instead of a wire, a plank having an illuminated sig
may be shifted by repeated trials into the plane of the te
then being used as the underground base-line (8). In all
repeated observations should be made, with the telescope e

Transit at bottom of shaft. A method exactly similar t
employed, stretching the adjustable wire across the top of the sh
will be ascertained later. Or, the wire may be fixed in known
brought into the same plane with it by repeated trials. It is no
to be vertically under the wire; it may be tilted by its leveling
seen to lie in the plane of rotation of the telescope, even though
If then two points be fixed in the workings *at equal elevations*, wh
center of the transit, the line connecting these two points will be
top, and its bearing can be used for starting the undergrou
must be transferred, as before, by a single plumb-bob.

Measuring depth of shaft. (*a*) By direct measure
the timbers, made by two men, one on the roof of the c
attached to the hoisting rope about 100 ft above the cag
third man to give signals promptly. (*b*) By measuring
two marks a known distance apart, one near the hoistin
the sheave. Take a reading on a leveling rod held on
(freely suspended) at the bottom landing, or mark the p
point of the cage against the guide, for future reference.
a mark on the rope, with a piece of string or wire, opposi
the sheave. Hoist the cage until the mark on the rop
mark at the drum, and measure accurately between thes
on the rope has gone past the fixed mark, call the readi
minus. Set a new mark on the rope at the point near the
operation until the cage reaches the surface. Take anot
cage, or mark its designated point against the timbering
number of readings by the distance between fixed points,
of the readings, and transfer the rod readings on the cage
lieu of rod readings, to bench marks at top and bottom of
ing a plumb-bob wire. The operation is similar to that
at the marks can be set at exact distance apart, say 100

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the wire, due to its diminishing weight as it comes out of ed, because the measurements are made on the wire at the n it were hanging vertically.

9. Notes and Computations

of notes, and every engineer is entitled to his own prefer- tem is equally clear to him and to the engineering staff. le to use a printed form, with headings printed or written mns than necessary. The transit man can then tell at a tem of information pertaining to each set-up is recorded, e next station. If a certain measurement is not required heck-mark should be made in the appropriate space, to t overlooked. The following headings are suggested for traverses.

Azimuth Traverse

	Tape	Vert angle	H I	H S	Remarks

Deflection Traverse

ouble def	Tape	Vert angle	H I	H S	Remarks

Angle Traverse

econd read	Tape	Vert angle	H I	H S	Remarks

lection traverse the narrow column under "Deflection" ials *R* and *L*. After dividing the doubled deflection (or alue may be entered in the first column, on the line below. f any measurement are taken, they may be entered one same column; it does not pay to economize note-book entered one after another in regular order, following the that station. They are preferably lettered, to distinguish ed traverse stations. In the case of an angle traverse by devote a whole or a half page to each set-up. The record be entered thus:

Angle 7–8–9

	First reading			Fourth reading			Final value		
	°	′	″	°	′	″	°	′	″
...	—	—	—	—	—	—	—	—	—
...	—	—	—	—	—	—	—	—	—
	Average.....			.....			—	—	—

The data relating to courses 8-9 and 8-7 would then be ent page in their appropriate columns. Sketches and remarks right-hand pages, which should have coordinate ruling.

Loose-leaf systems. Among the advantages are: (a) Page since a particular page need not spend more than one day und is possible to file together all the notes relating to a given section though they may have been taken by different engineers a (c) A complete record of all notes is constantly on file in the working at a distance from the office, the notes can be sent putation and mapping.

The forms should be printed on bond paper cards, say 5 by fit the clasps of a leather or aluminum cover, which should fully to protect the edges of the sheets. Duplicate copies are able, in order that the engineer may have a complete set as One way to make these is to have white envelopes ruled and like the cards, into which the cards fit snugly. One card and paper are put into each envelope, and a number of envelopes cover for the day's work. The engineer then retains the emp returns the carbon copies to the office.

Computations. The first step is to convert azimuths, defl into bearings, as described in Sec 17, Art 11. Next, red tances to their horizontal and vertical components, multiplying sine respectively of their vertical angles. Gurden's traverse help in this operation. Next compute latitude differences and courses, or take them from traverse tables. If a closure ca pute and distribute the error in proportion to the length of From the balanced latitudes and departures compute the c stations. If no closure can be made the coordinates are ca unbalanced latitudes and departures. Lastly, compute elevatio from the vertical distances, the H I and the H S, as describe following form will be found convenient for assembling the d putations.

Course	Azimuth	Bearing	Inclined distance	Vertical angle	Horizontal distance	Latitudes		Departures		Corrected latitudes		Corrected departures		Coordinates		Vertical distance	Height of Instrument
						+N	−S	+E	−W	+N	−S	+E	−W	Lat	Long		

The whole aim and object of a mine survey is thus to determi and elevations of as many points as possible. If the whole

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d on the same set of coordinate axes, nearly every prob-
ring can be readily solved by the application of the princi-
or analytical or descriptive geometry. (See Sec 36 and
the bibliography.) To minimize the possibility of error
vell to take as the origin of coordinates a point well to
of the entire workings; negative coordinates are thus
ne reason elevations should be referred to a plane well
kings.
lems are the ones most frequently encountered in laying
s.

line connecting any two points. If x_a, y_a, z_a are re-
le, latitude, and elevation, of station A, and x_b, y_b, z_b are
for station B:

line A–B is that angle whose tangent $= (x_a - x_b) +$
ill indicate in which quadrant the bearing lies.

nce A–$B = \sqrt{(x_a - x_b)^2 + (y_a - y_b)^2}$.

vation between A and $B = z_a - z_b$.

the line A–B is that angle whose tangent =

$- z_b) + \sqrt{(x_a - x_b)^2 + (y_a - y_b)^2}$.

A–$B = \sqrt{(x_a - x_b)^2 + (y_a - y_b)^2 + (z_a - z_b)^2}$.

un a traverse by the shortest available route to Sta B;
t form part of a closed circuit, it can be made to close by
route, but using different stations. A serious error of
tigated, by repeating the work, if necessary, but a reason-
anced. This gives the coordinates and elevations of A
ring and slope of the connecting line A–B by formulas
e transit at either A or B; orient by sighting on a traverse
otion; set the horizontal vernier to read the computed
nd the vertical vernier to read the required slope, and
ich the mine boss can hang sighting cords.

a vertical shaft to connect with a winze or raise. From an
the winze or raise, measure bearings and distances to the corners,
thereof. Run a traverse to a station on the surface near the
ft, and compute its coordinates. Calculate bearings and dis-
to four points hav-
as those observed
nd 2 above). With
nrface station turn
outed bearings, and
g distances to the

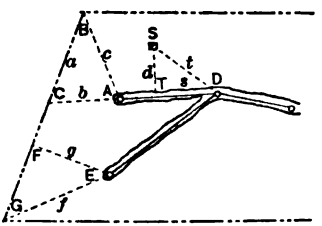

Fig 29

which a given line
line, for example,
The known data
of B–C, or a, and
of b), and the coor-
nd B; required, the
mpute bearing and
1 and 2. From the bearing of c compute angle B, by reference
also angle C in the same manner. Then $b = c \sin B \div \sin C$.

test line from a given point to meet a line of known bearing.
iously be perpendicular to the given line; hence its bearing can

be computed and set off at once by a transit on the given station. If connecting line is desired, the coordinates of some point on the given lin as G; whence compute bearing and distance E–G. In the right triang and the angle G are now known; whence $g = f \sin G$.

To determine the point on a given line from which to make the sh with a given point, as, for example, to drive a crosscut from the drift D of the shaft S. The shortest line will obviously be perpendicular to bearing is known, as are also the coordinates of D and S (the latter from Compute bearing and distance of D–S, or t, from formulas 1 and 2; angle S. Then $s = t \sin S$.

The above are given only as suggestive examples. In gener solution of a problem will become readily apparent by the sim plotting the known factors.

10. Mine Maps (See also Sec 19)

The map of a growing mine should be drawn on muslin-backe in a roll long enough to accommodate the most extended worki 1 in = 100 ft is as small as should be used for a map required t and a larger scale is better if the map can be brought within the

The first step is to rule the paper with fine, light-blue lines constructed squares of 4, 5, or 6 in, depending upon the scale to b ber these coordinate lines along the margins of the paper. P stations by coordinates only; inasmuch as the coordinates ar puted, no time is saved, and avoidable errors are introduced, by tractor for any of this work. Indicate each traverse point by a and write its number and elevation in black. It is not advisab coordinates, as the figures would be crowded, and the coordina available in the office records; it is advisable, however, to writ on the map to give a better three-dimensional idea when inspecti traverse points by fine red lines, stopping at the circles.

Plot the outlines of the workings by offsets from the traverse sections, or by angles and distances (reduced to horizontal), as t A protractor is allowable for this latter work. A paper prot diameter, with a semicircular opening cut out of its interior, is ne as the more expensive instruments. Many special forms of prot for this work are in the market. Those carrying a scale on the r excellent where much mapping on the same scale is to be done. described by J. J. Bristol (13) diminishes the labor of plotting, b reducing inclined distances to horizontal. Draw the outlines in b If the mine workings lie vertically under one another, on only thr colors may be used to distinguish them. If more than this num to make a separate map for each level, all oriented and co To get a combined view, tracings of the several levels can be Conventional designs to indicate the various underground feat in Sec 19, Art 1.

The following nomenclature was suggested by W. E. Sanders (1) for workings:

Horizontals: a = Main or subsidiary exterior horizontal passageway tunnels.
d = Main and subsidiary interior horizontal passageways ways, gangways, etc.
j = Junctions, stations, etc.
x = Main and subsidiary interior horizontal transverse crosscuts and headings.

Inclines: i = Main and subsidiary inclined shafts.

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d subsidiary vertical shafts.
rock-ways, mill-holes, etc.
ays, manways, etc.

talls, rooms, chambers, etc.
reasts, bottoms, or sides in ore.
all kinds.

ap should contain: (I) A title in simple lettering, and ver right-hand corner recommended) stating: (*a*) Name hereof (the words "map," "plan," etc, can safely be he surveyor and draughtsman; (*c*) date of survey and ated thus: 1 in = 100 ft, not 100 ft to the inch. (II) true (or assumed) meridian, and another indicating the his is of any interest. (III) A simple border.
g more elaborate maps in glass, wood, etc, are described

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SECTION 19

MINE GEOLOGIC MAI MODELS

BY

RENO H. SALES

GEOLOGIST TO THE ANACONDA COPPER MINING CO, B

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Art
1. Mine Geologic Maps.....
2. Mine Models.....
Bibliography at end of each article

1. Mine Geologic Maps

Geologic study and mapping are especially useful in comple of this nature, in conjunction with mine development and e chiefly by large mining companies, though similar study is p at small mines. The surface and underground notes and sk office maps, omitting less important notations from field boo overcrowding. Where structure is complicated, geologic pla by vertical sections or by models.

Underground observations are best made in new broken rock and ore. In wet ground, the surfaces soo black slime, derived from moisture, dust, and products o ceals mineralogic characteristics and adds to the difficu deductions. In moving or faulty ground, also, inspecti be made immediately after blasting and before timberin tion is frequently found in wet or heavy ground. Pron always desirable, to enable those in charge to direct wo the waste of misdirected exploration. Observations sho level workings, but should cover stopes and all other o faults, sill-floor workings in wide veins become heavy, closely lagged. Plans and cross-sections must then be m (Fig 4). Frequently, when engineer or geologist is calle ogy, much of the ground is stoped and no information obtainable.

Each ore-deposit presents its peculiar problems; in taking keep in mind the purpose of his work, and the point of view of plicated vein and fault systems structural features demand s cases, mineralogic associations are paramount. It is importa from non-essential geologic data.

t recorded in loose-leaf notebooks, with aluminum covers.
d hard underground usage, and sketches in bound books

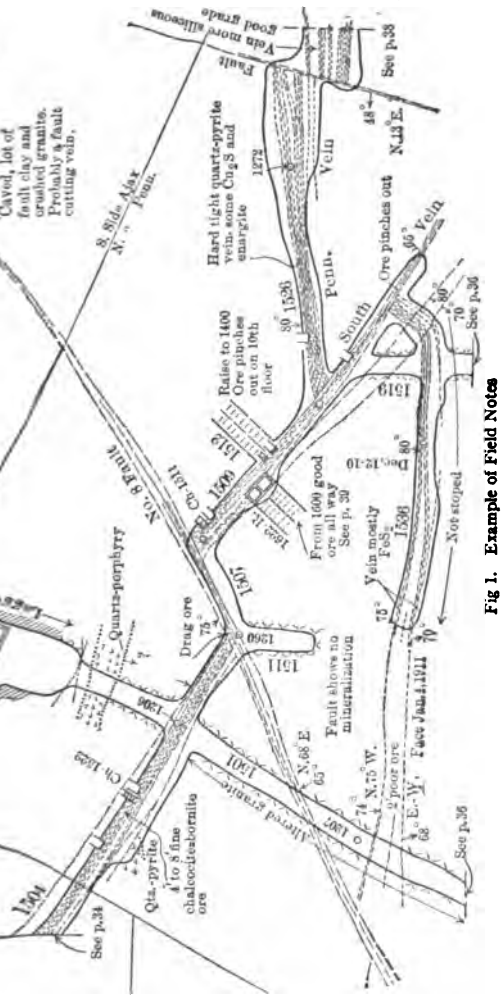

Fig 1. Example of Field Notes

loose-leaf system, only the sheets covering workings
n underground. Also, with bound books, growth of the
rowding of notes and sketches, or scattering of them in
eaf records can be arranged as desired, and made eas

accessible in office by a simple filing system. Scale and
chiefly on character of deposit; sometimes on scale of n
of notebook available. If minute details be necessary
tion of geologic data, use scale not less than 20 ft to 1 in.
use in wet or hot places, a 6 by 8-in page is desirable, ru
green indelible ink in 5-ft squares, with every fourth l
more favorable conditions, an 8 by 11-in page is prefe
ation of structural relations. Maps on white-print paper,
underground instead of notebook, but are unwieldy an
soiled. When possible, be
workings direct from min
orienting them on the pag

Fig 2. Conventional Signs, Rocks

Underground sketchi
are used to denote certain g
red for vein minerals, an
other dynamic effects.
matter by red, distinction
ore is left to side notes.
confusing, and for mixed n
of varieties is impossible.
narrow dikes, is best desig
signs (Fig 2). Chief ob
veins or faults, as contras
portant wall rocks. Vein
to represent correctly in
conditions are seldom inte
observers; hence careful,
essential. Observer must
others may have to int
sketches, wrong inferences
result from inaccuracy.
indicate clear understandi
muddled, mussy sketches
ideas.

Facts to be sketched
in general, everything hav
bearing on life of mine. Fi
cal examples of field notes
company employing a mine geologist. Conventional sig
along margins of workings. In crosscuts, drifts, stopes,
faults, slips, and dikes are plotted, with side notes at inter
dip, and mineralogic character, with their changes, if any;
located by tape measurement with relation to a mine su
fixed point (Fig 1 and 4). Pacing is excluded, except as rou
portant details. Keep sharp lookout for cross-faults, strik
vein. At vein intersections, or unions of veins, note cha
alogic character; also, influence of faults and direction
fault. Avoid too copious notes, a common fault of ine
note essentials briefly. Base generalities on careful w
facts.

Certain matters are of scientific interest, even if not of immedia
such as: [illegible] paragenesis of minerals, depth of oxi
and pre[illegible]n and country rock.

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pocket compass and tape. Brunton pocket transit is
yle transit, Attwood clinometer, and ordinary sighting com-
Metallic linen tape is accurate enough for most purposes.
e may be used. For notebook plotting, a 6-in transparent
cale, graduated
recommended;
and protractor

Geologic maps
nine in its early
the geologist is
nine many work-
lly or in part.
us. A common
ad air in dead
ngs lacking ven-
in pursuit of in-
lead to taking
bad air or deadly
suspected, get
before entering
ch that a candle
force matters by
lene lamps, both
air too poor in
. Other dangers
open stopes and
ose ladders, the
in presence of
In freshly broken
ed holes or loose
. Always watch
nd chutes, loose
ope floors, trolley
cars. Geologists
, as their work is
e is unwise.

For a small
ogic plans, with
-sections, or a
will suffice, ex-
plex structure.
operating more
may be dupli-
the superinten-
all-scale maps
ning mines, and
ailed maps of

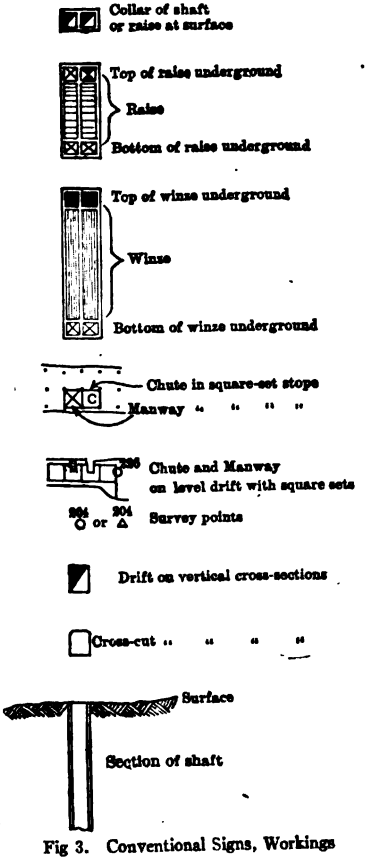
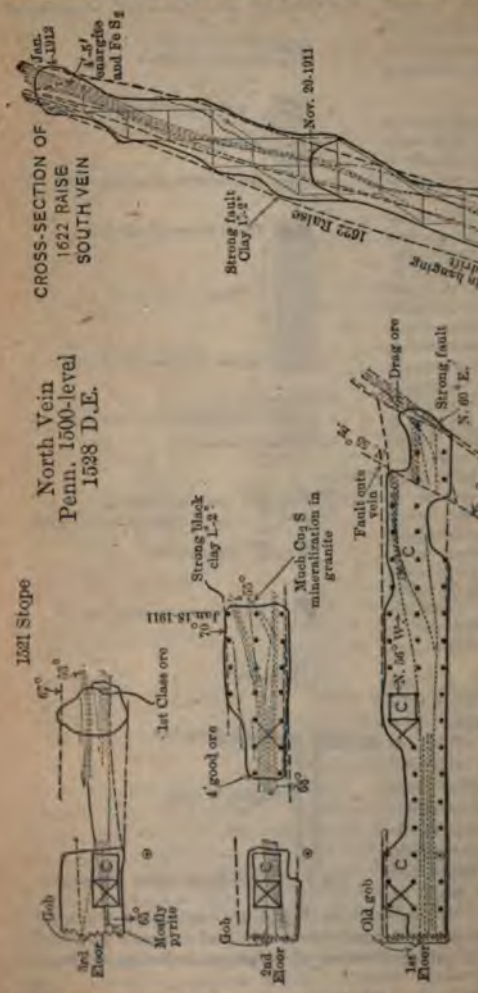


Fig 3. Conventional Signs, Workings

tion. Vertical-sections (Fig 5) are essential in conjunction
s are taken at regular intervals, and as nearly as possible
ike of principal veins; all openings, including stopes, are

ps. Tracing cloth is in general use, having many advan-
aterial, like drawing paper or unprepared blue-print cloth. A
es surface map and separate sheet for each main working
for intermediate levels or stopes are added when ne-

sary for correct interpretation of geologic structure. property lines, contour lines, and all surface openings are admissible only for reference, and are indicated by



fault outcrops must be accurately plotted. Approx craled outcrops, determined by projection from known dicated by dotted lines. Mine-level sheets (Fig 6) ar to register with surface sheet and with one another.

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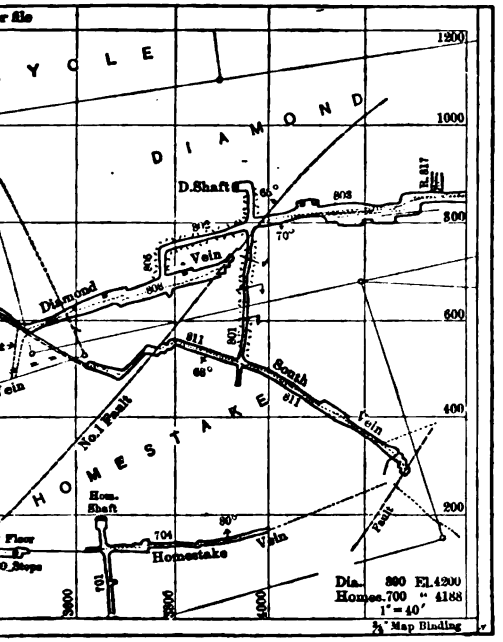
ll openings intersected by that section. Claim lines are eet, to show relative position of a working or of a certain y level.

number and elevation of level shown, is placed at lower right- eet. If mine numbers are used, place them along their respective zes, and stope chutes are shown along the drifts. Geologic n field books, closely to scale, with colored pencils or ink, corre- ed in field work. In map construction, however, wider latitude and different veins or faults may be drawn in different colors. vantageous when veins or faults are not of same geologic age. ed on higher or lower levels, but not intersected by any working



; Geological Sheet (on Tracing Cloth with Edge Bindings)

ideration, are projected in their probable positions. Reasonably geologic data on unopened levels assist the superintendent in ment.

ends on area covered, nature of deposit, and complexity of necessity of reducing or enlarging, it is convenient to use ne survey maps. Except for special uses, tracings should 6 by 42 in, though it is advantageous to include on one ne level as possible, and some allowance should be made for workings. For simple structure a scale of 100 ft to 1 in e areas, 200 ft to 1 in. But, on small-scale maps, accurate little side notation is admissible, and it is difficult to give

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due prominence to veins, stringers, faults, and slips. T
of veins should be evident at a glance. For complete
ground notes, a large scale is necessary. If a single she
whole mining territory, the area may be divided into bloc
sheets for each. In general, a scale of 40 ft to 1 in is good
Accurate field work, followed by careful transfer of not

Filing maps. Wear and tear on geologic maps are
come wrinkled, dirty, and opaque, and hence less useful.
corners and edges may be prevented by a ½-in bindi
print cloth; made by cutting cloth in 1-in strips, folding
stitching with a sewing or carpet-machine (Fig 6). Ma

Fig 7. Vertical Cross-section on Coordinate 3

filing cases or drawers, never folded, nor rolled if it can
non spring-arch file serves well for level maps attached
compo board, or thin pine board, cut 1 in larger than m

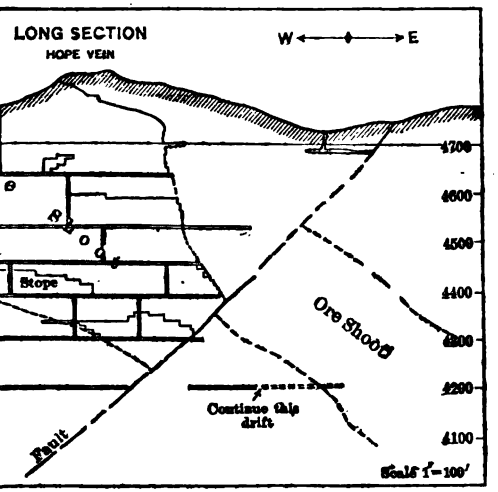
Thus mounted, tracing cloth soon tears out where punched
essary. This may be supplied by punching the holes in the m
a patch of blue-print cloth where hole is to be made. Maps ar
board, same as filing board, with slot to go over the Shannon
sections, and other less used maps, the Beck filing case is good

Vertical cross-sections. Sections are on planes pe
vein; or, in case of two or more veins or faults, sectio
structural relations will be best brought out. In bedde
may be taken perpendicular to strike of stratification
of deposit to inclosing strata. All workings cut by a sect

gs are specially important, to exhibit relation of advancing ures of level above.

ojection of unknown features to be fairly accurate, the prospect- the faulted vein segments are readily planned for minimum-cost. ecially useful for projecting veins, faults, dikes, or rock strata, stances on dip, to determine possible intersections or unions; s with reference to claim boundaries, under law containing the vision.

projections determine structural relations within plane the strike. All workings should be shown; whether in on plane approximating true dip, must be decided in each f longitudinal sections is to show extent of stopes. In cting veins, faults, or changes in country rock are the



g 8. Vein Map, Longitudinal Projection

es that can be represented. The pitch or general trend of shoots or irregular shapes is well shown. A cross-fault, orizontal displacement on plan, often causes considerable in plane of vein (Fig 8). Longitudinal sections bring out junctions or intersections of cross-veins, faults, or changes

geologic maps: (*a*) to assist in reducing mining costs; mine; (*c*) to preserve a geologic record of mine for future erstanding of the mine geology by the superintendent, *in* is of prime importance for avoiding useless underground geology is complicated, it is rarely possible for the mine ntricate structural relations of veins and faults, however with the mine workings. Best results are obtained by n and vertical-section geologic maps, possibly aided by

glass-section models. Veins and faults are projected
probable positions indicated by dotted lines.

Such advance interpretation of geologic structure assists su
in planning the larger development of mine, and in searchin
segments of known veins. Level workings intended as perma
air or haulage ways, shafts, pump stations, and raise connec
laid out to avoid faults and heavy ground. Maintenance c
directly affected by nature of ground encountered; hence the
maps, showing probable areas of heavy ground. To have t
the geologist must follow closely the advance of mine openin
cut off by a fault or dike, no money is wasted in search for the
man and geologist must therefore work in harmony, so tha
faulted vein, the first round of holes may be properly dire
wide, or composed of more than one fissure, its entire width s
search for faulted segment is begun. Owing to irregularitie
quent tendency of vein fissures to branch, or change dip or str
structural details often assists in keeping an exploratory drift
fissure; and the proper subsequent procedure for exploring
gests itself. If the deposit be in sedimentary rock, mapping
or genetic relations, if any, between ore and certain rock strat
rences to minor fissures, bedding, or cleavage planes may also
of recorded observations.

Comprehensive knowledge of the geology of a mine
its length of life. The period of profitable operation
covery of faulted segments of veins; or new orebodies
study of the mine geology and of the workings of conti
logic investigation often reveals the nature of factors
tion. When a mine shows signs of failure, through faul
with depth, those in charge must decide on future cou
tions call for a shut-down, or a further expenditure base
sound judgment.

Geologic maps are valuable in case of apex controver
greatest moment to have reliable, detailed records of g
mineralogic character of exhausted orebodies and par
become inaccessible through caving. Such maps and
by models, may constitute the chief court exhibits in m
mine is to be abandoned and allowed to fill with water,
to date at time of closing, are of scientific value for subs
district, as well as of vital importance for future engine
properties. Lastly, each mine geologic record is useful
of the science of ore-deposits.

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2. Mine Models

rking models, to supplement mine maps and elucidate com-
e. *B.* Court exhibits in law suits. *C.* Models for exhibition
o are unfamiliar with underground conditions or lack facility
A model may be intended chiefly to illustrate a mining method,
glected or broadly idealized; in this case the scale must gen-
a part of the mine can be shown. At the other extreme is the
eposits with reference to boundary lines, as for court use, in
e made paramount. An intermediate position is occupied by
nal models, which combine detailed geologic structure with
ine workings. According to type of construction, there are
king models; (*b*) vein models; (*c*) models of mine timbering;
models.

portant features of a model for mine use and to illustrate
e and rapidity of construction; facility of making additions to,
urability of construction; preferably not requiring services of
reasonable cost. For court use, designer must remember that
rmation of laymen, not engineers. Hence, design is simple, to
es, and on a scale large enough for court and jury to see model
s need not be built with regard to future use at mine, nor con-
ns. Beauty of design may be given more attention and there
to cost.

nodel, based on mine maps, is a small-scale reproduc-
substance, of shafts, drifts, stopes, etc, which are mod-
rted in true relative position by light iron or wooden
are best, being less bulky and not easily confused with
s extend to base of model they are usually of steel, to
supports. The principal workings (other than stopes)
d, like maple. Stopes are best made of pine, for readily
hapes. Level workings are cut in the rough with jig-
l or knife, and fastened together with brads, wooden

pins or glue. The trusses are supported by a smooth, wooden
are drawn the claim lines and coordinates of the mine system.
is given its true gradient, by previous determination of many
showing correct relations to adjacent shafts or mines.

The skeleton model, not intended for geologic study, may be painted
ably light gray. For portraying geologic features, also, the ore-deposit
painted in contrasting colors, like red, blue, and yellow. Trusses and o
painted black. Names of shafts and veins, and working numbers
waterproof ink, or by pasting on printed labels. Surface topography m
(*a*) iron wires, each bent to represent a contour and fastened at interv
above and below, and all held in position by a light metal frame, attach
trusses; (*b*) ordinary wire window screen, or a sheet of gelatin, shaped
Proper scale of model depends on dimensions of mine: 40 ft to 1 in is
conditions; 60 ft to 1 in is too small, and 10 ft to 1 in generally too la

Skeleton models are not satisfactory for illustrating geologic s
there are many continuous workings in the ore-deposit. They a
court use, if ore-deposit is well stoped out and not excessive
absence of connections between levels, vein continuity can be ind
strings, celluloid or gelatin sheets. Cross-faults, or slightly devel
position of which can be approximated, may be shown by glass or

Vein models are made to scale from cross-section and level ma
vein boundaries. The vein is built of solid wood, beginning wi
or the level sections are made separately and afterward combin
regular, the whole, or large portions of it, may be shaped from a
wood. Drifts, raises, and stopes are outlined with ink on the
raphy and claim lines are shown as for skeleton models. By
model with skeleton model, the position of vein referred to m
shown. Though well-designed vein models may exhibit effects
tions, faulting, and changes in strike and dip, they have little
To build them requires a skilled cabinet maker, and for solving
lems they are inferior to sheet-glass models. When much of vein
large portions must be idealized, giving definite thickness and fo
factors are wholly unknown.

Mine timber models are possible only where square-set tim
and accurate stope records are kept. Drawbacks: high first
quent cost of keeping model posted up to date, expert workman
and when finished additions are difficult. This type is chiefly
and court use, but if on large scale, the method of timbering is

Solid glass models are composed of a series of superposed pl
in which are cut openings corresponding to those in mine.
plans or sections are required at 8-ft intervals, throughout the
be modelled, each glass sheet corresponding to a plan or sectio
of each opening is marked on glass, sawed out as in scroll-sa
surfaces of openings painted any desired color. Plates are cem
by Canadian balsam. These models are costly, and can not
extended.

Solid wood models are of wooden sheets, usually ½ in to
sheet representing a vertical slice through mine. From the map
and geologic structure are drawn on the wood; sometimes open
correspond with mine workings. Lower edge of each sheet re
upper edge being shaped to represent topography of the slice. A
the claim lines, outcrops, buildings, etc, are drawn on upper surf
access to any section of model, the slices may be hung on hinges

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nly, as in RELIEF MAPS, wood, clay, plaster of paris, and may be used. Concrete has a limited application for large s of mines, to show methods of timbering; or for topographic e cross-section is de- y except for perma- eums and schools.

odels, consisting of al sections, are the effective for geologic with mine develop- ital-section model is the mine level work- so spaced in vertical how the workings in horizontal relations. and geologic maps pre-requisites. The d on the glass by a paint with gold size consistency to flow line pen, but not so or make sharp lines proof ink does not d washing and dust- he geology, use tube esponding to those of working outlines on geologic features on rectly by placing the cal map. Additions made. Dips of veins dded, but side notes ty lines are drawn on at all will show true Put names of claims only, together with e openings, and shaft

test dimensions should ount of weight and dan- is facilitated by using ine maps. Otherwise, is not complex, a scale ctory, permitting large small sheet. Ordinary hough picture glass is

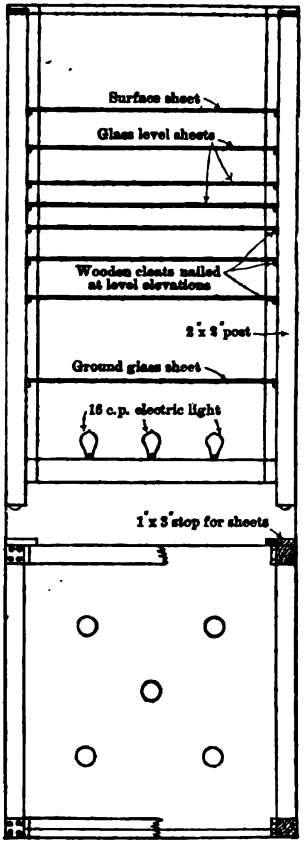
Surface sheet
Glass level sheets
Wooden cleats nailed at level elevations
2" x 2" post
Ground glass sheet
16 c.p. electric light
1" x 3" stop for sheets

Fig 9. Glass Model, Front Elevation and Plan

flaws and more uniform in thickness. Plate glass is heavy and
MENT CONSTRUCTION the sheets are spaced vertically in model
h mine level elevations. Fig 9 shows the older type of construc-
arried on cleats. In another form, the sheets are supported in
age of glass by possible spreading of sides of frame, metal runners
bottom edges of grooves. FOR TEMPORARY USE, instead of 4-post
t up with 2-in wooden blocks between them, of heights equal to
f the successive mine levels. When no longer needed, the sheets
and used again. A difficulty in using ordinary glass sheets for

deep mines is that, not being perfectly transparent, where ther cut off from those near bottom. A partial remedy is to set e of model, or instead of wooden base use a sheet of ground glass also confused where there are many levels.

Modified glass model for deep mines. The diffi are successfully met by a design used by the Anacon (Fig 10). Model similar to those o Sections are of 1 each holding a c sheets. The secti ently, or may be height; all are of except base sectio port and means of

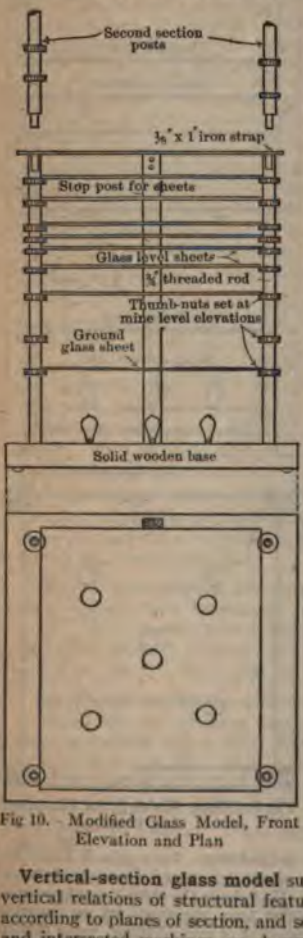

Fig 10. Modified Glass Model, Front Elevation and Plan

Mounted vertical base are four ¾-in 2-in outside diamete supporting the level rest directly on nuts by 30 in. large was The rods are held ri across their tops. ½-in square wooden graduated to differe level sheets are space tions by the nuts and into position from th upright stop at the horizontal straps con For high and heavy frame is added. Su as base section with ends of the vertical turned down and th in tops of rods of fi also made to the scal graduation to top of

Advantages of (*a*) adaptable to ma frame may be used only being necess sheets may be file needed; (*d*) mode stored; (*e*) with sli per sections can be vertical sheets, for model with supple

Vertical-section glass model supplements the horiz vertical relations of structural features of the deposit. according to planes of section, and set in grooves in woo and intersected workings are shown, as on the horizon more than a few hundred feet deep, a scale of 100 ft Size of glass should be sufficient for making additions, a This type is especially useful for study of deposits having

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tical extent, such as the disseminated copper deposits. f geology and workings may be shown by drawings on etween vertical sheets.

lass section-model are generally designed to show geo- tinuity from sheet to sheet. The following construction wood the raises, stopes, and other intermediate openings; lor as that given to vein in which they are made; attach "Caementium," which readily adheres to wood or glass. tion of the glass-sheet and skeleton types, is unsurpassed ntinuity between levels, where there are raises, but no N GELATIN SHEETS may also be used to represent vein and els; when slightly warmed, they are readily cut and ed shape. The gelatin sheets are placed in position be- following variations in strike and dip, and fastened to ent. Thus, width of each gelatin sheet equals distance een levels. Areas removed by raises, winzes, and stopes ainted on the gelatin. This construction is not entirely r more closely spaced veins, or in cases of complicated combines in a thoroughly practical manner the longi- the glass-sheet geological model. VERTICAL GLASS SHEETS ES BETWEEN HORIZONTAL SHEETS may also be used to show rkings between levels. Workings intersected by the plane the vertical sheets, with the geologic facts disclosed. As horizontal- and vertical-sheet models, this combination is hough the vertical strips furnish some support for large y restrict the view of observer. For small models, the e continuous, horizontal relations being shown on narrow but the tendency of narrow horizontal strips to sag is

uction of the supporting framework is desirable for all glass-section type, the sheets of which soon lose trans- st. Complete glass cases may be supplied for large per- loose wooden box cover of light construction, for models Cloth covers are not dust-proof.

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SECTION 20

MINE ORGANIZATION ACCOUNTS

BY

J. R. FINLAY

CONSULTING MINING ENGINEER

Art	MINE ORGANIZATION	Page
1.	Business Management	1268
2.	Technical Management	1269
	MINE ACCOUNTS	
3.	Principles	1273
4.	Cost-keeping	
5.	Mine Records	
6.	Major and Minor	
	Bibliography	

Note. — Numbers in parentheses in text refer to Bibliography

Introduction. A full discussion of this subject would be len relations between the objects of the business and the field of te practice. The aim here is to exhibit methods of mine organiz chiefly by means of concrete examples. The examples given are lated. Thus, the daily reports given under Cost-Keeping (Art 4 the discussion of Mine Records (Art 5), and both subjects are su fied by the extensive series of cost tables in Sec 21.

MINE ORGANIZATION

1. Business Management

Essentials of efficient operation are: correct unde ducing possibilities of the mining property; good techni ating skill; adoption of correct methods of finance. which the working organism rests is the stockholders, w the conduct of the business. In the case of the large corp actual control is usually exercised through leadership holders. Such a situation sometimes results in abuse because the body of stockholders feel that their votes ha the conduct of the business. Controlling groups some holders by paying unearned dividends. Conduct of th owner or by a small group of partners is usually mor necessarily so. In large corporations the management cally self-perpetuating committee of stockholders, calle tors" (see below). This board usually has power to buy

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s, to dispose of its funds, and to borrow money; it can
business, it is the mainspring of efficiency of the enter-
cess or failure.

may be divided into two groups: (*a*) those which
to working a given property, and when that is exhausted
ss; (*b*) those which contemplate perpetuating the busi-
end on a single mine, but by constantly setting aside part
om time to time obtaining new capital, or both, provide
opment of new mines. The second type of corporation
policy, and the Board of Directors is usually composed
branch of the mining business, to whom organization of
matter of vital personal concern (2).

Refining Co is an example of the second type of corporation.
ion of a group of three companies engaged in mining, smelting,
r, lead, zinc, gold, silver, and other metals. Combined capital,
presented by fixed investments and working capital in different
thousands of stockholders and probably 100 000 employes.
mate $10 000 000, and large sums are set aside annually to buy
old ones. Its general business has steadily grown, because it
d intelligently managed. Its Board of Directors comprises 27
ht be described as investors or capitalists primarily, though of
active administrators of the business. The remaining 18 direc-
ployes, who have been and still are experts in transportation,
nd executive work. The Executive Committee consists of 17
rs; the Finance Committee of 4 members of the executive com-
eral officers; President, 4 Vice-presidents, Treasurer, Secretary,
itor, Auditor, General Counsel and Assistant, and an Assistant
these officers are directors. The Vice-presidents are executive
artments. There are also 11 minor department heads and con-
m 4 are directors. The minor departments comprise sales of
y, heads of several important groups of mines and smelters,
gers. The entire group of about 40 business heads, financiers,
re the responsible organizers of the business. Subordinate to
mines or smelters.

Missouri, a much smaller corporation, is another example of the
out 1 100 stockholders (bond and note holders, number un-
eneral officers, 2 consulting engineers, 3 000 employes; total,
te investment: mining, milling, and power plants and acces-
ng plant, $2 000 000; transportation plant, $7 000 000; total,

2. Technical Management

f this part of mine organization are illustrated by the
h Lead Co (Art 1). Under the President of that com-
Managers; one for mines and mills (3 complete plants),
way, and 1 for the general service unit. In general, the
nagers, and the scope of their duties, vary in different
neral manager is put in control. Sometimes, as in the
of Lake Superior, this officer is called an agent. But,
ctive man, it is seldom necessary to interpose a separate
elf and the heads of naturally independent departments.
that one man shall be in direct control of a department,
d manager, general manager, superintendent, or general
aterial. Each such head has charge of all expenditures
accounts for them to the treasurer, but he does not
ount for the receipts from his department. These are
e general office, under direction of the president. The

department manager makes requisitions on the gene
to meet his expenses; hence his department really pr
he has no control, and for which he does not accoun
only the work done under him, and is called cost-kee

Organization data. Details of an elaborate organizatio
actual figures are for operations on a somewhat curtaile
force, there are 80 000 to 85 000 man-shifts per month, th
ployed each day would be only 10 to 12% greater than sho
curtailment results partly from reduction of employes, and
of working days. Normal production, 7 000 tons crude o
plants, Rivermines, Leadwood, and Bonne Terre, employ
office force of 51 men includes superintendents, master mec
ing agent, cost-keepers, time-keepers, stenographers, etc.
offices employ 10 surveyors and draftsmen, whose principal
mine maps. For sampling and laboratory work there are
department has a total force of nearly 70 men, comprisi
"division of information," engaged in obtaining and tabulat
for supervising the work. In the other departments of th
ployed in the division of information are proportionately e

In the mines of this company the orebodies are irregu
limestone, and are at an average depth of 400 ft. The oper
tion by vertical diamond drill holes from surface; (*b*) deve
orebody; (*c*) mining without timbering, by blasting in
(Sec 10), shoveling ore into cars, hauling to shafts by com
and skips (chiefly by electric hoists working in counterb
electric triplex plunger pumps, but also by steam pumps),
portation of half the ore to mills by railroad, the other h
mill from shafts; (*e*) concentration by jigs and tables, wit
(*f*) transportation of concentrates 35 miles to smelter; (*g*)
in blast furnaces, refining; (*h*) transportation of pig lead t

Variations in organization of operating staff are illustra
cerns like the Goldfield Consol Mines Co, and Nevada
Examples 2 and 24). These, with the St Joseph Lead
organizations; a low-grade, large-scale, concentrating lead
gold mine on a fissure vein; and a concentrating copper mi

An outstanding fact is that the majority of workers in a
laborers, but trained operatives of machinery. It is true t
ing is practiced, and ore is shoveled into the mine cars, whi
but these are almost the only important acts in mining and
the power.

Daily mine records. Contrary to a common id
are more important in securing results than the perm
a series of working faces, under constant attack, and
conditions to be met from day to day. A manage
costs and production statements as a basis for ma
might find that many or all parts of the mine had
he got his information. Since the chief problem is
profit all the time, any work which has ceased to pa
ately. To determine this, the manager must know
production and the cost of each part of the work (1

Daily estimates of profit. For these, formal st
The manager can usually calculate them readily, i
men employed and the tonnage produced. Then, tot
by number of shifts of labor employed, gives cost p
case, the total cost charged for mining, milling, fr
for one month was $54 302; number of man-shifts,
therefore $5.77. So long as the mine continues to w

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Lead Company and Subsidiaries — Organization Data

day for November, 1914. Month ending 7 A M, Dec 1, 1914

This month	Last month		This month	Last month
		TOTAL MINES		
23	23	Average men	1699	1856
397	467	Average tons ore mined	6270	6473
174	155	*Average tons per man	3.37	3.30
27	33	POUNDRY (25 day)	4	4
10	31	HERCULANEUM (30 day basis)		
54	51	Office	19	18
3	10	Roaster (19 day basis)	184	128
3	3	Smelter	173	177
13	15	Refinery	36	40
6	8	Power house	30	29
10	10	Mechanics	35	42
2	2	Construction	29	20
722	808	Yard	60	66
2718	3061	Laboratory and sampling	9	9
6.84	6.53	Bag house	5	4
		Total	580	533
13	12	Average tons pig lead produced	184	216
285	293	*Average tons per man	0.359	0.404
93	85	B. T. FARMING AND CATTLE CO (25 day basis)		
23	22			
59	55			
6	6	Office	6	6
6	6	Store	13	14
7	8	Farms	9	9
23	25	Repairs	5	6
3	3	Streets	2	2
		Water supply	2	2
6	5	Teamsters	10	12
6	8	Library	2	2
10	10	Electrical dept	1	1
9	10	Total	50	54
1	7	M. R. & B. T. RAILWAY (25 day basis)		
550	555			
1802	1732	Gen mgr and clerical force	9	9
6.32	5.92	Accounting dept	13	13
		Traffic department	3	3
15	15	Mechanical	91	93
259	275	Transportation	124	133
3	52	Maintenance of way	66	78
14	17	Miscellaneous	5	5
9	10	Total	311	334
17	16	NEW YORK OFFICE (6 day)	11	11
17	17	GRAND TOTAL MEN	2655	2792
7	8	Average tons ore mined	.....	.....
6	5	Average tons per man	.....	.....
9	11	REMARKS:		
17	18	Actual shifts worked	63 167	74 481
2	3	Actual tons mined	129 014	155 355
4	4	Actual tons per man	2.04	2.09
2	4			
25	22			
14	7			
7	9			
427	493			
1750	1680			
6.76	6.12			

Note. — Correct tonnages are stated in this report; estimated in weekly report.

* Averages figured from actual tons and shifts.

is practically constant, and may be counted upon calculations. With a labor tabulation showing 400 m the manager of this mine can instantly calculate hi $2 300. Then, if he can calculate that the ore will day, he at once figures his daily income at about $ This illustrates the principle underlying the daily rec

Calculation of daily yield, however, is only mine, the records may appear somewhat as follows:

Working place	Tons	Grab sample assay, ounces per ton
A	26	1.06
B	19	1.96
C	110	0.32
	165	0.61

From this, the ore apparently is worth about $12 per ton. grab samples, as a rule, run high, and that it is safe to figu in this case, $9 per ton. Furthermore, if the 110 tons only 0.32 oz ($6.40 per ton, of which 75% = $4.80), and ton, stope *C* is plainly losing money, and it should immedi

Daily Statements. Some simple process of calculatio mine, and for each important working place in it. Referrin that the men employed in each subdivision might be ent Such a statement can be made on a horizontal line of a for day in the month. By filling it in daily, the number of m detail; also the daily product from the working places for month. Information of this kind is needed also by supe their several departments. Though the interpretation of telligently only upon the complete periodical cost stateme manager, superintendents, and foremen are intimately conn given in Art 4). The formal monthly statements are regu men, who receive them with only a mild interest and use sions drawn from the daily reports.

Mine maps. Sec 19 gives details of mine maps geologist's standpoint. Geology of a mine is an in formation conveyed by maps; but maps also ex namely: location of workings; depths, dimensions, tion of boundaries and approach to areas of disput more effective handling of ore, timber, waste rock, maps is an important aid in attaining efficiency o maps, from which dated blue-prints are made at inte furnishes the history of the development of a mine a casting its future. Lacking a comprehensive syste tion may be lost. In working a mine, details ac difficult to correlate unless they are recorded at once scheme. In general, the maps should be made in fol

1. Property maps, showing ownerships, boundaries
2. Surface maps, showing topography, roads, buildi
3. Surface geologic maps.

Note. — A single map may cover functions 1, 2, and 3.

4. General maps of workings, showing: (a) areas e (c) general geologic facts.

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wing: (*a*) underground geology; (*b*) occurrences of ore; ords; (*d*) assays; (*e*) depths and elevations; (*f*) other facts.

stopes, developments, workings (drifts, cross-cuts, raises d ore chutes; also detail assay maps for individual work-

n be made conveniently uniform in size, by adopting different amount of detail required; and all should be systematized for to another (17, 18, 19). Their utility is determined by the ease e desired facts, and display the critical mining conditions.

MINE ACCOUNTS

3. Principles

dly speaking, are adopted for keeping accounts of mining xpenditures are charged to production at the time they , certain expenditures are carried as "Capital Accounts," epreciation (1, 6, 7, 12, 13, 14, 21, 22, 23). Following principles of each method.

atement of Mohawk Mining Co, Mich, for 1913

lb copper @ 15.36¢		$887 618.50
, as per statement	$601 890.77	
Y and Boston expense	67 263.17	669 153.94
		$ 218 464.56
	66 972.17	
	27 652.87	94 625.04
all expenditures)		$ 123 839.52
12 (balance of quick assets)		897 316.40
		$1 021 155.92
13		500 000.00
13 (balance of quick assets)		$ 521 155.92

ents of Chino Copper Co, Hurley, New Mex, for 1913

ncome account. Year ended Dec 31, 1913

1 661 lb @ 15.0884¢		$7 621 419.35
	$1 635 915.17	
d freight	2 135 453.68	
	76 947.43	
	582 810.00	4 431 126.28
it		$3 190 293.07
vidends, rentals, royalties, etc		137 533.40
		$3 327 826.47
dvances	$69 862.87	
me tax	23 930.09	93 792.96
ar		$3 234 033.51
ar		1 919 070 00
(not a balance of quick assets)		$1 314 963.51
f plant and equipment		212 146.56
its		$ 102 816.95
vided profits, Dec 31, 1912		2 461 350.14
tments:		
urplus, now transferred	$99 000.00	
ar	80 000.00	179 000.00
t a balance of quick assets)		$3 743 167.09

Statement of Assets and Liabilities, Dec 3

Assets

Original purchase price of property	$1 802
Mining property purchased	113
Expenditures for construction and equipment	4 694
Development of properties	885
Investments	190
Deferred stripping expense	1 315
Materials and supplies on hand	537
Accounts receivable	137
Prepaid insurance and other expense	4
Value of copper on hand and in transit, less advances received	2 378
Ore in mill bins	5
Ore stored at mine	103
Cash	124
	$12 293

Liabilities

Capital stock outstanding	$4 302
First mortgage bonds	236
Notes payable	
Accounts payable	420
Freight, treatment, and refining charges not yet due.	571
Reserve for depreciation	212
Surplus from securities	2 806
Undivided profits	3 743
	$12 293

Following figures show what the statement of the C would be if reported in same form as the Mohawk stat

(*a*) Net profit shown in income statement	
Less expenditure for:	
(*b*) Property purchased	
(*c*) Construction and equipment	
(*d*) Development	
(*e*) Deferred stripping expense	
Total expenditures not charged in income statement	
Receipts from operations above all expenditures	
(*f*, *g*, *h*, *i*) Receipts from stocks and investments	$2
(*j*, *k*) Less bonds redeemed, and investments	2
Net receipts from stocks, bonds, and other investments	
Total receipts above all expenditures	
Dividends paid	
Decrease in quick assets	
Balance of quick assets, Dec 31, 1912	
Balance of quick assets, Dec 31, 1913	

4. Cost-keeping

General. Table 2 shows that detailed cost-keeping of the financial records of a mining company, but is a s kept separately by each department, the total cost re agreeing with total cost shown on operating books. nishes a financial record for the management, while co for the operating heads, to enable them to regulate exp

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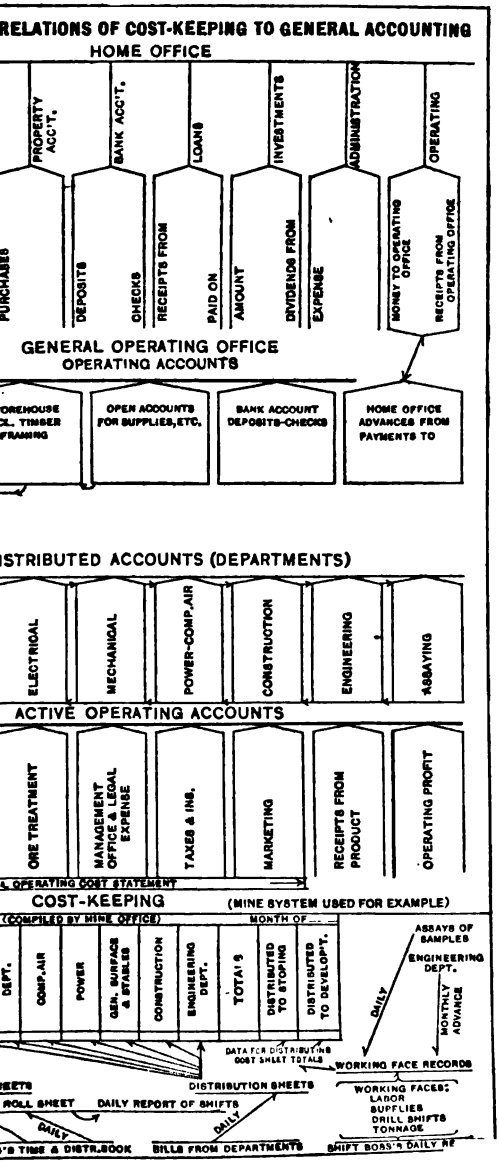
Table 2


RELATIONS OF COST-KEEPING TO GENERAL ACCOUNTING
HOME OFFICE
PROPERTY ACC'T.
BANK ACC'T.
LOANS
INVESTMENTS
ADMINISTRATION
OPERATING
PURCHASES
DEPOSITS
CHECKS
RECEIPTS FROM
PAID ON
AMOUNT
DIVIDENDS FROM
EXPENSE
MONEY TO OPERATING OFFICE
RECEIPTS FROM OPERATING OFFICE
GENERAL OPERATING OFFICE
OPERATING ACCOUNTS
OPEN ACCOUNTS FOR SUPPLIES, ETC.
BANK ACCOUNT DEPOSITS-CHECKS
HOME OFFICE ADVANCES FROM PAYMENTS TO
ISTRIBUTED ACCOUNTS (DEPARTMENTS)
ELECTRICAL
MECHANICAL
POWER-COMP.AIR
CONSTRUCTION
ENGINEERING
ASSAYING
ACTIVE OPERATING ACCOUNTS
ORE TREATMENT
MANAGEMENT, OFFICE & LEGAL EXPENSE
TAXES & INS.
MARKETING
RECEIPTS FROM PRODUCT
OPERATING PROFIT
OPERATING COST STATEMENT
COST-KEEPING
(MINE SYSTEM USED FOR EXAMPLE)
(COMPILED BY MINE OFFICE)
MONTH OF
DEPT.
COMP.AIR
POWER
GEN. SURFACE & STABLES
CONSTRUCTION
ENGINEERING DEPT.
TOTALS
DISTRIBUTED TO STOPING
DISTRIBUTED TO DEVELOPT.
ASSAYS OF SAMPLES
ENGINEERING DEPT.
DAILY
MONTHLY ADVANCE
DATA FOR DISTRIBUTING COST SHEET TOTALS
WORKING FACE RECORDS
DISTRIBUTION SHEETS
WORKING FACES: LABOR SUPPLIES DRILL SHIFTS TONNAGE
ROLL SHEET
DAILY REPORT OF SHIFTS
DAILY
DAILY
'S TIME & DISTR. BOOK
BILLS FROM DEPARTMENTS
SHIFT BOSS'S DAILY RE

Cost-sheet. Accounts kept by the mining department f
vary according to the mine (14, 21, 23). In any case the fina
be brief and deal with totals carried forward from detailed dai
accounts can not be directly distributed to working faces ea
on the mine cost-sheet it is desirable to deal only with gene
ferring to Table 2, under the head of cost-keeping, the ac
follows: For DEVELOPMENT direct: miners, muckers, trammer
For STOPING direct: miners, shovelers in stopes, trammers
expense for filling. For GENERAL ACCOUNTS: hoisting, pumpi
explosives, sampling and assaying, surveying, watchmen, mine
dent, foremen, shift bosses, blacksmithing, repairs and rene
repairs and renewals to machinery, mine and shaft repairs,
pense, new construction, new machinery, tools, miscellaneo
heating. To these accounts the charges are made in the v
cost-sheet. The total of the mine accounts is apportioned
counts, development and stoping; in some cases a third acc
or exploration, is added. It is not necessary to keep a set
system. Best method is a system of daily reports, distrib
loose-leaf record sheets (23). To facilitate filing, all reports
loose-leaf form.

Besides the mine accounts, the following division of costs n
WORKING FACE RECORDS (Table 4), after cost-sheet totals have
to development and stoping. For EXPLORATION: trenching
diamond drilling, churn drilling. For DEVELOPMENT: shafts,
raises, winzes, shaft stations, and pockets. For STOPING, th
divided according to the different methods of stoping empl
the system farther, these accounts may be distributed to each
totals agreeing with cost-sheet total (10).

Table 3. Shift Boss Report

MINE Shaft No 4 DATE Dec 1, 1914

Working place		Mach shifts		Timber								
Class	No	H	P	10×10	8×10	2×12	4×6	8×8	6×8	4×4	3×6	3×
Stope	401	2		4		30						

Explosives		Labor								
		Miners		Shovelers		Trammers		Timbering		Pi
Powder	Caps	4.00	3.75	4.00	3.75	4.00	3.75	4.00	3.75	4.0
20	4	2			1		2	1	1	

Remarks Please note any new working numbers

New Drift 901, started
R

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Table 4. Working Face Record

CLASS Stope MONTH Dec, 1914

Shift boss report

Timber

10 8X10	2X12	4X6	8X8	6X8	4X4	3X6			Round
	30								

Shift boss report, cont'd

Shifts labor								Hoisting	
Shovelers		Trammers		Timbering		Filling			
4.00	3.75	4.00	3.75	4.00	3.75	4.00	3.75	Cars ore	Cars waste
	1		2	1	1		2	35 40	5

Estimated cost

Explosives	Total labor	Hoisting	Gen charges	Total

desired during month, or at end of month

Production

	Total ounces	Tone ore to date	Avg assay to date	Face samples
	31.5 45.0	67.5	113	

Advance by Eng Dept (Monthly)
Working No, 401 stope

Table 4 are continuous horizontally on one large sheet; the
ys of month from 1 to 31.

5. Mine Records

Working-face records (Table 4) are compiled fr
(Table 3), which gives for each working face and
miners, muckers, shovelers, trammers, timbermen, a
piston drill-shifts, hammer drill-shifts; also dyna
(fuse may be averaged per cap used), amount and
and tonnage of ore and waste hoisted. In addition t
reported on shift-boss report, any other supplies s
chargeable directly to a particular working face ma
Assays of ore and face samples are also entered, as
The columns of estimated cost are worked up when
general account averages (11, 12, 23).

The above data give: direct charge for labor (wh
taken from payroll distribution); direct charge for su
assay records, and advance made. Hoisting is appo
ported, and total machine drill cost on basis of machine
charges that may be distributed directly. The rema
cost-sheet may be made in proportion to the totals

Labor records. When a man is hired he shoul
slip, stating character of his work. This slip furnish
authority for putting the name on the payroll. A
should be kept, giving name, signature, local addr
for each employe.

Table 5 is a shift-bosses' loose-leaf TIME AND DIS
work satisfactorily at a property where it had been
duction, to get the shift-bosses' daily reports of wo
time book. By first making the distribution in tim
readily filled out. A daily distribution of payroll
keeper from the shift-boss time books. Entries are
to the monthly PAYROLL SHEET (Table 6), which s
wages paid. To check the shift-boss time books, t
to the number in first column of payroll, may be g
in a box when coming off shift. Employes may be i
by asking their number and by reference to the identi
in the last column of the payroll is for the pay ch
receipt.

Supply records. Distribution of supplies is a co
if all supplies are kept in a central storehouse, and tre
Requisitions should be drawn only for quantities
sumption. Requisitions should show the distributio
storekeeper makes note of this distribution on the
mine office. These bills are entered on a SUPPLY DI
accounts as are kept for the cost-sheet, the totals
checked with a monthly storehouse statement. Sl
between supplies purchased and supplies shown o
owing to small balances on hand, but these differe
selves from time to time (8).

Department charges, for services and power rend
uted as for bills for supplies. Detailed costs of p
sheets, and department distributions should be kep
cerned. At properties where scale of operations doe
into departments, it is desirable at least to keep thes

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×	—		×	—		—	×	—	—	×	—	—	0	—	—	×	⅜	*Discharged*			
Class work *1*			Class work *5*			Class work *6*			Class work *1*			Class work —			Class work *5*			Class work —		Class work —		

Class of work:

No 1. Miners, or Drill Runners
No 2. Muckers and Trammers. Development
No 3. Shoveling in Stopes
No 4. Tramming to Chute or Shaft
No 5. Timbering
No 6. Filling Stopes
No 7. Pumping
No 8. Pipemen
No 9. Mine and Shaft Repairs

☞ Use these numbers to denote class of work

Shift boss	From	To
Richard Doe	*1*	*3*
Frank Doe	*3*	

OCCUPATION *Miner*
NAME *John Doe*
RATE *$4.00*
MONTH *Dec* SHAFT *4*
PAY ROLL No *106*

NOTE.—IN MARKING TIME, MARK — Full Shift, ×; Off, *0*; Hours, *1–8*, *2–8*, etc.

(**Note.**— The upper part of this form is carried out to provide for month of 31 days.)

Table 6. Pay Roll

Number 4 Shaft — Month of *Dec.*

No	Name	Occupation	Days of month, 1 to 31						To shi
			1	2	3	4	5	etc	
106	*John Doe*	*Miner*	1	1	1	1	1.1		5
	Total.....								
	Daily Total of Shifts								

No	Name	Deductions		Amount received	Received in full of t opposit spectiv
		Store	Hospital		
106	*John Doe*	10.00	1.00	9.50	*John Do*
	Total.....				
	Daily Total of Shifts				

6. Major and Minor Operating Units

This distinction is exemplified in the organization of the St Joseph The major units of this property are: 3 complete mining and millin smelter, and 1 general service organization. (The last is the Bonne Cattle Co, which controls the surface of mineral lands, houses for em supply, etc.) These 6 units are not necessarily dependent upon on road could haul ores for other mines, and any mining plant could shi smelter.

Major units can be divided into smaller, or MINOR UNITS, independent. Thus, the mining plants consist of the mines, power plants. Carrying subdivision still further, the mines c of shafts (with their connecting workings), which may be qui one another, but are dependent upon the power plant and a and each may have a choice of several mills for treatment of

Economy demands that each major unit be run at full capacity. more ore than is sent to it, part of its equipment and first cost is versely, if more shafts are operating than are required to supply th viously being worked at reduced capacity, and one or more of them s In the case of the St Joseph Lead Co, though 13 shafts out of 24 h within the past few years, the total output has been greatly incr wholly dependent on one another are illustrated by the items of Hoisting engine, pumps, haulage plant, shaft timbering, etc. must b in repair, whether the output is large or small. Therefore none of th be developed out of proportion to the others.

Efficiency engineering consists in a study of units, in ord mum output from each. To this end it is necessary to asc are effective and which are only relative.

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machine drill is to mine ore cheaply; the units comprised
st of the machine and repair parts, compressed air, explo-
, for breaking say 30 tons, is about $8, comprising: wages,
ir, $1.50; explosives, $3. Economy lies in getting a large
t be uneconomical to emphasize any of the minor units
ence of the others. Thus, by reducing wages, inefficient
lts; excessive economy in repair parts might cause costly
of compressed air would result in feeble drilling; reducing
ecessitate an excessive footage of drilling. It would be
the cost of operating a drill from $8 to $7 per shift at the
age broken from 30 to 20. Conversely, it would be good
hift if the output could be increased in greater proportion.
chine drilling is the cost per ton of breaking ore, and not
line of reasoning leads to effective management and cost-

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SECTION 21

COST OF MINING

BY

J. R. FINLAY

CONSULTING MINING ENGINEER

Example

1. A California Gold Mine (Tables 1–23)
2. Goldfield Consolidated Mines Co (Tables 24–33)
3. Alaska-Treadwell Mining Co (Tables 34–38)
4. Camp Bird, Limited (Table 39)
5. Portland Gold Mining Co (Table 40)
6. Liberty Bell Gold Mining Co (Tables 41–45)
7. El Oro Mining and Railway Co (Tables 46, 47)
8. Oriental Consolidated Mining Co (Tables 48–54)
9. Sons of Gwalia (Table 55)
10. Transvaal Gold Mining Costs (Tables 56–58)
11. Tonopah-Belmont Development Co (Tables 59–63)
12. Kerr Lake Mining Co (Table 64)
13. Daly-Judge Mining Co (Tables 65, 66)
14. Daly-West Mining Co (Tables 67, 68)
15. Bunker Hill and Sullivan M & C Co (Tables 69–79)
16. Lead Mining Costs, Southeastern Missouri (Table 80)
17. Zinc Mining Costs, Joplin (Table 81)
18. Tin Mining Costs, Tasmania (Table 82)
19. Churn Drilling Costs (Table 83)
20. Copper Mining Costs (Tables 84-87)
21. Copper Range Consolidated Co (Tables 88, 89)
22. Wolverine Copper Mining Co (Table 90)
23. Miami Copper Co (Tables 91–104)
24. Nevada Consolidated Copper Co (Tables 105–111)
25. Iron Mining Costs, Mich (Table 112)
26. St. Lawrence Pyrites Co (Tables 113, 114)
27. Coal Mining Costs (Tables 115–118)

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Introduction

been prepared by Heath Steele, along the lines adopted in
Finlay, published 1909. They present typical examples
s, showing actual costs as well as illustrating systems of
acticable to discuss costs of mining intelligently unless their
tood. Three distinct conceptions have been adopted: (*a*)
including dividends, sums loaned out, nor mere book en-
l money spent on property, on construction of plant and
actions; (*b*) Amount spent in operating mines and mills,
n plant, financial transactions, etc. These costs take the
d for a long period. This average sum (termed "depreciа-
rating expenditure, to give total cost. This cost will never,
h expenditure for any one year, but should approximate
umber of years; (*c*) Amount spent on operation, omitting
will not agree with actual expenditure, invariably falling
are given, of periods preceding the Great War.
business as a whole is first exhibited, followed by successive
s are as minute as practicable. The first table for a given
departments. Thus, Example 2, Table 24, shows operating
ated Mines Co: (*a*) Total expenditures are checked up by
s a gain, or minus a loss, in quick assets for each year; (*b*)
r 19 different headings, the sum of which equals total cost.
fall under the direct head of mining and milling operations.
own as a lump sum, occupying one line, accompanied by a
n Table 26 the mining operations are shown for 3 years,
items of labor, supplies, power, and general charges; (*d*)
Table 28, which shows costs divided into 25 or 30 items
average of the four.
much less in detail than in total. To do exactly the same
lves no differences in cost other than those due to the pro-
But the total cost per ton shows great differences, because
s of five steps, while another may require fifty. In general
obtained on rather a large scale by successful organizations.
iat these costs can be duplicated by small-scale operators,
ict.
ted are representative of different kinds of mining, the plan
estion of availability of information in different fields. In-
blic in non-competitive business, such as gold mining; but,
e and more vital, as in mining low-priced and abundant
re, operators are more secretive. Hence, the number and
not in proportion to the importance of the fields covered.
es from public reports, and all is authentic. Credit is given
e kindly supplied information, besides that made public in
nt cases contributors have desired to remain anonymous.
more detailed information as to coal and iron mining costs
the broad facts relating to Michigan iron mines are set forth,
e districts. Favored properties in each district do cheaper
ested in iron mining will need to consult more figures than
"Iron Mining in Minnesota," by Charles E. van Barneveld,
l of Mines; "Handbook of Steam-Shovel Work," by Bucy-
Vis; for general discussion of the iron mining problems, see
The meager costs relating to coal mining are disappointing.
esented.
way to discuss mining costs would be to adopt a unit basis.
a broad manner. Having defined and tabulated the steps in
rgical processes, the labor, supplies, and power required for
ed, the total cost being the sum of those of a given amount
ace.

EXAMPLES OF COS

Example 1. A California Gold

This mine is served by a vertical shaft 1 600 ft deep, inter dip, then turning and following vein on the incline. Th (1913) and practically no stoping is going on below 4 000- is 23°; average stoping width of milling material, 4.25 ft; av 2 ft; per cent of profitable ground to total ground opened sent to mill, $10 per ton. In upper levels the vein walls a the 4 000-ft level, of granodiorite. The main working levels ever ore is encountered raises are put through to level abo shoots on the strike is 400 ft; average length of vein prospe the raise is put up in center of ground to be mined, and a slices; intermediate tracks are laid from the raise every 30 For stoping air-feed drills of one standard make are used; for ing where necessary in the stopes, Water Leyners, No. 8. T 20 Leyner, and 4 Jack-Hammer drills on hand. Maintenan expense, and, to prevent loss of time, two drills are provide the time the following data were taken 60% of the ore was 3 700-ft levels. This ore was lowered to main skip-pocket 4 000 ft the skip turns to the incline and is loaded direct fr

Costs and Data for June, 1913. Total shifts worked, mining and 2 522 on surface. Total development work 41 drifts and 199 ft of raises. Drill shifts: stoping, 1 300; dev shifts. Ore: from stopes, 8 457 tons; development, 1 030 to

Table 1. Summary of Costs

Development } Mining }	Table 2
Milling	Tables 16 and 17
Concentrating	Tables 18 and 19
Cyaniding	Tables 20 and 21
General expenses	
Head office, approximate	
Bullion charges, approximate	
Taxes, approximate	
Accidents and pensions, approximate	
Improvements, approximate	
Total	
Average cost for first six months 1913	

Table 2. General Items, Mining and D

Items	See Table	Development per foot	Development per t min
Breaking rock	3	$ 3.95	$0.1
Timbering	4	0.20	0.0
Shoveling	5	.....	.....
Tools	6	0.68	0.0
Machine drills	7	1.43	0.0
Blacksmiths and mechanics	8	0.35	0.0
Repairs	9	0.08	.....
Lowering ore in main raise	10	.....	.....
Tramming	11	2.36	0.1
Hoisting	12	0.57	0.0
Pumping	13	0.67	0.0
Office, foremen, bosses	14	0.73	0.0
Miscellaneous	15	0.36	0.0
Total	.....	$11.38	$0.5

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A California Gold Mine (*Continued*)

Table 3. Breaking Rock

Stopes					
			Supplies		
	$3.25	$ 692.25	Powder, 276 boxes		$1 602
	3.00	3 025.50	Fuse, 58 600 ft		229
	3.25	75.00	Caps, 9 800 pc		116
	3.25	136.00	Candles, 31 boxes		117
	3.00	172.50	Other expense		6
	2.50	20.00			
	2.25	6.75			
		$4 128.00			$2 070
		49¢			24¢

, 73¢.
, 65¢.
for six months, 6.67 tons.
n per ton of ore stoped, 1.69 lb.

velopment (drifts and raises)

		Supplies	
25 } 00 }	$2.21 per ft	Powder, caps, etc, per ft	$1.48
25 } 00 }	1.72 "	"	1.74
25 } 00 }	1.87 "	"	2.41
00	1.72 "	"	1.93
25 } 00 }	2.34 "	"	2.31
	$158		
	$1.97		$1.98

per ft of development, $3.95.
18¢.
per ft of raise, 16 lb; per ft of drift, 12 lb.

n stoping and development per ton mined, 83¢.

Table 4. Timbering

			Supplies	
@	$3.25	$198.25	Lumber, 2 700 ft	$ 62
@	3.00	94.50	Poles, 7 500 ft	399
@	3.50	38.50	Candles, 13 boxes	46
@	3.25	497.25	(Note 1, following Table 23)	
@	3.00	54.00		
@	3.00	198.00		
@	2.50	182.50		
@	2.25	230.62		
@	2.50	43.75		
@	2.25	14.63		
23)				
		$1 552.00		$507

6.5¢.
filling per ton stoped, 23¢.
of development work, 20¢.
filling per ton of ore mined, 22¢.

Example 1. A California Gold Mine

Table 5. Shoveling

Labor					
80.5 shifts	@	$3.00	$ 241.50	Candles,
533.0 "	@	2.50	1 332.50	Shovels,
684.0 "	@	2.25	1 539.00	Foundry
Totals			$3 113.00	
Cost per ton stoped			37¢	

Total cost shoveling per ton stoped, 38¢; per ton mined, 3
Net tons per shoveler per shift after sorting out 15% wa

Table 6. Tools

Labor					
Tool boys,	52 shifts	@	$2.50	$130.00	Candles,
" "	102 "	@	2.25	229.50	Fuel oil,
" sharpeners,	104 "	@	3.50	364.00	Drill stee
" "	108 "	@	2.75	297.00	
Totals				$1 020.50	
Cost per ton mined				11¢	

Total cost per ton stoped, 13¢; per ton mined, 15¢; per ft
Tool sharpener and helper average from 275 to 325 pcs cro
of Leyner steel per 8-hr shift.
Average consumption of cross steel per ton of ore mined, 0.28
(Note 3)

Table 7. Machine Drills

Labor					S
Pipe man,	28 shifts	@	$3.25	$ 91	Pipe
"	2 "	@	3.00	6	Drill
Repairs,	125 "			441	Mild
					Bolts
	Shifts			Mach Shifts	Lubr
Air-feed stopers	1 087	Distribution:			Tool
Water Leyner	315	Stoping		1 221	Cand
Piston	33	Intermediate track		79	Powe
Jack hammer	41	Development		176	
Total machine shifts	1 476	Total		$1 476	
Totals				$538	

Cost per ton mined, 31¢, made up of: labor, 6¢; supplies,
mined.
Cost per ton ore stoped, 28¢; per ft of development, $1.43.
(Note 4)

Table 8. Blacksmiths and Mechanics

Labor					
Engineers and mechanics,	145 shifts			$423	Tool
Blacksmiths,	51 "			173	Pipe
Carpenters,	48 "			145	Elect
Electricians,	17 "			42	Steel
					Nails
					Bolts
					Black
					Char
Totals				$783	

Total cost per ton mined, 10¢, made up of: supplies, 2¢; lab
Total cost per ton stoped, 9¢; per ft of development, 35¢.

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A California Gold Mine (*Continued*)

Table 9. Repairing

Labor				Supplies	
2 shifts	@ $3 50		$42 00	Lumber	$55
7 "	@ 3 25		55 25	Poles, etc.	21
3 "	@ 3 25		9 75	Candles, 1 box	4
2 "	@ 3 25		6 50		
6 "	@ 3 25		19 50		
		$133 00			$80

er ton mined, 2¢; per ft of development, 8¢.

Lowering Ore in Main Raise (Note 6)

Labor				Supplies	
90 shifts	@ $3 00		$270 00	Lubricants	$100
145 "	@ 2 50		362 50		
77 "	@ 3 00		231 00		
75 "	@ 2 25		168 75		
		$1 032 25			$100

ise, 5 498; cost per ton handled, 20¢.
12¢; per ton stoped, 13¢

Table 11. Tramming

Stopes

Labor				Supplies	
ifts	@ $2 50		$132 50	Lumber	$ 8
"	@ 2 50		135 00	Candles, 6 boxes	24
"	@ 2 50		125 00	Ties, rails and fittings	21
"	@ 2 50		257 50	Lubricants	10
"	@ 2 50		30 00	Hay and grain	70
work, 20 shifts			56 50	Foundry work	18
		$736 50			$154

sl, 11¢, made up of: labor, 9¢; supplies, 2¢.
pes, 1 200 ft, with two men per 6-car train.
pes, 100 ft.

Development

Labor			Supplies	
hifts	Rate	Amount		
17	$2 50	$ 42 50	Lumber	$ 15
2½	2 25	5 63	Candles, 7 boxes	27
41	2 50	102 50	Ties, rails, and fittings	104
37	2 50	92 50	Lubricants	6
112½	2 50	281 25	Foundry work	4
105½	2 50	263 75		
1	2 25	2 25		
13	.....	39 25		
		$829 63		$156

t: labor, $1 98; supplies, 38¢; total, $2 36 per ft.
a stopes and development per ton mined, 20¢

Example 1. A California Gold Mine

Table 12. Hoisting

Labor						
Main hoist brakemen,	90	shifts	@	$3.50	.....	$315.00
N. S. hoist "	90	"	@	2.50	.....	225.00
Timbermen,	17	"			.....	55.25
Repairing hoist,	11	"			.....	38.00
(Main hoist driven by compressed air. See Table 22)						

Totals..... $633.25

Cost per ton mined, 17¢, made up of: labor, 7¢; supplies,
Cost per ton of ore from stopes, 16¢; per ft of development
Tons hoisted: ore, 9 487 tons; waste, 625 tons; total, 10 11
Average hoisting depth is equivalent to 1 700 ft vertical;
min; average load hoisted 2.1 tons.
Air consumption equals 130 cu ft free air per ton hoisted
free air per shaft h p hr.

Table 13. Pumping

Labor					Supp
Pumpmen,	No 1, 120	shifts	....	$427.50	Lubricants, N
"	No 2, 33	"	....	107.25	" N
Mechanics,	No 1, 8	"	....	27.00	Iron and steel
Electricians,	No 1, 7	"	....	18.00	Power, No 2 (water).....
(Note 8)					Power, No 1 (electric)...

Totals..... $579.75

Cost of pumping per ton mined: labor, 6¢; supplies, 1¢; po
Cost per ton stoped, 19¢; per ft of development work, 67¢.

Table 14. Office, Foremen, Boss

Labor		Candles, 6 bo
Officials.....	$ 847.00	
Foremen.....	300.00	
Bosses.....	708.00	
Survey helper.....	36.75	

Totals..... $1 891.75

Cost per ton mined, 20¢; per ton stoped, 19¢; per ft of dev

Table 15. Miscellaneous

Labor			
Timekeepers, 53	shifts.....	$159.00	General hardw
Watchmen, 61	"	168.00	Waste.....
Sundry surface labor.....		293.12	Candles, 1½ b
			Handles.....
			Sundries.....
			Belting.....
			Packing.....
			Brick and cem
			Lights.....
			Electric railw

Totals..... $620.12

Cost per ton mined, 13¢; per ton stoped, 12¢; per ft of dev

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1. A California Gold Mine (*Continued*)

Table 16. No 1 Mill

6½- to 7-in drops per min. Screens punched 288 0.04-in holes per sq in

Labor		Supplies and power	
emen, 30 shifts..	$270	Gen hardware, etc.....	$ 19
anics.....	102	Lub, belting and packing	10
90 " ..	300	Dies.....	67
57 " ..	142	Screens.....	46
.....	5	Castings.....	249
.....	38	Miscellaneous.....	133
		Power (Table 23)..... (Note 9)	605
.....	$857	.....	$1 129

280. Cost per ton treated: labor, 20.1¢; supplies, 12.2¢; 5.4¢ per ton treated.

Table 17. No 2 Mill

50-lb stamps with same drop, etc, as No 1

Labor		Supplies and power	
emen, 30 shifts..	$270	Gen hardware, etc.....	$ 6
nics.....	32	Lub, belting and packing.	45
90 " ..	300	Dies.....	83
60 " ..	150	Quicksilver.....	40
.....	10	Miscellaneous.....	40
		Power (Table 23)..... (Note 9)	675
.....	$762	.....	$889

90. Cost per ton treated: labor, 17.4¢; supplies, 4.8¢; power, r ton treated.

Table 18. No 1 Mill Concentrators

(9 Dodd Buddles)

Labor		Supplies and power	
eman.....	$ 35	Fuel oil.....	$ 3
s.....	273	Lubricants.....	2
.....	30	Foundry work.....	?
.....	50	Assay supplies.....	5
		Power and electric railway (Table 23)	95
.....	$388	.....	$105

80. Cost per ton: labor 9.1¢; supplies, 0.2¢; power, 2.2¢; eated. Tons of concentrators produced, 92.

Table 19. No 2 Mill Concentrators

(9 Dodd Buddles)

Labor		Supplies and power	
eman.....	$ 35	Fuel oil.....	$ 3
s.....	270	Lubricants.....	2
.....	30	Assay supplies.....	5
.....	50	Power and electric ry..... (Table 23)	80
.....	$385	.....	$ 90

90. Cost per ton: labor, 8.8¢; supplies, 0.2¢; power, 1.8¢; ented. Tons of concentrates produced, 70.

Example 1. A California Gold Mine (*Cont*

Table 20. No 1 Cyanide Plant

Labor		Supplies and
Superintendent and foreman.	$210	Fuel oil
Blacksmiths and mechanics..	16	Tool steel
Assayers	85	General hardware, etc.
Cyaniders	329	Lubricants, packing, bel
		Assay supplies
		Lime (4.21 lb per to
		Zinc (0.216 "
		Cyanide (0.61 "
		Chemicals
		Miscellaneous
		Power (Table 23)
Totals	$640	

Tons treated, 4 180. Cost per ton: labor, 15.3¢; supplies, 20
total, 42.5¢ per ton treated.

Table 21. No 2 Cyanide Plant

Labor		Supplies and
Superintendent and foreman.	$205	Fuel oil
Blacksmiths and mechanics..	24	General hardware, etc.
Assayers	90	Lubricants, packing, belt
Cyaniders	487	Assay supplies
		Electrical supplies
		Lime (4.21 lb per to
		Zinc (0.216 "
		Cyanide (0.61 "
		Chemicals
		Miscellaneous
		Power (Table 23)
Totals	$806	

Tons treated, 4 452. Cost per ton: labor, 18.1¢; supplies, 20.6¢;
44.7¢ per ton treated.

Table 22. Compressed Air
(One month)

Cost of air		Distribution	
			Cu ft free air
Water power: 6 260 miner's in @ 700 ft head or 406 h p month delivered at nozzle	$1 002	Main hoist and dolley cage	22 000
Elect power, 113 h p month delivered at transformers	456	Und'ground hoists	1 500
Attendance at compressor	540	" small pump	1 000
Ditch tender	45	Air drills	29 000
Superintendent	75	Blowing	2 500
Lubricants and waste	111	No 1 cyanide mill	3 300
Maintenance, labor	59	Blacksmith shop	3 500
Maintenance, supplies	68	Assay office	250
		Machine shop	1 250
		Known leakage	17 000
		Unaccounted for	3 000
Totals	$2 356	(Note 12)	84 300
Cost per 1 000 cu ft of free air, 2.78 ¢			

STANFORD LIBRARIES

1. A California Gold Mine (*Concluded*)

Table 23. Power

ower			Distribution of power		
	Cost	Per h p		H p mo	Cost per mo
r			No 1 mill:		
e			Rock breakers....	8 elect	$ 33.20
..	$1 659.04	$2.690	Stamps.....	77 "	312.10
t			"	50 water	285.00
e			Concentrators,	8 elect	33.20
..	3 105.30	4.160	No 2 mill:		
f			Rockbreakers and		
o.	4 764.34	3.500	water.....	43 water	145.00
..	125.00	0.092	Stamps.....	130 elect	522.00
s.	196.00	0.144	Concentrators....	10 "	40.30
..	540.00	0.397	No 1 cyanide:		
..	90.00	0.066	Compressors.....	19 "	77.50
..	8.75	0.006	Pilters and pump.	10 "	41.30
..	17.00	0.012	Miscellaneous....	5 "	21.10
s.	68.64	0.050	Water.....	26 water	150.00
..	100.00	0.073	No 2 cyanide:		
..	106.00	0.078	Tube mill.....	10 elect	40.30
..	120.00	0.088	Compressor.....	18 "	72.00
			Pilters and pump.	20 "	80.60
..	$6 135.73	$4.506	No 2 pump (Note 8)	60 "	343.84
7)	120.00	0.088	No 1 pump (Note 8)	276 "	1 115.00
			Shops.....	6 "	19.00
			General.....	15 "	86.00
			Lights and railway.	59 "	241.70
			Main compressors (Table 22).....	519	$2 356.59
..	$6 015.73	$4.418	.....	1 369	$6 015.73

pruce poles cost 1¢ per in of diam per linear ft of pole. Pine diam per linear ft. Avg cost per running ft of both, 5.33¢.

he gravity tram is used in lowering ore from stopes down track requires.

ir distribution for sharpening drills shown in Table 22.

ir-drill repair parts are made in the company's own shop.

his work consists of miscellaneous jobs around the shaft; taking e, and other small jobs.

This account is for handling ore from the 3 400- and 3 700-ft levels s at the 4 000-ft station.

Fuel oil for reheating, to the amount of $120 and included in e 23, should be charged direct to hoisting.

No 1 is an 8-stage pump lifting 240 000 gal per 24 hr, 1 400 ft ing 140 000 gal per 24 hr, 475 ft. The 8-stage pump has been age pump, effecting a saving of $400 per month in power. No 2 g 220 000 gal per 24 hr, total head 725 ft, avg head about 350 ft. nd 17. The average loss per ton of ore treated is: for shoes, ; for dies, 0.525 lb cast iron; liners, 0.094 lb cast iron; cam-iron, per ton of ore.

and 17. Of the rock sent to the mill the rock breakers at the % as waste.

and 19. Mill foreman looks after concentrators, and cyanide nills.

These figures were obtained by air-meter tests on the various akage was obtained by the number of revolutions required on the e air pressure at 100 lb. The leakage had been reduced from er month, or over 25% of compressor capacity.

Example 2. Goldfield Consolidated Mines Co

Table 24. Summary of Operating Res

For year ended Oct 31	1909	1910
Bal quick assets forwarded	$1 064 194	$ 2 225 318
Bal quick assets, end period	2 225 318	2 350 576
Gain in quick assets (*loss)	1 161 124	125 258
Dividends paid during period	3 201 238	7 118 271
Bal over expenditures	4 362 362	7 243 529
Receipts including selling chg's	6 898 571	10 322 115
Total expenditures	2 536 209	3 078 586
General items:		
Mining (Table 26)	839 178	1 032 059
Transport ore to mill (Table 26)	18 158	36 800
Milling (Table 26)	460 646	562 881
Conc't treatment (Table 26)	45 525	82 713
Bullion tax	117 463	130 808
General expenses	.....	.....
Administration	Sundries	39 684
General and Secretary's office	20 855	26 571
Legal expense	4 543	36 304
Corporation expense	Sundries	27 823
Damages	"	19 334
Association dues	"	7 338
Property tax	2 345	15 189
General maintenance	Sundries	24 022
Automobiles	"	8 770
Sundries	62 609	33 270
Insurance	929	.....
Selling charges on products	299 701	372 130
Ore purchased	.....	.....
Total operating expenditure	$1 871 952	$ 2 455 696
Construction and other expenses	664 257	550 027
Federal income tax	.....	72 863
Total expenditure	$2 536 209	$ 3 078 586
Tons of ore produced (Table 25)	194 480	266 867
Operating cost per ton ore	$9.63	$ 9.24
Total expenditure per ton ore	$13.04	$11.54
Average grade of ore mined	$37.98	$40.72

Table 25. Production from Mines

Year ended Oct 31	1910
Ore produced from development work, tons	34 699
Ore produced from stopes, tons	232 168
Total tons of ore produced, all mines	266 867
Average grade of ore from development	$25
Average grade of ore from stopes	$43
Average grade of all ore produced (gold)	$40.72
Tons of ore treated at mill	265 352
Tons of ore shipped	1 515

STANFORD LIBRARIES

Goldfield Consolidated Mines Co (*Continued*)

e 26. Operating Costs per Ton Mined

d Oct 31	1910	1911	14 mos to Dec 31, 1912
elopment:			
.....	$2.08	$2.22	$2.34
.....	1.28	0.99	0.89
.....	0.046	0.14	0.16
.....	0.464	.....	.....
e 28).....	$3.87[1]	$3.35	$3.39
nage:			
.....	$0.079	$0.06	$0.06
ce.....	0.057	0.03	0.03
re to mill.....	$0.136	$0.09	$0.09
tonnage)			
.....	$0.558	$0.43	$0.39
.....	1.258	1.17	0.94
.....	0.305	0.29	0.33
.....	$2.121	$1.89	$1.66
.....	.....	.....	.....
per ton milled:			
.....	$0.05	$0.04	$0.07
.....	0.236	0.32	0.27
.....	0.026	0.02	0.05
nt per ton milled....	$0.312	$0.38	$0.39
per ton conc't:			
.....	$0.93	$0.63	$1.24
.....	0.44	0.40	0.76
.....	4.48	5.09	4.50
't treated.....	$5.85	$6.12	$6.50
treated.....	14 152	20 306	24 376
ork performed.....	41 938	46 739	48 146
levelopment.....	$9.05	$7.51	.....

ren in Table 26 cover operations during periods when the 100-mill in operation. The company operated a 20-stamp mill up efly, the operations in this mill consisted of: crushing by 8- by eying by 14-in by 165-ft belt, running on a 20° incline to stamp stamps, 109 6½-in drops per min, through 12-mesh screen; ing, reamalgamating; separating to sands and slimes; con- s and agitating slimes; filtering through Butters' filters; zinc and precipitates to bullion and concentrates to smelter. The was practically all oxidized. Table 27 gives an example of the ained at the plant, excluding depreciation of plant.

ble 27. Operations, 20-Stamp Mill

Average value per ton, $37.15. Net recovery, 93.80%, as ion, 41.40%; concentration, 18.20%; cyaniding, 34.20%.

..	Nov	Dec	Jan	Feb	Mch	Apr	May	June	July	Aug
..	$1.77	$2.06	$1.73	$1.93	$1.65	$1.73	$1.67	$1.65	$1.65	$1.58
..	1.45	1.74	1.59	1.28	1.38	1.45	1.11	1.17	1.16	1.21
..	0.23	0.31	0.18	0.22	0.18	0.19	0.18	0.20	0.18	0.18
..	0.49	0.61	0.48	0.63	0.53	0.67	0.66	0.63	0.42	0.42
..	0.24	0.24	0.23	0.28	0.26	0.36	0.21	0.35	0.08	0.08
..	0.29	0.36	0.26	0.27	0.23	0.34	0.37	0.36	0.36	0.36
..	$4.47	$5.32	$4.47	$4.61	$4.23	$4.74	$4.20	$4.36	$3.85	$3.83

Example 2. Goldfield Consolidated Mines Co

Table 30. Employes in Mining Department.

	No of men	Aver daily wage
Mining:		
Machine drill men	155	$4.00
Shovelers	159	3.75
Timbermen	30	4.18
Trammers	30	3.75
Topmen	21	3.77
Stationmen	11	4.00
Watchmen, underground	1	5.00
Samplers	3	4.61
Hoisting engineers	14	5.00
Pipe and trackmen	10	4.10
Pumpmen	3	4.00
Nippers (powder and tool men)	11	3.75
Foreman and shift bosses	12	5.64
Underground supt.	1	Salary
Total	461	.....
Offices:		
Timekeepers	3	4.42
Storekeeper	1	6.67
Storehouse clerk	1	4.17
Bookkeeper	1	6.67
Stenographer	1	4.17
Purchasing agent	1	Salary
Janitor	1	3.50
Total	9	.....
Electrical dept:		
Chief electrician	1	6.67
Sub-station attendants	2	4.00
Sub-station attendant and repairman	1	5.00
Repairmen	2	5.00
Total	6	.....
Surveying:		
Chief surveyor	1	10.00
Surveyor	1	5.83
Surveyor's assistants	2	5.00
Total	4	.....

Machine shop and su		
Master mechanic		
Blacksmith		
Blacksmith helper		
Blacksmiths and sharpeners		
Compressor men		
Machinists		
Machinists' helpers		
Carpenters		
Watchmen		
Sawyer and yard man		
Laborers		
Dry house		
Surface supt.		
Total		
Assay dept:		
Day foreman		
Night foreman		
Balance men		
Parter		
Fluxman		
Furnace men		
Clerk and helper		
General		
Total		
Diamond drill:		
Foreman		
Drill runners		
Drill runners' helpe		
Total		
Mill water supply:		
Pumpmen		
Pipe lineman		
Total		

See Table 31 for mill employes.

Tons of ore mined per annum .....

Average amount of development work performed per annum .....

Drifts and crosscuts about 5 by 7 ft in clear. Raises ave clear.

Cost of mining plant per ton of annual production....

Total cost of mining plant ..... $

STANFORD LIBRARIES

ield Consolidated Mines Co (*Continued*)

in 100-Stamp Milling Department, 1913

Aver daily wage	See Table 30 for mine employes	No of men	Aver daily wage
	Concentrate treatment(*con*):		
$4.50	Solution men	3	$4.50
4.00	Solution men helpers	3	3.75
	Laborers	2	3.87
4.50	General:		
4.00	Foremen	3	6.11
4.50	Master mechanic	1	8.33
	Storekeeper	1	5.83
4.50	Time-keeper	1	5.00
4.00	Electrician	1	5.00
	Chemist	1	5.00
4.00	Carpenters	2	5.25
	Carpenters' helper	1	4.00
4.50	Machinists	2	5.25
	Machinists' helper	1	4.00
5.83	Blacksmith	1	5.00
4.50	Blacksmith helper	1	4.00
	Sampler	1	4.00
4.50	Pipe fitter	1	5.00
4.75	Pipe-fitter's helper	1	4.00
	General repairmen	2	4.25
4.50	Laborers	10	3.55
4.00	Railroad:		
4.50	Conductor and foreman	1	6.50
	Engineer	1	5.00
4.00	Fireman	1	4.00
4.00	Hostler and repairman	1	4.50
4.00	Brakemen	3	4.00
	Track foreman	1	4.00
5.83	Track laborer	1	3.50

Milling Costs and Data, 1913

d	$0.457
	1.105
	0.414
	1.976
346 000	
$1 148 000	
of annual capacity $3.318	
n of ore used:	
	1.92 lb
	13.62 "
	0.61 "
	0.95 "
	0.47 "
	0.29 "
	3.92 "
	316 gal
	1.19 h p

Example 2. Goldfield Consolidated Mines

Cost of Principal supplies laid down in Goldf

Explosives:
- Blasting caps....XXXXX.... $35.65 per case....
- Fuse.....Comet..... 22.80 " " ....
- Fuse.....Eclipse..... 26.75 " " ....
- Powder, E L F 40% 1⅛-in .....
- Powder, Judson F 10% N G L.....

Candles, Granite, extra hard (240 in case).....

Timber:
- Posts and caps framed.....
- Ties.....
- Lagging.....
- Miscellaneous lumber, mining grade, Oregon pine...
- " " native .....
- Wedges.....

Round Timber:
- 5 to 6-in small end.....
- 8 to 9-in " " .....
- 10 to 12-in " " .....

Shovels, Kentwood, long handle, round point.....
" " " " square point.....

Drill steel, Phœnix.....
" " Corona.....
" " Midvale.....

Drill hose, ¾-in 5-ply Falcon .....
" " ¾-in 5-ply Diamond.....
" " 1-in 5-ply " .....

Mine track: Rails, ties, plates, bolts and spikes.....

Scale of Wages at Mines

- Shift bosses.....
- Carpenters.....
- Blacksmiths.....
- Hoisting engineers.....
- Cagers.....
- Pumpmen.....
- Top trammers.....
- Miners.....
- Muckers.....
- Shaftmen.....
- Timbermen.....
- Nippers.....
- Pipemen.....
- Trackmen.....

STANFORD LIBRARIES

Goldfield Consolidated Mines Co (*Continued*)

ng Costs, All Mines, June, July, August, Sept, 1913

	Stopes			
	Quantity		Cost in dollars	
	Total	Per ton	Total	Per ton
.....	60 400 pc	0.544	434.91	0.004
.....	376 400 ft	3.387	1 505.61	0.013
.....	97 227 lb	0.875	13 611.75	0.123
.....	153 360 pc	1.380	1 948.45	0.018
.....	165 000 bft	1.485	45 793.09	0.412
.....	43 338 ft	0.390	8 234.25	0.074
.....	93 740 pc	0.843	703.05	0.006
.....	.....	.....	2 150.55	0.019
.....	90 pc	0.001	77.72	0.001
.....	386 pc	0.003	359.37	0.003
.....	4 275 lb	0.040	342.05	0.003
.....	573 ft	0.005	160.41	0.002
.....	.....	.....	424.75	0.004
.....	.....	.....	.....	.....
s.....	.....	.....	4 671.09	0.042
.....	.....	.....	155.88	0.001
.....	.....	.....	80 572.93	0.725
	Shifts			
	Total	Per ton		
.....	.....	.....	.....	.....
.....	6 402	0.058	25 608.00	0.231
.....	2 576	0.023	10 304.88	0.093
ers.....	13 447	0.121	50 427.57	0.453
.....	3 636	0.033	14 546.86	0.131
.....	3 292	0.029	12 344.35	0.111
.....	983	0.009	3 814.68	0.034
.....	30 336	0.273	117 046.34	1.053
.....	.....	.....	21 273.30	0.191
.....	.....	.....	1 624.16	0.014
sharpener.....	.....	.....	7 709.11	0.069
.....	.....	.....	2 667.04	0.024
.....	.....	.....	6 746.83	0.066
.....	.....	.....	4 646.11	0.042
.....	.....	.....	2 936.44	0.026
.....	.....	.....	11 148.56	0.100
.....	.....	.....	3 351.24	0.030
.....	.....	.....	10 395.55	0.093
.....	.....	.....	2 934.37	0.026
.....	.....	.....	985.20	0.009
tools).....	.....	.....	2 529.98	0.022
.....	.....	.....	3 642.16	0.032
.....	.....	.....	3 372.55	0.030
bor and supplies)	.....	.....	85 962.60	0.774
.....	.....	.....	283 581.87	2.552
lopment.....	111 136	.....	.....	.....

Example 2. Goldfield Consolidated Mines

Table 33. Operating Costs, All Mines, June, July, Aug.

Drifts and crosscuts					
Quantity		Cost in dollars		Quantity	
Total	Per ft	Total	Per ft	Total	Per ft
15 400	1.706	111.07	0.012	4 700	1.30
95 400	10.571	381.78	0.042	30 000	8.33
24 716	2.739	3 460.22	0.383	7 686	2.13
41 760	4.627	529.51	0.059	12 000	3.33
243 603	26.992	6 699.09	0.743	105 234	29.23
6 829	7.667	1 297.25	0.144	2 580	0.71
15 551	1.723	116.63	0.013	5 800	1.61
.....	.....	660.00	0.073	.....	.....
25	0.003	22.54	0.002	10	0.00
106	0.011	98.17	0.011	47	0.03
1 266	0.140	101.31	0.012	512	0.12
166	0.018	46.51	0.005	66	0.01
.....	.....	116.92	0.013	.....	.....
3 692	0.409	369.19	0.041	.....	.....
.....	.....	1 276 41	0.141	.....	.....
.....	.....	41.38	0.004	.....	.....
.....	.....	15 327.98	1.698	.....	.....
Shifts				Shifts	
Total	Per ft			Total	Per ft
408	0.045	1 632.00	0.181	.....	.....
2 166	0.240	8 664.00	0.960	1 153	0.32
848	0.094	3 394.07	0.376	390	0.10
4 139	0.458	15 521.17	1.720	1 206	0.33
1 064	0.118	4 257.23	0.471	319	0.08
864	0.096	3 241.48	0.359	291	0.08
224	0.025	870.47	0.097	85	0.02
9 713	1.076	37 580.42	4.164	3 444	0.95
.....	.....	4 416.14	0.490	.....	.....
.....	.....	391.24	0.043	.....	.....
.....	.....	1 878.16	0.209	.....	.....
.....	.....	1 062.50	0.118	.....	.....
.....	.....	2 711.03	0.301	.....	.....
.....	.....	1 861.84	0.207	.....	.....
.....	.....	1 161.26	0.129	.....	.....
.....	.....	2 713.83	0.301	.....	.....
.....	.....	815.92	0.090	.....	.....
.....	.....	2 535.17	0.282	.....	.....
.....	.....	711 81	0.079	.....	.....
.....	.....	241.71	0.022	.....	.....
.....	.....	614.56	0.069	.....	.....
.....	.....	886.83	0.098	.....	.....
.....	.....	819.91	0.091	.....	.....
.....	.....	22 821.91	2.529	.....	.....
.....	.....	75 730.31	8.391	.....	.....
9 025 ft				3 599 ft	

Old track was used almost exclusively.

STANFORD LIBRARIES

a-Treadwell Mining Co, Douglas Island, Alaska

Consumed Per Ton, Over a Period of Five Years

e mined

.....	1.925 lb
.....	4.612 ft
.....	0.922 pc
.....	0.113 lb
.....	0.125 lb
.....	0.039 lb
.....	0.006 ft
.....	0.25 lb
.....	0.064 lb
.....	0.744 ft
.....	0.008 gal
.....	0.018 ft

Milling, per ton ore milled

240-stamp mill:
Shoes, 0.331 lb per ton milled.
Dies, 0.193 lb per ton milled.
Mortar extras, 0.135 lb per ton milled.
Cams, tappets, boss heads, 0.030 lb per ton milled.
Mortars crushed, 60 054 tons per mortar.
Quicksilver, 1 flask for 17 158 tons milled.
Cam shafts, milled 90 081 tons per cam shaft.
Stamp duty, 4.59 tons per 24 hr.
Stamp weight, 850 lb per stamp.

day per ton ore

Gen	Total
0.160	1.443
0.178	1.434
0.254	2.045
0.265	2.030
0.296	2.116
0.322	2.252

300-stamp mill:
Shoes, 0.396 lb per ton milled.
Dies, 0.223 lb per ton milled.
Mortar extras, 0.142 lb per ton milled.
Cams, tappets, boss heads, 0.032 lb per ton milled.
Mortars crushed, 536 530 tons per mortar.
Quicksilver, 1 flask for 14 295 tons milled.
Cam shafts, milled 13 167 tons per cam shaft.
Stamp duty, 5.51 tons per 24 hr.
Stamp weight, 1 050 lb per stamp.

ummary of Receipts and Expenditures, 1912

		Per ton ore
.....	$1 152 524.62	$1.2918
.....	6 876.82	0.0077
trates treated.....	1 046 487.43	1.1729
reated.....	$2 205 888.87	$2.4724
est.....	11 750.74	0.0132
.....	36 959.02	0.0414
.....	5 218.75	0.0059
and other sources.....	$2 259 817.38	$2.5329

ble 35).....	$1.1848
(Table 35).....	0.0171
(Table 35).....	0.0199
Table 35).....	$1.2218
rcial profits, etc, shown in receipts.....	0.0605
r year.....	$1.1613
.....	0.6961
tion.....	$1.8574

Example 3. Alaska-Treadwell Mining

Table 35. Detailed Summary of Expen

Operating costs:	
Mining, development, stoping (Table 37)....	$ 7
Milling, 892 192 tons (Table 38).....	1
Concentrate treatment, 17 397 tons (Table 36)	
San Francisco office expenses.....	
London office expenses.....	
Paris office expenses.....	
Legal expenses.....	
Taxes.....	
Bullion charges.....	
Total operating charges.....	$1 0
Construction:	
Central shaft.....	
Electric locomotive round house.....	
Foundry coke house.....	
Pattern shop store room.....	
Loss on dwellings for year.....	
Loss on boarding house for year.....	
Loss on wharf for year.....	
Total charges against ore production (depreciation not included).....	....

Table 36. Conc't Treat't Costs

	Per ton
Labor.....	$0.520
Supplies.....	0.096
Train service.....	0.108
Sulphuret car expense..	0.016
Assaying.....	0.018
Cyaniding.....	3.122
Douglas Island gen exp	0.054
San Francisco office, freight, refining, etc	0.634
Total.....	$4.568
Tons treated, 17 397	

Rate o
Machine drillers.
Machine helpers.
Mine laborers...
Amalgamators..
Feeders.....
Vannermen...
Machinists and he
Blacksmiths.....
Tool sharpeners..
Blacksmith helper
Average number
Average wage paid
Tons of ore mine per man emplo

STANFORD LIBRARIES

Alaska-Treadwell Mining Co (*Continued*)

etails of Mining Costs, 1912 (per ton milled)

	Development 1 258 ft C per ft	Stoping 861 973 tons	Tramming, 892 192 tons	Hoisting, 892 192 tons	Pumping per ton milled	Total per ton milled
.....	$3.188	$0.143	.....	.....	.....	$0.183
.....	.....	0.003	.....	.....	.....	0.004
.....	1.551	0.114	$0.026	.....	.....	0.158
.....	0.072	0.010	.....	.....	.....	0.011
.....	0.018	0.001	.....	.....	.....	0.001
ers...	.....	.....	0.003	0.020	.....	0.023
.....	.....	.....	.....	0.004	.....	0.004
.....	.....	.....	0.015	.....	.....	0.014
s....	.....	.....	0.029	.....	.....	0.030
en...	0.069	.....	0.003	.....	.....	0.004
.....	.....	.....	.....	.....	0.003	0.003
es...	0.177	0.009	0.002	.....	.....	0.014
harp-l ....J	0.153	0.006	0.001	.....	.....	0.008
.....	0.056	0.002	.....	.....	.....	0.003
nen..	0.244	0.003	0.001	0.001	.....	0.007
.....	0.022	.....	.....	.....	.....	0.001
.....	1.241	.....	0.003	.....	.....	0.020
.....	0.025	0.001	.....	.....	.....	0.001
.....	$6.816	$0.292	$0.083	$0.025	$0.003	$0.489
.....	1.239	0.146	.....	.....	.....	0.157
.....	0.120	0.016	.....	.....	.....	0.017
.....	0.078	0.006	.....	.....	.....	0.007
.....	0.131	0.006	.....	.....	.....	0.008
.....	0.022	0.003	0.001	.....	.....	0.006
.....	.....	.....	0.001	0.008	.....	0.009
.....	0.095	0.003	.....	0.001	.....	0.006
.....	0.007	.....	0.001	0.001	.....	0.002
.....	0.001	.....	.....	.....	.....	.....
.....	.....	.....	0.003	.....	.....	0.003
.....	0.031	0.001	.....	.....	.....	0.001
.....	0.192	0.002	0.003	0.001	0.001	0.009
.....	$ 2.016	$0.183	$0.009	$0.011	$0.001	$0.225
.....	0.022	0.001	.....	.....	.....	0.002
.....	0.039	.....	.....	.....	0.001	0.002
.....	0.169	0.002	0.002	0.008	.....	0.015
.....	0.236	0.007	0.001	0.030	0.003	0.054
.....	0.041	0.002	.....	.....	.....	0.003
.....	0.013	0.001	.....	.....	.....	0.001
.....	0.052	0.001	.....	.....	.....	0.002
nse..	0.675	0.031	0.008	0.004	0.001	0.051
.....	0.494	0.018	0.001	.....	0.001	0.027
.....	0.019	0.001	.....	.....	.....	0.001
.....	0.137	0.005	0.001	0.001	.....	0.008
sup- ....J	$ 1.907	$0.069	$0.013	$0.053	$0.006	$0.166
nts..	$10.739	$0.544	$0.105	$0.089	$0.010	$0.880
.....	1.803	.....	.....	0.001	0.002	0.028
elop- re...J	$ 8.936	$0.544	$0.105	$0.088	$0.008	$0.852

STANFORD LIBRARIES

Example 3. Alaska-Treadwell Mining Co (*Concluded*)

Table 38. Details of Milling Costs, 1912 (per ton milled)

	240-Stamp mill, 371 308 tons					300-Stamp mill, 520 884 tons				
	Crushing	Tramming	Stamping	Concentrating	Total	Crushing	Tramming	Stamping	Concentrating	Total
Laborers	.....	.....	.....	.....	.....	.....	.....	$0.002	$0.001	$0.003
Feeders	.....	.....	$0.025	.....	$0.025	.....	.....	0.017	.....	0.017
Amalgamators	.....	.....	0.015	.....	0.015	.....	.....	0.010	.....	0.010
Vannermen	.....	.....	.....	$0.016	0.016	.....	.....	.....	0.011	0.011
Samplers	.....	.....	.....	0.001	0.001	.....	.....	.....	.....	.....
Foremen	.....	.....	0.003	0.002	0.005	.....	.....	0.002	0.002	0.004
Oilers	.....	.....	0.003	0.003	0.006	.....	.....	0.002	0.002	0.004
Carpenters	.....	.....	.....	.....	.....	.....	.....	0.002	0.001	0.003
Stationary engineers	.....	.....	0.006	.....	0.006	.....	.....	.....	.....	.....
Watchmen	.....	.....	.....	.....	.....	.....	.....	0.001	.....	0.001
Shoes and dies	.....	.....	0.016	.....	0.016	.....	.....	0.017	.....	0.017
Cams and cam shafts	.....	.....	.....	.....	.....	.....	.....	0.003	.....	0.003
Tappets, keys, and mortars	.....	.....	0.002	.....	0.002	.....	.....	0.002	.....	0.002
Mortar extras	.....	.....	0.004	.....	0.004	.....	.....	0.004	.....	0.004
Screens	.....	.....	0.002	.....	0.002	.....	.....	0.002	.....	0.002
Quicksilver	.....	.....	0.002	.....	0.002	.....	.....	0.003	.....	0.003
Lubricants, vanner supplies, etc.	.....	.....	0.002	0.004	0.006	.....	.....	0.001	0.002	0.003
Miscellaneous	.....	.....	0.002	0.002	0.004	.....	.....	0.005	0.003	0.008
Trailings account	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

4. Camp Bird Limited, Ouray, Colorado

Costs Per Ton of Ore Treated, 14 Mos ended June 30, 1912

....	$0.33	Cyanide mill:	
....	0.46	Labor	$0.17
....	0.35	Supplies, including lights.	0.31
g...	0.73	Assaying	0.03
....	0.17	Foremen, etc.	0.06
....	0.01	Maintenance	0.03
....	0.12	Other expenses	0.07
l...	0.02	Total cyanide expense	$0.67
ving	0.04	Shipping and selling:	
....	0.19	Guards, etc.	0.05
....	0.29	Mint charges on bullion	0.10
....	0.28	Freight and treat on conc't	1.98
....	0.06	Total shipping and selling expense	2.13
....	$3.05	General expenses:	
		Consulting engineer	0.07
....	0.18	Manager's office	0.48
....	0.03	Telephone and telegraph	0.02
e...	0.21	Legal expense	0.08
		Audit fees and expense	0.01
....	0.63	Insurance	0.16
el...	0.20	Other maintenance	0.13
g...	0.06	Taxes	0.79
....	0.04	Miscellaneous	0.24
....	0.26	Total general expense	1.98
		Depreciation, plant and mach'y	0.59
....	1.19	Total working expense	$9.82

Miscellaneous income, per ton	$26.13
Ore treated (not deducting London office expenses)	16.31

etc:	Per ton
.....	$17.19
.....	8.86
.....	0.03
s.....	0.02
ances...	0.03
.....	$26.13

Total tons ore treated, 66 505. Aver contents, Au, 1.144 oz; Ag, 3.69 oz; Pb, 14.38%; Cu, 0.23%. Aver extraction of gold contents, 94.58%. Tons ore actually cyanided, 44 430.

land Gold Mining Co, Cripple Creek, Colorado

s 40. Summary of Operating Results

	1908	1909	1910	1911
rded.....	$ 487 357	$ 470 106	$ 334 072	$ 313 704
year.....	470 106	334 072	313 704	372 889
s (*loss).....	$* 17 251	$* 136 034	$* 20 368	$ 59 185
.....	480 000	330 000	240 000	240 000
.....	$ 462 749	$ 193 966	$ 219 632	$ 299 185
and treated....	1 834 081	1 438 650	1 354 421	1 485 622
.....	2 788	3 772	5 045	9 665
.....	$1 836 869	$1 442 422	$1 359 466	$1 495 287
.....	1 374 120	1 248 456	1 139 834	1 196 102
.....	94 312	83 908	113 752	171 220
.....	14.57	14.88	10.02	6.98
on.....	19.45	17.15	11.90	8.67

Example 5. Portland Gold Mining Co (

In 1910 the Portland commenced treating its waste dumps; tons were treated, yielding $113 253. In 1911, 120 961 tons, $3.51 per ton, were treated. The yield from this ore was $345 5 81.4% and profits $128 738, indicating a cost of $1.80 per ton fo Regular ore mined amounted to 50 259 tons, averaging about total cost of mining and realization was apparently about $1 mine ore.

Example 6. Liberty Bell Gold Mining Co, Tel

Table 41. Summary of Operating Receipts and

1912			191
		Totals	
Receipts:			
Ore proceeds, 175 340 tons ore		$1 203 726.43	$6
Stamp bullion	$564 718.82	.....	3
Cyanide products	484 771.78	.....	2
Concentrates	154 235.83	.....	0
Slags		.....	
Boarding house profit (Table 44)		1 634.11	
Tram receipts		4 811.69	
Rental receipts		3 162.80	0
Commissary profit (Table 44)		3 485.91	
Interest and profits from investments		25 357.34	
Total Receipts and Aver per Ton Milled		$1 242 178.28	$ 7
Expenses:			
Gen expense, 175 340 tons (Table 42)		$ 106 793.51	$ 0
Mining, 175 052 tons (Table 42)		324 595.79	1
Operating	$294 750.99	.....	1
Development, 3 636 ft	29 844.80	.....	0
Tramming, 174 103 tons (Table 42)		28 653.37	0
Milling, 175 340 tons (Table 42)		257 522.84	1
Treatment and shipment costs		49 449.66	0
Stamp bullion, 70 416 oz	$ 2 488.76	.....	0
Concentrates, 2 892 tons	43 350.00	.....	14
Cyanide bullion, 221 226 oz	3 610.90	.....	0
Total Operating Expenses		$ 767 045.17	$ 4
Total Profit and Averages per Ton		$ 475 133.11	$ 2

Summary of Ore Proceeds and Expenses, 1899

Stamp bullion, gross value	$1 9
Concentrates " "	1 0
Cyanide bullion " "	2 6
Cyanide precipitates, gross value	1
Slag " "	
Total proceeds, 1 217 952 tons of ore milled	$8 7
Total expense, less miscellaneous receipts	6 5
Profit balance	$2 2
Expended on mining property and construction, less depreciation already charged, about	3
Receipts over all expenditures	$1 9

STANFORD LIBRARIES

Liberty Bell Gold Mining Co (*Continued*)

42. Details of Operating Expenses, 1912

erintendence:	
exp..	$ 7 671.79
.....	18 270.25
.....	16 982.05
onnec-	
on....	2 192.22
.....	5 566.44
.....	758.56
.....	250.70
irs...	1 503.01
.....	30 539.17
.....	23 059.32
e.....	$106 793.51
43):	
.....	11 886.55
f vein	62 218.37
.....	98 176.64
g.....	38 063.28
.....	27 639.17
.....	174.70
.....	2 030.93
.....	13 136.75
irs...	27 042.20
.....	44 227.20
tons.	$324 595.79

Cable Tram (Table 43):	
Foremen.....	$2 272.60
Loading station labor...	9 145.55
Discharge station labor..	7 032.10
Depreciation, 5%.....	2 111.31
Maintenance and repairs.	7 252.78
Supplies.....	839.03
Total tram, 174 103 tons.	$28 653.37
Milling (Table 43):	
Superintendence.....	$ 4 692.10
Crushing.....	10 717.80
Concentrating.....	9 914.27
Stamping.....	29 689.76
Amalgamating.....	8 983.59
Regrinding.....	10 994.39
Settling and agitating...	11 275.76
Filtering.....	9 796.88
Precipitation.....	11 999.42
Pumping.....	6 406.10
Cyanide.....	46 828.03
Lime.....	9 400.99
Tools.....	904.78
Sundries.....	22.00
Assaying and melting...	6 239.76
Heating and lighting....	13 509.67
Depreciation.....	16 963.60
Maintenance and repairs.	43 876.85
Lubrication.....	2 081.65
Tests and watchmen....	3 257.44
Total milling.....	$257 552.84

nts of ore and per cent of recovery per ton milled

	1912						1899 to 1912	
	Gold		Silver		Value		Value	
	Ounces	%	Ounces	%	$	%	$	%
	0.149	50	0.239	9	3.22	41	4.10	46
	0.025	8	0.523	20	0.88	11	0.87	9
	0.105	35	1.027	38	2.77	36	2.25	25
	0.279	93	1.789	67	6.87	88	7.22	80
	0.022	7	0.891	33	0.95	12	1.74	20

Tons of dry ore produced

Milled	Year	Mined	Milled
16 026	1906	92 221	92 900
21 500	1907	102 429	102 104
54 197	1908	116 138	116 353
67 439	1909	126 436	125 681
83 373	1910	134 321	133 899
45 811	1911	155 593	155 950
27 379	1912	175 052	175 340

dwork to date, 54 137 ft, consisting of: drifts, 35 614 ft; unnels, 3 183 ft; raises, 13 749 ft; sinking, 368 ft.

Example 6. Liberty Bell Gold Minin

Table 43. Details of Principal Supplies, Mai

	Total, 191
Mine:	
Powder, 120 250 lb	$17 438
Fuse and caps	3 713
Candles, 1 039 boxes	3 802
Timber and lumber	27 196
Coal for heating, 306 tons	2 537
Buildings and heating plant	1 472
Cars	4 701
Locomotives	1 975
Drill plant	8 703
Electric plant	5 675
Hoists	892
Bins	980
Skips	2 301
Power	15 799
Cable tram:	
Rope and couplings	2 355
Buckets and trolley	1 485
Towers and stations	2 288
Power	470
Milling:	
Crusher	1 397
Quicksilver, 18 767 oz	831
Shoes, 195 pc	1 737
Dies, 632 pc	1 894
Screens	182
Coal for heating, 1 279 tons	7 961
Cyanide, 184 000 lb	46 598
Zinc, 91 145 lb	8 732
Lime, 1 311 870 lb	8 572
Sulphuric acid, 14 362 lb	662
Lead acetate and litharge	5 528
Tube mills	2 076
Tube mill pebbles	2 025
Tube mill liners	1 502
Batteries	9 645
Agitators	1 548
Pumps and compressors	4 856
Filters, Moore plant	8 153
Zinc boxes	1 765
Building and heating plant	11 289
Electric plant	3 078
Concentration plant	2 541
Settlers	2 347
Pipe lines	893
Classifiers and sizers	1 038
Power	3 190

Table 44. Special Accounts, 1

Boarding house:		Commissar
Labor	$11 469.50	Total cos
Provisions	31 882.12	Profit for
Fuel and lights	5 579.34	Total rec
Rental	2 400.00	
Other expense	2 799.94	Stable oper
Total boarding house exp.	$52 130.90	Laboratory
Receipts	53 765.01	Machine sh
Commissary:		
Mdse on hand, 1911	$ 8 305.60	
Purchases and expenses	18 370.19	
Total expense	$26 675.79	
Mdse on hand, 1912	7 880.00	

STANFORD LIBRARIES

Liberty Bell Gold Mining Co (*Concluded*)

Construction and Equipment to Sept 30, 1912

	Expense, 1912	Total to date	Deprec chg'd off	Book value
.....	.....	$ 6 343.14	$ 3 111.67	$ 3 231.47
.....	.....	5 113.55	2 983.17	2 130.38
.....	$1 192.33	332 665.73	154 952.24	177 713.49
.....	.....	2 643.27	1 786.40	856.87
.....	401.58	20 147.13	9 017.66	11 129.47
.....	429.38	66 780.25	26 640.15	40 140.10
.....	2 810.45	34 636.75	12 742.94	21 893.81
.....	237.25	25 978.21	10 600.99	15 377.22
.....	.....	25 376.14	10 340.78	15 035.36
.....	3 067.46	69 077.08	24 155.95	44 921.13
.....	.....	387.11	57.34	329.77
.....	.....	3 559.42	1 852.38	1 707.04
.....	.....	19 945.75	8 289.44	11 656.31
.....	.....	460.76	245.82	214.94
.....	.....	1 890.34	754.78	1 135.56
gs.....	.....	13 900.28	6 527.25	7 373.03
.....	.....	54 749.57	30 463.25	24 286.32
.....	$8 138.45			
r construction, etc..		$683 654.48		
f to date.....			$304 522.21	
oks, Sept 30, 1912.....				$379 132.27
o profit and loss.....		$ 13 433.93	$ 1 629.85	
l depreciation since		$697 088.41	$306 152.06	

: 30, 1912, 1 218 300 tons; total development, 54 137 ft; ore possible to mine above tunnel, 1 630 000 tons. Aver rs, 3 ft. One ton in solid equals 13 cu ft.

Oro Mining & Railway Co, El Oro, Mexico

6. Summary of Expenditures Per Ton

	1911	1912	1913
.....	$1.55	$1.89	$2.09
ined.....	0.74	0.83	1.42
er ton treated.....	0.17	0.18	0.18
" " "	0.91	0.93	0.78
" " "	0.02	0.01	0.02
" " "	0.22	0.02	0.20
" " "	0.29	0.23	0.21
.....	$3.90	$4.09	$4.90
n treated.....	0.05	0.062	0.042
" "	0.17	0.197	0.024
" "	0.02	0.026	0.024
" "	0.04	0.035	0.031
.....	.....	.....	0.025
.....	$4.18	$4.41	$5.046
n treated.....	0.27	0.30	0.326
"	0.07	0.22	0.095
"	$0.34	$0.52	$0.421
.....	3.84	3.89	4.625
er ton treated.....	0.34	0.31	0.28
ined and treated.....	4.18	4.20	4.90
eated.....	4.18	3.80	3.42
milled.....	360 294	302 698	253 434
.....	none	84 459	180 274
n from which ore was	.....	.....	650 ft

STANFORD LIBRARIES

Example 7. El Oro Mining & Railway (

Table 47. Data on Mill and Cyanide

Year	1911	
Mill ran, days	349	
Tons per stamp per 24 hr	10.31	
Tons treated:		
Ore crushed, tons	360 294	302
Tailings retreated	.....	84
Total tons treated	360 294	387
Assay value of ore treated:		
Gold	$5.99	
Silver	1.64	
Assay value of tailings retreated:		
Gold	.....	
Silver	.....	
Average assay value per ton treated:		
Gold	$5 99	
Silver	1.64	
Theoretical extraction:		
Gold	91.47%	
Silver	73.29	
Total	87.58	
Actual recovery assay val in bullion:		
Gold	90.77	
Silver	73.23	
Total	87.00	

Example 8. Oriental Consolidated Mining
Korea, Asia

Table 48. Summary of Operations, Year Ende

Summary of costs per ton ore		Summary o
Mining (Table 49)	$1.62	Bullion from m
Milling (Table 50)	0.50	Concentrates..
Concentrate treatment (Table 51)	0.20	Store profit....
Transporting ore	0.01	Interest and ot
General expenses (Table 52)	0.36	Profit on tribu
Total operating expense	$2.69	Total receipts.
Development outside mines	0.05	
Construction	0.09	
Total expenditure	$2.83	

Summary of production		
Tons of company ore mined and milled	321 343	Value per ton se gamation....
Tons of tribute ore milled	2 360	Gross value con
Total tons ore milled	323 703	Net yield conc'
Avg gross value company ore per ton	$5.87	Net saving fron
Total yield of ore per ton treated	4.77	
Value of mill tailings per ton ore	0.59	
Value conc't tailings per ton ore	0.51	

Oriental Consolidated Mining Co (*Continued*)

Table 49. Mining Costs

	Tons	Per ton
.....	121 420	$1.30
.....	98 199	1.54
h mine.....	11 108	0.75
n mine.....	13 847	0.57
al mine.....	726	0.23
.....	27 465	1.97
.....	38 481	2.59
e.....	10 097	4.07
l mines (Table 54)	321 343	$1.62

Table 50. Milling Costs

	Tons	Per ton
.....	121 420	$0.45
.....	126 755	0.46
.....	26 950	0.54
.....	38 481	0.73
.....	10 097	0.88
l mills (Table 53).....	323 703	$0.51
d.....	2 360	0.75
illed.....	321 343	$0.50

Table 51. Concentrate Treatment

	Tons conc't treated	Cost per ton conc't	Cost per ton ore
.....	11 732	$1.88	$0.18
.....	12 823	1.29	0.13
.....	2 234	1.77	0.15
.....	10 097	0.88	0.88
.....	1 015	13.68	0.36
.....	37 901	1.73	0.20

2. General Expense Items (per ton of ore)

ncies....	$0.02885	Extraordinary.....	$0.00036
e.....	0.00320	Road repairs.....	0.00192
nse.....	0.02565	Annual tax, Korea.....	0.03890
urse'ts).	0.00396	Exchange cost.....	0.01553
oard at		Insurance.....	0.02137
.....	0.07065	Legal.....	0.00042
pense...	0.01082	Gen manager's house and	
nd from		grounds.....	0.00017
.....	0.02041	Christmas gifts to white em-	
.....	0.02014	ployees.....	0.03112
.....	0.00764	Telephone repairs.....	0.00317
.....	0.01780	Unclassified, general.....	0.00028
.....	0.00115	House repairs and maint'ce..	0.00437
.....	0.00122	Cordwood indemnity.....	0.01309
.....	0.00276	Chorrie dam land damage...	0.00026
.....	0.00130		
.....	0.00628	Total general expense.....	$0.35402
.....	0.00123		

Example 10. Transvaal Gold M

Table 56. Costs Per Ton Milled (3 Mos

Name of mine	Aurora West United Mine	Cinderella Consolidated Mine	Meyer and Charlton Mine	Ne Go Go Mi
Mining	$2.75	$3.87	$2.09	$2.
Sorting, crushing, and transporting	0.14	0.09	0.09	0.
Milling	0.42	0.46	0.44	0.
Cyaniding	0.45	0.41	0.54	0.
General charges	0.42	0.67	0.60	0.
Office expense	0.09	0.18	0.45	0.
Mine dev't redemption	0.73	0.73	0.23	0.
Permanent works	.....	.....	.....	0.
Total working costs	$5.00	$6.41	$4.44	$3.
Working profit	1.60	.....	2.07	1.
Revenue per ton milled	$6.60	$6.40	$6.51	$5.
Expended on capital account:				
Permanent works	$0.141	$1.17	$0.17	...
Excess development	0.065	0.76	.....	...
Machinery and plant	0.022	0.79	0.054	$0.
Furniture	0.004	.....	0.005	...
Buildings	.....	0.01	.....	0.
Live stock, vehicles	.....	.....	0.004	...
Total capital expended	$0.232	$2.73	$0.233	$0.
Less credits	0.025	.....	.....	...
Total expended per ton	$5.207	$9.14	$4.673	$3.
Tons ore mined	52 079	64 458	44 214	97
Tons waste sorted out	9 980	9 328	2 480	16
% discarded as waste	19.1	14.3	5.4	17
Tons ore crushed	41 699	55 170	42 014	81
Days running time	84.7	84.28	84.20	81.
Tons per stamp per day	6.15	8.01	6.65	8.
Stamps operated	80	80 to 85	75	
Tube mills operated	.....	3	2	
Feet of development	3 394	3 323	1 245	1
Earnings included in revenue stated above	1.49¢	7.3¢	.....	1.
Approximate stoping width, inches	41	40	46	

Note. See Table 57 for general data on Aurora We Charlton.

STANFORD LIBRARIES

Transvaal Gold Mining Costs (*Continued*)

from the Aurora West United & Meyer & Charlton Mines

d Oct, 1913	Aurora West United Gold Mining Co	Meyer & Charlton Gold Mining Co
ode:		
.....	42 in	25 in
.....	14 in	.....
n annually.....	216 000 tons	179 731 tons
stoping areas.....	180 000 milling tons	92.11%
ken ore is moved to te 13).....	200 ft	115 ft
tance to shaft.....	750 ft	700 ft
h.....	1 079 ft	1 430 ft
f ore extracted.....	cost 0.64¢	10. 12 ft poles 10. 6 ft lagging
rformed during 1912:		
.....	251.5 ft	473 ft
.....	8 346 ft	2 713 ft
.....	4 398.5 ft	1 674 ft
able by development	194 605 milling tons	205 133
.....	.....	379 ft
d from mine in 24 hr	150 000 gal	65 000 gal
er is pumped.....	1 000 ft	1 650 ft
ired:		
.....	70 h p	100 h p
gs.....	(See Note 14)	140 "
.....	"	560 "
.....	hand	hand
.....	150 h p	425 h p
very to mill.....	35 to 40 h p	55 "
.....	5 h p	10 "
.....	300 "	280 "
.....	.....	80 "
.....	100 h p	150 "
.....	40 "	50 "
.....	140	128
.....	1 345	920

pes run on an incline of about 30°; backs average 200 ft, irectly below.

eater portion of stoping is done by manual labor, native ham- It requires 750 h p to operate compressors used in driving nt faces and in stopes where machines are worked.

cost data for these mines.

Example 10. Transvaal Gold Mining C

Table 58. Summary of Operating Re

Name of mine	Modder-fontein B Gold Mines	M
Tons of ore crushed	388 570	5
Gold revenue per ton milled	$9.07	
Total working costs (see *A*)	4.29	
Sundry expenses	0.25	
Construction, etc, chg'd to capt acc't	0.28	
Total Expenditure per Ton Milled	4.82	
Profit on ore per ton	4.25	
Miscellaneous profits	0.07	
Total profit over expenditure	4.32	
Government income tax on profits	48.6¢	
(*A*) Items of Working Cost per Ton		
Mining (see *B*)	$2.25	
Development charged	0.44	
Milling expenses (see *C*)	1.03	
General expenses	0.44	
Renewals and replacements	0.13	
Total Working Cost per Ton Milled	$4.29	
(*B*) Items of Mining Costs per Ton Mined		
Total tons ore mined	437 306	6
Tons ore from stopes	406 178	6
Tons ore from development	31 128	
Average stoping width in inches	55	
Stoping	$0.71	
Timbering and rockwalling	0.24	
Old workings	.....	
Shoveling and tramming	0.71	
General and other costs	0.34	
Sand filling	.....	
Total Cost per Ton Mined	$2.00	
(*C*) Items of Milling Cost per Ton Milled an		
Tons of ore received from mine	437 306	6
Per cent waste sorted out of mine ore	11.1%	
Average value of waste sorted out	49¢	
Tons of ore stamped	388 570	5
Stamps in operation	80	
Tons per stamp per 24 hr	14	
Tons of pulp cyanided	387 487	5
Per cent of total yield by mill	57.5%	
Per cent of total yield by cyaniding	42.5%	
Ore sorting and crushing	$0.08	
Transporting from crusher to mill	0.02	
Stamp milling	0.20	
Tube milling	0.18	
Amalgamation	0.04	
Cyaniding	0.51	
Total per Ton Milled	$1.03	

STANFORD LIBRARIES

Transvaal Gold Mining Costs *(Continued)*

ummary of Operating Results, 1912 *(Concluded)*

Village Deep Ltd	Ferreira Deep Ltd	Crown Mines Ltd	Bantjes Cons Mines Ltd	Durban Roodeport Deep
96 900	559 800	1 920 700	286 453	293 995
$7.25	$9.70	$7.78	$7.17	$7.27
4.84	5.16	4.45	5.85	5.91
0.05	0.07	0.13	0.10	0.07
0.34	0.02	0.53	0.14	0.25
5.23	5.25	5.11	6.09	6.23
2.02	4.45	2.67	1.08	1.04
0.04	0.18	0.02	0.06	.....
2.06	4.63	2.69	1.14	1.04
20.5¢	41¢	32¢	11.7¢	11.5¢
Items of Working Cost per Ton Milled				
$2.79	$3.22	$2.94	$3.16	$3.79
0.79	0.30	0.28	1.09	0.60
0.89	1.15	0.91	1.09	1.03
0.37	0.39	0.32	0.51	0.49
.....	0.10	.....	.....	.....
$4.84	$5.16	$4.45	$5.85	$5.91
of Mining Cost per Ton Mined and Other Data				
98 124	663 084	2 183 305	327 710	357 270
15 415	626 044	2 081 222	310 366	319 678
52 709	37 040	102 083	17 344	37 592
60	65	64	41	46
$1.05	$0.77	$1.01	$1.58	$0.95
0.20	0.14	0.14	0.08	0.40
.....	0.59	0.10	0.02	0.04
0.45	0.32	0.57	0.51	0.49
0.69	0.77	0.73	0.57	1.24
.....	0.12	0.04	.....	.....
$2.39	$2.71	$2.59	$2.76	$3.12
Milling Cost per Ton Milled and Other Data				
8 124	663 084	2 183 305	327 710	357 270
14.8%	15.7%	11.9%	12.6%	17.7%
26¢	65¢	57¢	63¢	49¢
6 900	559 800	1 920 700	286 453	293 995
180	220	660	80	100
9.5	8.0	10.3	11.0	8.6
6 860	?	1 915 716	287 270	293 508
70.4%	68.5%	70.1%	60%	70.5%
29.6%	31.5%	29.9%	40%	29.5%
$0.12	$0.14	$0.10	$0.14	$0.18
0.02	0.04	0.04	0.02	0.02
0.22	0.32	0.31	0.22	
0.14	0.18	0.12	0.14	0.46
0.08	.....	.....	0.10	
0.31	0.47	0.34	0.47	0.37
$0.89	$1.15	$0.91	$1.09	$1.03

STANFORD LIBRARIES

Example 11. Tonopah-Belmont Development Co, Tonopah, Nevada

(A Silver-Gold Mine)

Table 59. Comparative Statement of Assets and Liabilities

	Amount, Feb 28, 1909	Increase or *decrease in account at				Total Feb 28, 1913
		Feb 28, 1910	Feb 28, 1911	Feb 28, 1912	Feb 28, 1913	
Assets:						
Property account	$2 556 521	$ 7 015	$ 144 331	$ 81 952	$156 979	$2 943 889
Investments	90 000	*6 500	*58 200	*300	.....	25 000
Development	50 000	.....	*50 000	.....	.....	.....
Charges paid in advance	2 208	*229	771	114	*794	2 070
Stores at mill	31 703	4 759	17 272	*7 798	*17 481	28 456
Stores at mine	6 136	2 198	24 309	29 534	73 920	136 099
Ore stocks at mill	35 099	*35 099	41 444	3 934	73 502	118 881
Accounts receivable	96 997	*42 002	97 607	64 342	73 016	289 962
Cash in banks	52 979	145 318	461 545	266 704	*277 381	649 166
Trustee stock	500 000	.....	.....	*500 000	.....	.....
Collateral loans	.....	.....	350 000	*200 000	100 000	250 000
Totals	$3 421 643	$75 551	$1 026 081	$261 515	$181 761	$4 443 525
Liabilities:						

Tonopah-Belmont Development Co (*Continued*)

Analysis of Accounts in Table 59 and Summary of Production for Period

ssets		Current liabilities		
909	1913		1909	1913
000 00	$ 25 000 00	Acc'ts payable	$ 6 560 79	$109 873 70
840 14	164 555 31	Unpaid vouchers	36 372 91	.....
099 48	118 881 45	Unpaid wages	19 760 55	36 048 10
997 61	289 962 13	Accrued ch'gs	4 797 21	13 294 49
979 77	649 166 35			
208 68	2 070 19			
125 98	$1 249 635 43	Total	$67 491 46	$159 216 29

ver current liability, Feb 28, 1909	$ 217 641 32
" " " " 1913	1 090 419 14
ts during 4-year period	$ 812 784 92
8, 1910, to Feb 28, 1911	450 000 00
1911 " " 1912	1 500 000 00
1912 " " 1913	1 500 000 00
all expenditures for 4-year period	$4 262 784 92

Receipts for 4-year period

Sale of mine product	Miscellaneous earnings	Dividends on investments	Total income for year
$ 795 873 59	$ 80 79	$15 300 00	$ 811 254 38
2 597 278 96	16 170 62	4 750 00	2 617 199 58
3 268 514 81	25 605 51	11 550 00	3 305 669 32
3 160 751 38	22 113 92	17 500 00	3 200 365 30
$9 822 417 74	$64 270 84	$48 100 00	$9 934 788 58

vidends and increase in current assets as	$4 262 784 92
wn by receipts and profits	$5 672 003 66

ed during period, 363 696 tons.
ton, including construction and all selling charges, $15 51
rill, about $14 20 per ton).
as receipts from sale of mine product include all selling
ons.

STANFORD LIBRARIES

Example 11. Tonopah-Belmont Develop

Table 61. Total Operating Cost of Mine Year Ended Feb 28, 191

Production and treatment losses	Dry to ore
Mine production:	
Ore milled at Millers	43 03
Ore milled at Tonopah	84 88
Ore shipped to smelter	56
Ore stock additions	1 05
Total Mine Production	129 53
Losses in treatment:	
Mill tailings at Millers	43 03
Mill tailings at Tonopah	84 88
Conc't and slags at Millers	43 03
Conc't and slags at Tonopah	84 88
Smelting losses in shipment	56
Total Loss in Treatment	129 53
Value Realized	129 53
Costs	**Dry to ore**
Mining costs (Table 62):	
Stoping	118 21
Development	11 32
Transportation:	
Freight to Millers mill	43 03
" to smelter	56
" on ore stock treat	1 05
Treatment costs (Table 63):	
Milling at Millers mill	43 03
" at Tonopah mill	84 89
Marketing bullion, Millers	43 03
" " Tonopah	84 88
" conc't, Millers	43 03
" " Tonopah	84 88
" smelting ore	56
General Expense: Administration, taxes, etc.	129 53
Total Operating Expenses	129 53
Realization from Operations	129 53

Tonopah-Belmont Development Co *(Continued)*

. Comparative Statement of Mining Costs

	Year ended Feb 28, 1914		Year ended Feb 28, 1913	
	Amount	Per ton	Amount	Per ton
	$ 54 749 38	$0 474	$47 781 02	$0 369
ers	25 663 01	0 222	26 348 68	0 203
pers	9 737 77	0 084	10 356 14	0 080
	53 450 69	0 445	73 991 51	0 572
	39 175 17	0 339	46 950 18	0 362
	22 196 61	0 192	27 036 36	0 209
pers	112 991 39	0 978	88 019 80	0 679
	4 895 52	0 042	4 218 90	0 033
, and mains	5 804 69	0 050	5 425 95	0 042
s, and maint	3 489 78	0 029	6 580 30	0 051
g	8 158 35	0 071	8 502 99	0 065
	33 099 35	0 286	42 623 43	0 329
	35 711 59	0 309	36 432 18	0 281
	10 847 99	0 094	4 745 14	0 037
ding	31 592 00	0 273	13 132 15	0 101
ing	5 457 08	0 047	6 270 78	0 048
	0 481 87	0 050	4 677 81	0 036
houses	13 801 66	0 137	16 106 77	0 124
	14 926 50	0 129	15 530 68	0 120
	18 731 47	0 162	16 504 58	0 128
	5 414 63	0 049	5 152 73	0 040
	4 609 16	0 041	7 784 14	0 060
	0 039 97	0 001	3 652 77	0 028
gears, buildings	2 969 62	0 025	792 08	0 006
nach's and tools	2 624 27	0 024	3 878 39	0 030
ge lines, tanks	1 962 15	0 016	4 184 58	0 032
ailroad spur	947 53	0 008	117 63	0 001
ole line	241 99	0 002	13 56	0 000
	7 008 39	0 062	19 102 44	0 147
	1 995 32	0 017	3 879 10	0 030
Mining	$543 317 62	$4 720	$349 695 54	$1 243
	7 867 30	0 068	8 079 41	0 062
vision	10 436 42	0 089	12 809 53	0 096
	4 158 12	0 039	2 999 60	0 023
	40 954 68	0 351	10 487 06	0 081
	15 028 41	0 130	15 028 41	0 116
	4 487 01	0 039		
Mining	$ 83 132 15	$0 719	$ 48 984 05	$0 378
Mine	$628 539 77	$5 439	$398 677 58	$1 621
	115 563		129 537	
	11 005 ft		12 314 ft	
	186 ft		193 ft	
ation	1 845		1 210	

Example 11. Tonopah-Belmont Developm

Table 63. Milling Costs, both Mills, Year E

Items	Millers Mill	
	Amount	Per t
Crushing and conveying	$ 6 719.06	$0.1
Stamping	32 630.40	0.5
Tube milling	6 679.53	0.1
Concentrating	6 679.53	0.1
Classifying	.....	.....
Elevating and separating	2 368.92	0.0
Thickening	.....	.....
Leaching sands	13 899.29	0.2
Conveying and discharging sands	4 032.54	0.0
Agitating slimes	70 667.89	1.1
Piltering and discharging slimes	13 168.46	0.2
Precipitation	13 713.71	0.2
Assaying	4 603.36	0.0
Refining	8 972.67	0.1
Lighting	1 146.87	0.0
Shift bosses	5 424.20	0.0
Watchmen	1 460.00	0.0
Surface and plant	2 506.72	0.0
Sand dump	1 702.92	0.0
Other expense	7 567.25	0.1
Total Direct Cost of Milling	$203 943.32	$3.3
Administration	$ 7 006.25	$0.1
Office and store house	2 834.18	0.0
Taxes and insurance	7 108.95	0.1
Depreciation	30 000.00	0.4
General charges	375.85	0.0
Total Indirect Milling Cost	$ 47 325.23	$0.7
Total Milling Cost	$251 268.55	$4.1
Dry tons of ore milled	60 359	.....
Labor per dry ton	.....	$0.9
Miscellaneous supplies, dry ton	.....	2.5
Power, per dry ton	.....	0.7
Total Milling Cost	.....	$4.1
Average number stamps operated	36	.....
Average tons ore per stamp, 24 hr	4.2	.....

STANFORD LIBRARIES

2. Kerr Lake Mining Co, Cobalt, Canada

mmary of Operations for Year Ended Aug 31, 1913

er. 2 160 878 oz lter, 1 977 870 oz 8.5% not included in costs given	Total expenditure for period	Per ton ore hoisted
nent:		
	$ 18 710.10	$0.435
	29 036.81	0.674
heat	27 569.25	0.639
ging	15 346.55	0.356
	28 963.53	0.673
	8 060.02	0.187
	17 011.79	0.396
	760.33	0.017
	9 316.17	0.216
	11 217.22	0.260
d buildings	5 950.31	0.138
	5 206.12	0.121
	4 165.06	0.096
enance	11 130.29	0.254
	989.55	0.023
	24 780.00	0.574
	779.93	0.018
Development	$218 993.03	$5.077
, etc:		
	1 349.24	0.031
	2 643.49	0.061
	9 323.65	0.216
nse	92 632.61	2.147
	4 891.22	0.113
	3 608.95	0.084
l general	11 736.23	0.272
	$345 178.42	$8.00

Detail of ore hoisted

	379 tons	Total ore	33 738 tons
s	161 "	Mine waste	9 396 "
t	192 "		
	16 625 "	Total rock hoisted	43 134 "
tables	16 381 "		

. Daly-Judge Mining Co, Park City, Utah

e 65. General Statement of 1911 Costs

Items	Tons ore	Cost per ton
	59 461	$3.09
	51 875	0.79
	59 461	0.25
	22 606	1.03
	59 461	2.38
		0.08
		0.04
		0.11
ton mined	59 461	$7.77

n for freights, smelting, etc, $10.80 (Table 66)

STANFORD LIBRARIES

Example 13. Daly-Judge Mining C

Table 66. Content of Ore per

	Tons	Oz Ag	Oz Au	% Pb
Crude	7 586	32.44	0.077	23.56
Concentrates	12 237	23.37	0.036	24.74
Zinc middlings	2 783	10.19	0.020	6.84

Total metal contents of product (

Oz Ag	Oz Au	Lb lead	Lb zinc	Lb
560 699	1 080.14	10 027 070	7 431 176	311 8

Approximate value of metal content, at average N Y 19

Approximate deduction for freights, smelting charge
smelting and selling charges
Approximate deduction per ton of ore mined, $10.80; p

Example 14. Daly-West Mining Co,

Table 67. Result of Operations, 1900 to

Year	Lb copper	Tons Pb	Oz silver	Oz gold
1900	1 193 839	6 587	1 388 755	1 310.69
1901	1 326 448	7 793	1 755 454	1 606.72
1902	2 403 670	14 903	3 575 796	2 943.96
1903	3 160 824	17 372	4 382 222	3 195.28
1904	2 023 005	12 826	3 064 224	2 376.36
1905	1 225 731	8 386	1 798 628	1 396.85
1906	830 307	6 075	1 200 857	1 016.62
1907	673 890	4 043	1 042 062	949.91
1908	459 722	2 494	665 625	592.48
1909	521 273	4 820	762 730	644.48
1910	574 506	7 414	887 525	771.41
1911	877 414	8 881	1 197 796	1 206.61
1912	711 694	5 401	1 036 427	828.44
Total	15 982 323	106 995	22 758 101	18 839.81

Gain in cash during period
Total balance over expenditures
Total expenditure at mine and mill
Total value of ore, after deducting freights, smelting,

Tons of ore shipped
Tons of sorted ore milled
Total tons of ore

Average value of ore per ton, after deducting freights, sm
Total net expenditure at mine and mill per ton ore
Miscellaneous earnings credited to costs, per ton ore
Total expenditure, not deducting miscellaneous credits
Smelting, freight and selling charges, and smelting los
probably about the same as for the Daly-Judge (Tabl
(See Table 68 for details of costs.)

14. Daly-West Mining Co (*Concluded*)

Details of Mining and Milling Costs per Ton

	1910	1911	1912
ted ore:			
.....	$0.71	$0.25	$0.78
.....	0.76	0.69	1.26
.....	1.07	1.05	1.36
.....	0.55	0.49	0.62
.....	0.21	0.30	0.34
.....	0.08	0.04	0.05
.....	0.05	.....	0.02
.....	0.13	0.21	0.44
ing.....	0.01	.....	0.11
.....	0.06	0.08	0.30
.....	.....	.....	0.12
.....	0.07	0.04	0.08
.....	$3.70	$3.15	$5.48
ted ore:			
.....	$0.03	$0.02	$0.03
.....	0.36	0.30	0.20
ing.....	0.53	0.63	0.34
es.....	0.12	0.10	0.08
.....	0.14	0.07	0.03
.....	0.09	0.07	0.05
.....	0.09	0.08	0.06
.....	0.05	0.05	0.09
.....	0.21	0.31	0.21
.....	0.31	0.26	0.15
.....	0.10	0.10	0.20
.....	$2.03	$1.99	$1.44
d ore:			
.....	$0.18	$0.18	$0.19
.....	0.08	0.12	0.23
.....	0.03	0.03	0.05
.....	0.03	0.02	0.03
.....	0.01	0.01	0.02
.....	0.38	0.62	.....
.....	.....	0.16	0.36
.....	$0.71	$1.14	$0.88
Milling, and General, ...	$6.50	$6.15	$7.55
.....	4 363	7 083	17 497
.....	85 465	81 058	44 260
ste.....	2 527	1 137	1 369
d.....	87 482	87 004	60 151

Example 15. Bunker Hill & Sullivan Mining
Kellogg, Idaho

Table 69. Result of Operatio

Year ended May 31	1909	1910
Receipts:		
Gross value shipping ore and conc't produced	$3 199 976	$3 307 826
Boarding house	1 583	1 081
Rents	4 392	5 573
Sale of elec power and light, net	9 597	11 157
Sale of water, net	3 822	4 213
Interest, etc	675	2 283
Through head offices	32 370	24 421
Total receipts	$3 252 415	$3 356 554
Expenditures:		
Operating (Table 70)	$ 761 374	$ 788 977
Frt, treat, and disc't on shipments (Table 71)	1 343 043	1 318 297
Improvements (Table 73)	165 264	147 150
Taxes	52 835	45 997
Insurance	6 601	5 347
Real estate	19 157	4 242
Mining property purchased	7 027	12 102
Mine office fixtures	1 685	1 267
Litigation	109 743	188 254
Exploration (Table 72)	44 152	42 596
Head offices	5 082	4 977
Tailings dump land	.....	.....
Total Expenditures	$2 515 963	$2 559 206
Balance of receipts over expenditures	$ 736 452	$ 797 348
Production and data:		
Tons con'ctg ore produced	341 700	376 200
Tons shipping ore produced	2 770	1 330
Total tons ore produced	344 470	377 530
Tons concentrate produced	67 710	63 530
Ratio of ore to concentrate	5.05 to 1	5.89 to 1
Assay of ship'g ore and conc't:		
Lead	46.13%	50.90%
Silver	16.35 oz	18.54 oz
Average price of metal:		
Lead	3.947¢	4.064¢
Silver	51.514¢	51.9¢
Gross assay value shipping product per ton	$44.89	$51.00
Expense per ton ore mined	7.30	6.78
Sundry credits (as above) per ton mined	0.15	0.13
Net cost per ton ore mined	7.15	6.65

STANFORD LIBRARIES

nker Hill & Sullivan Min & Conc't'g Co (*Continued*)

Operating Costs per Ton Mined at Wardner

led May 31	1909	1910	1911	19 mos to Dec 31, '12
.....	$1.563	$1.450	$1.423	$1.602
5).....	0.083	0.069	0.067	0.074
les 76 and 77).....	0.382	0.408	0.400	0.374
.....	0.047	0.046	0.038	0.034
nse (Table 79).....	0.086	0.078	0.073	0.073
(Table 79).....	0.041	0.031	0.048	0.086
.....	0.008	0.008	0.007	0.007
rges per Ton Ore.....	$2.210	$2.090	$2.056	$2.250

Freight, Smelter, and Discount Charges, for 19 Mos ended Dec 31, 1912

	Tons shipped	Assays				Per ton shipped	
		Lead	Silver	Iron	Silica	Frt chgs	Discounts
s....	65 941	57.81	20.52	2.7	5.1	$15.56	$4.98
.....	1 323	34.79	13.99	....	....	13.08	2.81
.....	28 112	47.12	19.05	....	....	14.35	3.82
.....	891	15.87	5.38	....	....	1.75	5.23
.....	3 430	49.87	16.94	....	....	14.61	3.99

72. Development Cost per Foot and Ton

led May 31	1909	1910	1911	19 mos to Dec 31, '12
hs, and machinists....	$0.497	$0.522	$0.394	$0.312
penters.....	0.258	0.171	0.015	0.036
.....	1.437	1.485	1.598	2.499
.....	0.009	0.004	0.002	0.004
.....	1.793	2.417	2.042	1.649
.....	1.715	0.088	0.762	0.013
.....	0.403	0.538	0.317	0.176
.....	0.131	0.128	0.109	0.089
.....	1.431	1.300	1.495	0.987
.....	0.103	0.103	0.097	0.085
.....	0.058	0.051	0.043	0.021
.....	0.044	0.072	0.054	0.043
es.....	0.395	0.499	0.432	0.322
.....	0.493	0.124	0.354	0.403
.....	0.367	0.549	0.467	0.348
.....	0.373	0.556	0.272	0.161
.....	.....	.....	.....	0.002
per Foot.....	$9.507	$8.607	$8.453	$7.150
rment.....	4 644	4 949	5 928	11 050
st per ft.....	$12.50	$10.22	$10.86	$10.09
t per ft.....	7.39	7.58	7.49	5.32
per ton ore.....	0.128	0.113	0.114	0.113

Example 15. Bunker Hill & Sullivan Min & C

Table 73. Expenditure on Improvements

Year ended May 31	1909	1
Mine surface	$0.055	$0
Concentrator building	0.011	0
Tailings plant	0.004	0
Wardner News and printing plant	0.001	0
Power plant	0.081	0
Office building	0.016	0
Cheyne electric plant	0.003	. .
Wardner water system	0.004	0
Mill No 1	0.303	0
Mill No 2	.....	0
Hospital building	0.002	. .
Y M C A building	.....	0
Mine boarding house	.....	0
Total Cost of Improvements per Ton Ore	$0.480	$0

Table 74. Cost of Stoping per Ton

Year ended May 31	1909		1910		
	Conc't ore	Ship ore	Conc't ore	Ship ore	Co c
Foremen, blacksmiths, and machinists	$0.185	$0.183	$0.178	$0.267	$0.
Timbermen, carpenters	0.082	0.052	0.071	0.079	0
Miners	0.343	0.271	0.362	0.298	0
Carmen	0.067	0.069	0.051	0.102	0
Shovelers	0.386	0.951	0.327	0.639	0
Power labor	0.027	0.025	0.024	0.030	0
Repair labor	0.026	0.025	0.025	0.023	0
Explosives	0.094	0.081	0.082	0.077	0
Illuminants	0.022	0.020	0.021	0.020	0
Lubricants	0.005	0.004	0.003	0.003	0
Iron and steel	0.008	0.008	0.014	0.013	0
Miscellaneous supplies	0.035	0.028	0.033	0.031	0
Timber and lagging	0.192	0.141	0.200	0.230	0
Power supplies	0.030	0.022	0.038	0.038	0
Wood	0.048	0.040	0.033	0.034	0
Stable and stock	0.005	....	0.004	....	0
Contractors and lessees	.....	....	.....	0.984	.
Total Cost per Ton	$1.555	$1.920	$1.466	$2.868	$1
Highest monthly cost	$1.750	$2.250	$1.646	$7.28	$1
Lowest monthly cost	1.312	1.706	1.354	1.697	1

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nker Hill & Sullivan Min & Conc't'g Co (*Continued*)

Tramming Costs per Ton Ore by Electric Power
(Including hauling timber and supplies)

d May 31	1909	1910	1911	19 mos to Dec 31, '12
.....	$0.004	$0.003	$0.003	$0.003
.....	0.030	0.026	0.022	0.023
.....	0.005	0.006	0.004	0.005
.....	0.014	0.014	0.019	0.023
.....	0.001	0.001	0.001	0.001
.....	0.031	0.001	0.001	0.001
.....	0.003	0.003	0.002	0.002
s.....	0.002	0.005	0.009	0.008
.....	0.016	0.011	0.008	0.008
.....	0.007	.....	.....	.....
t per Ton of Ore.....	$0.083	$0.070	$0.069	$0.074
ncluding waste.....	$0.077	.....	.....	.....
ctual working time..	530	535	620	621

76. Concentrating Costs Per Ton of Ore

.....	$0.040	$0.037	$0.034	$0.035
.....	0.136	0.145	0.137	0.121
.....	0.009	0.009	0.003	0.007
.....	0.008	0.007	0.006	0.004
.....	0.005	0.005	0.003	0.003
.....	0.013	0.017	0.018	0.019
.....	0.004	0.003	0.003	0.002
.....	0.005	0.004	0.005	0.004
.....	0.002	0.003	0.002	0.001
s.....	0.074	0.062	0.050	0.053
.....	0.005	0.007	0.006	0.006
.....	0.004	0.004	0.004	0.004
.....	0 005	0.005	0.006	0.006
.....	0.009	0.010	0.010	0.004
.....	0.043	0.069	0.074	0.081
.....	0.024	0.018	0.008	0.009
es.....	.....	.....	0.002	.....
Cost per Ton of Ore..	$0.386	$0.405	$0.371	$0.359
24 hr actual time....	1 018	1 095	1 270	1 256
ed.....	341 700	374 300	436 240	697 560

77. Cost Tailings Treatment per Ton

.....	.....	$0.073	$0.075	$0.076
.....	.....	0.031	0.001	0.002
.....	.....	0.001	.....	.....
.....	.....	0.001	0.001	0.001
.....	.....	0.020	0.015	0.020
.....	.....	0.004	0.002	0.001
.....	.....	0.027	0.026	0.031
.....	.....	0.008	0.017	0.015
.....	.....	.....	0.001	0.001
.....	.....	.....	.....	0.001
reated.....	.....	$0.165	$0.138	$0.148
treated.....	.....	15 500	101 300	78 100

Example 15. Bunker Hill & Sullivan Min &

Table 78. Shipping Expense per To

Year ended May 31	1909	1910
General labor	$0.015	$0.017
Laborers	0.058	0.050
Contractors	0.136	0.150
Illuminants	0.001	0.002
Miscellaneous	0.004	0.005
Lumber	.....	0.006
Representatives at smelter	0.013	0.033
Wood	.....	.....
Total Cost per Ton Shipped	$0.227	$0.263
Tons of concentrates shipped	68 008	63 686
Tons of ore shipped	3 514	326
Total tons shipped	71 522	64 012

Table 79. Superintendence, Office, and Co

Year ended May 31	1909	1910
Superintendence and office:		
Supt and surveying	$18 200.00	$18 300.0
Bookkeeping and assaying	7 145.92	6 415.3
Wood and illuminants	770.00	1 094.5
Office supplies	1 370.03	1 373.5
Assay supplies	863.00	1 086.4
Telegrams and telephones	889.37	821.4
Sundries	208.24	345.3
Total Supt and Office	$29 446.56	$29 436.5
Contingent expense:		
Secret service	$ 240.99	.....
Subn Wardner Indus Union	1 770.00	$ 1 500.5
Mine Owners' Ass'n dues	4 563.10	5 747.4
Perfecting titles	1 295.90	1 051.1
Fire protection	85.50	252.3
Electric power expert	333.00	.....
Politics	456.33	.....
Donations	686.65	922.6
Mine accidents, etc	4 750.70	2 670.6
Ore contract expense	.....	.....
Y M C A expense	.....	.....
Rescue apparatus	.....	.....
Smallpox epidemic	.....	.....
Mine fire expense	.....	.....
Special statistics	.....	.....
Special investigations	.....	.....
Sundries	.....	.....
Total Contingent Expense	$14 182.17	$12 144.5
Sundry Credits	.....	392.9
Net Contingent Expense	$14 182.17	$11 751.5

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16. Lead Mining, Southeastern Missouri

General Summary of Costs per Ton of Ore Milled

	Bonne Terre and Leadwood	Flat River
d drilling)	$0.04	$0.04
ining	1.00	0.81
lls	0.03	0.09
	0.36	0.30
milling expenses	0.04	0.07
ates to smelter or selling point	0.05	0.03
	0.49	0.51
	0.03	0.03
cluding administration, etc.	0.27	0.20
	0.04	0.04
n (estimated)	0.10	0.10
of Ore Milled	$2.45	$2.22
ntrates	16 to 1	15.4 to 1
ncentrates	63% lead	66.6% lead
d per ton of concentrates	1 197 lb	1 265 lb
d per ton of ore milled	74.75 lb	82.2 lb
lants in tons of ore milled	800 000	900 000
or miscellaneous income are figures.)		

amond drilling data for 161 holes in this district, totaling 77 135 ft, 57.93¢ per ft of hole, as follows: operating labor, 31.28¢; power, 4 per kt, 12.2¢; operating materials, 2.82¢; labor (maintenance and (maintenance and repairs), 3.14¢; depreciation, 0.63¢. For single om 50¢ to $1.50 per ft. The rate of drilling is about 25 ft per e of about 50 to 55 ft of holes are drilled per ton of coal used at

es and mills costs approximately 1.16¢ to 1.43¢ per kw-hr. Pur- osts about 1.62¢ per kw-hr. Steam power costs about 1.21¢ per range from 7¢ to 12¢ per ton and underground haulage from 6¢

Shift bosses, $3.10 per shift; drillers, $2.40; backhand drillers, Loaders, drivers, cagers, car hookers, trackmen, stablemen, ncline hoistmen, and pipe fitters, $2.40 per shift. Underground ressor engineers, shaft carpenters, and electricians, $2.10, $2.25 erground blacksmiths, $2.55; helpers, $2.15; surface blacksmiths, underground diamond drillers, $2.60. Hoisting engineers and . Machine shop mechanists and blacksmiths, from $2.10 to m $2.20 to $2.60. Mill shift-bosses, $3.10; mill workers, from

5.5 tons of ore are secured per day per man employed in the 20 to 25 tons are milled per day per man employed in the mills. at mines and mills, including operating heads, the average is a nan per day.

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Example 17. Zinc Mining, Jopl

(Furnished by J. L. Bruce, Mgr, Butte and Supe

The ore deposit of the mine whose costs are covered by "sheet ground." It occurs in a nearly flat vein or blanket, 100 and at a depth of 200 to 225 ft below the surface. Th and covers an area of 100 acres, most of which is compe gradually becoming poorer in valuable minerals in lateral ex grade ore unprofitable to mine and mill.

The valuable minerals are sphalerite and galena, which o interior of lenticular openings or completely filling former more generally as crystals enclosed in secondary flint whic primary chert formation. The gangue consists of one particu chert formation, together with the secondary flint, there bei no other rock material. There is present a small amount o the form of marcasite. That particular stratum of chert occurs is overlaid by 6 or 8 ft of chert of similar character, extending to the surface. Below the ore stratum are about which lie alternating strata of limestone and chert. The str orebody is capable of supporting itself over spans of 30 ft, a 100 ft. The limestone above this chert is even stronger, so approximately 10% of the area is allowed to remain (witho its valuable contents) as pillars from 15 to 20 ft in diameter.

In mining the upper 7 or 8 ft of the orebody the work rock drills, holes from 6 to 8 ft deep. After chambering ou with a small charge of dynamite, they are blasted with 7 to the hole. A round usually consists of 5 holes drilled parallel to which they are to break. Three holes in front are fired holes. After the face between two pillars has been advanc to 16 ft back from the face is made, a 3½-in drill on tripod hole under the bench to about 16 ft in depth. This hole is c 2 to 3 sticks of dynamite the first time and 15 to 30 the seco hole is large enough, from 75 to 125 lb of 40% gelatine powd hole may blast as much as 600 to 700 tons of ore, often loos of 25 to 30 ft. The broken ore is loaded into cars of 2 700-l mules to a central gathering track; whence it is taken to th locomotive and hoisted in automatic dumping cages, which grizzley (with bars set 6 in apart) above the mill bins. Pieces of grizzley are broken with a sledge and passed through if they sufficient quantity, otherwise this oversize is sent to waste d

Practically no development work was ever done on this carried on by churn drilling, the average cost of which was a cost of plant, exclusive of shaft sinking, was $60 000 or abo pacity. This includes cost of mill, power plant, compressor smith shop, etc.

The mill has a capacity of about 600 tons and the ore is into a 10 by 18-in Blake crusher, then to 30 by 16-in rolls, ji description of milling process employed at this property may *and Mining Journal*, 1912, Vol 93, pages 405, 459, 501, and

Power. Steam power is used for hoisting and compres mill-drive and pumping. Steam power has been generated gas, which costs 12.5¢ per 1 000 cu ft, fuel oil at a cost of 1.5¢ $2.25 a ton, delivered at the plant. The cost of steam pow plies, is about 0.8¢ per hp-hr; cost of electric power, about compressing air for rock drills averages about 19 h p for ea shift. Mill and mill pump consume about 165 h p and mine

Labor. On average production of 500 tons per day, mi and milled in 2 shifts, the distribution of labor is approximat

Shovelers	24	Ground foreman	
Machine men and helpers	30	Misc'l underground	
Mule drivers	6	Hoist man	
Locomotive drivers	1	Firemen	
Cagers	4	Blacksmiths	
Pumpmen	2	Carpenters	
Powdermen	2	Cullmen (ore-sorters)	
Roof trimmers	2		

Tons mined and milled per man per shift (total men)	4.7	Tons mined per roc	
Tons handled per shoveler per shift	21	Tons mined per und	
		Tons milled per mill	

17. Zinc Mining, Joplin, Mo (*Continued*)

gives cost of supplies per ton of ore mined. Explosives cost
b for 40% ngl; coal, $2.25 per ton delivered at mine; electricity
hilled iron, about 2.25¢ per lb.

he 3¼-in rock drills working in headings drill an average of
the drillmen doing their own mucking and setting up. Each
5 tons per shift. The 3⅜-in tripod drills average about 15 ft of
os of ore broken per shift.

ming is done by shovelers; all is by mules to a central track and
from central track 450 ft to main shaft. About 500 tons per day
rack by 6 mules and drivers.

le 17. Zinc Mining Costs, Joplin, Mo

1. Summary of Costs, Continental Zinc Co

	1908	1909	1910	1911	1912
.....	234	277	297	299	293
d.....	83 603	108 728	152 163	173 489	167 053
'ts produced....	2 029	3 048	4 330	4 663	4 684
y departments:					
.....	$0.011	$0.013	$0.010	$0.009	$0.010
.....	0.207	0.136	0.064	0.033	0.040
.....	0.359	0.376	0.353	0.323	0.349
ming.....	0.172	0.212	0.196	0.200	0.226
g.....	0.051	0.065	0.049	0.039	0.045
.....	$0.800	$0.802	$0.672	$0.604	$0.670
.....	0.272	0.258	0.224	0.221	0.247
.....	0.058	0.055	0.055	0.050	0.057
re.....	$1.130	$1.115	$0.951	$0.875	$0.974
emized:					
.....	$0.519	$0.542	$0.506	$0.476	$0.5280
.....	0.145	0.145	0.132	0.118	0.1120
.....	0.104	0.095	0.075	0.073	0.0986
.....	0.012	0.007	0.006	0.005	0.0056
.....	0.015	0.014	0.012	0.008	0.0091
.....	0.036	0.032	0.029	0.028	0.0317
.....	0.020	0.017	0.012	0.012	0.0124
olts.....	0.005	0.008	0.006	0.004	0.0059
.....	0.030	0.027	0.026	0.026	0.0294
.....	0.008	0.007	0.005	0.005	0.0043
.....	0.014	0.014	0.012	0.010	0.0120
se.....	0.006	0.007	0.012	0.010	0.0117
ssor and drills...	0.009	0.015	0.016	0.012	0.0142
.....	0.001	0.006	0.007	0.007	0.0050
.....	0.003	0.002	0.006	0.005	0.0060
.....	0.043	0.037	0.011	0.003	0.0036
ngs.....	0.023	0.011	0.003	0.002	0.0029
.....	0.003	0.004	0.003	0.003	0.0039
.....	0.006	0.005	0.003	0.002	0.0022
.....	0.006	0.004	0.004	0.004	0.0043
s.....	0.003	0.001	0.001	0.001	0.0017
.....	0.004	0.011	0.004	0.006	0.0053
.....	0.115	0.104	0.060	0.055	0.0642
re.....	$1.130	$1.115	$0.951	$0.875	$0.9740

Example 18. Tin Mining

Table 82. A Summary of 1911 Operations of the Bries Co, Briesis and Ringarooma Properties, D

Briesis drift and overburden sluiced	
Ringarooma upper drift sluiced	
Ringarooma overburden sluiced	
Total ground removed	
Output of black tin	
Output metallic tin	
Costs	
Sluicing, hydraulicking, cleaning reef, breaking, wh slacking fallen overburden and modern alluvial	
Extending pipes and races in faces, erecting elevators, tenance	
Cleaning up, dressing, and bagging	
Electric lighting	
Head races, maintenance, and repairs	
General stores, timber, and explosives	
Pumping, repairs, and renewals to plant, stable and ot expenses	
Salaries of staff, foreman, and office expenses	
Total cost of winning and delivering, dressed and bag ment at mine sheds	
Cartage, smelting, freights, etc	
Total Cost of Tin at Market	
Depreciation on head races not included in above	
It required about 18 500 gal of water to sluice 1 cu yd of d of dirt required to produce one ton of black tin was Average cost of removing 1 525 500 cu yd of overb London office expenses not included in costs given.	
(For working conditions see *Mining and Scientific*	

Example 19. An Exploratio

Table 83. Summary of Expense for C

23 653 ft of holes in S. W. United States, 1909, 1910, 1

Sup't salary	$0.169	Depreciatio	
Runners	0.417	Other expen	
Helpers	0.281	Holes by co	
Assaying and sampling	0.219	Total sun	
Teamsters and helpers	0.058	Labor	
Foremen	0.114	Fuel	
Clerks	0.025	Oil and grea	
Roads, etc	0.093	Repairs to p	
Total labor	$1.376	Repairs to p	
Fuel	0.317	Total Pur	
Oil and grease	0.022	Total at Mi	
Iron and steel	0.003	General offi	
Reps and new equipment	0.256	Salaries	
Cables	0.099	Stat'y and	
New drill tools, etc	0.107	Postage	
New surface tools, etc	0.011	Furniture	
New equipment	0.006	Office exp	
Reps to machines, etc	0.032	Telephone	
Lighting	0.001	Legal	
Barn supplies	0.035	Rent	
Building material	0.014	Travel	
Chemicals	0.031	Total Gen	
Stationery and printing	0.003	Grand t	
Telegrams	0.003	(For workin	
Transportation	0.032	*ing and M*	
Miscellaneous	0.032	page 501.	

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xample 20. Copper Mining Costs

g copper in 1912 from North American ores is given in following production was estimated at 1 489 168 562 lb refined copper. are mine productions, representing the amount of copper actually ith probably a small balance on hand at the mines. Taking the bout 72% of the North American copper production is accounted he world's production. For further details, see *Engineering and* 1913.

4. Summary of 1912 Production and Costs

(Total Expenditures)

(Table 85).....	205 636 620 lb Cu @ 10.58¢
le 86).....	249 712 570 " " @ 11.44¢
(Table 87).....	473 044 379 " " @ 10¢
.....	928 393 569 lb Cu
o (cost estimated).....	140 628 798 " " @ 8.5 to 9¢
ll expenditures, including de-	1 069 022 367 lb Cu @ 10.5¢
tures except deferred charges	9.55¢

Porphyry group, included in 11.44¢ average price, amount to roduced.)

ake Superior Group; Production and Costs, 1912

	Tons of ore stamped	Lb Cu per ton	Total pounds copper prod	Per pound copper		
				Expended	Credit	Net cost
....	2 806 610	31.22	67 856 429	.....	.....	9.86¢
....	1 309 253	15.80	20 634 800	12.5¢	0.72¢	11.6
....	1 246 557	14.8	18 413 387	10.36	0.41	9.95
)...	765 306	22.5	17 225 508	.....	.....	8.8
....	652 260	25.2	16 455 769	.....	.....	7.69
....	652 433	20.5	13 373 961	.....	.....	10.94
....	787 941	15.22	11 995 598	.....	.....	10.61
....	401 308	23.45	9 408 960	.....	.....	7.59
....	531 105	15.4	8 186 957	.....	.....	11.4
....	421 385	18.8	7 908 745	.....	.....	13.05
89).	366 663	19.04	6 908 713	.....	.....	11.73
....	333 618	16.56	5 525 455	.....	.....	13.4
....	106 517	16.36	1 742 338	.....	.....	13.4
....	.....	....	205 636 620	.....	.....	10.58¢

phyry Copper Group; Production and Costs, 1912

orphyry ores = about 61.4% in refined copper of estimated aver- reserve during 1912. (See *Mining and Scientific Press*, June 7,

Tons of e milled	Yield per ton lb Cu	Total pounds copper produced	Cents per pound of copper				
			Operating cost	Mis'c credits	Au and Ag credit	Cost reported	Deferred ch'gs
5 315 321	17.2	91 366 337	9.98	0.054	0.056	8.97	2.80
2 887 731	21.9	63 063 261	11.63	2.46	0.820	8.34	0.99
1 565 875	22.2	34 674 275	10.53	0.827	0.023	9.68	5.41
1 040 744	31.5	32 832 609	10.15	0.02	0.13	10.00	0.98
1 122 666	24.7	27 776 088	7.69	0.45	....	7.24	4.75
1 932 337	20.9	249 712 570	10.23	0.83	0.57	8.83	2.61

Example 20. Copper Mining Costs

Table 87. Miscellaneous Group; Production

Mine	Tons of ore milled	Yield per ton lb Cu	Total pounds of copper produced	Cen	
				Oper-ating cost	C s c
Anaconda	5 069 242	58.0	294 474 161	12.3	
North Butte	425 297	62.0	26 480 123	13.1	
Tennessee	444 289	29.8	13 352 634	....	
Shannon	.....	....	16 406 336	12.6	
Granby	739 519	17.9	13 231 121	17.3	
British Columbia	740 589	15.0	11 146 811	18.5	
Calumet and Ariz	159 513	92.1	16 490 229	10.66	
Sup and Pittsb'rg	288 429	127.0	36 618 399	7.67	
Mason Valley	241 822	66.5	16 058 493	14.8	
Old Dominion	.....	....	16 533 999	} 9.0	
United Globe	188 524	65.0	12 252 073		
Total and average	.....	....	473 044 379	12.16	
Phelps, Dodge & Co	.....	....	140 628 798	cost e	

Example 21. Copper Range Consolidated Co,

Table 88. Total Production, Receipts, and Expenditur Trimountain Mines, to Dec 31, 1

	Baltic Mine	C
Total pounds of copper produced	161 391 344	16
Receipts:		
From the sale of Capital Stock	$ 1 000 000.00	$ 1
From assessments	800 000.00	1
From sale of copper product	23 462 697.34	23
From interest	39 520.26	..
Total Receipts to Date	$25 302 217.60	$2
Expenditures:		
For mine location	$ 1 000 000.00	$ 1
For new lands	19 251.00	
For operation and construction	16 226 860.25	16
Total Expenditure to Date	$17 246 111.25	$17
Net balance of receipts	$ 8 056 106.35	$ 8
Dividends paid to date	7 750 000.00	7
Bal of Receipts Over Exps and Divs.	$ 306 106.35	$
Average price received for copper	14.5¢	
Operating, construction, and equipment cost, per lb of copper	10.1¢	
Operating profit per lb of copper, not including interest received	4.4¢	

Receipts from rents, etc, have been deducted from operat
(See Table 89 for details of 1912 costs.)

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. Copper Range Consolidated Co (*Concluded*)

oduction and Costs, 1912, for Three Principal Mines

	Baltic Mine		Champion Mine		Trimountain Mine	
	Totals	Per ton milled	Totals	Per ton milled	Totals	Per ton milled
...	$683 840.13	$1.050	$688 981.54	$0.900	$351 596.45	$0.961
...	34 850.86	0.054	44 016.64	0.057	19 289.25	0.053
...	49 163.59	0.076	38 899.46	0.052	32 069.74	0.088
...	50 017.54	0.077	62 189.81	0.081	52 459.44	0.143
s..	108 337.89	0.166	117 433.57	0.153	64 287.33	0.175
...	$926 210.01	$1.423	$951 521.02	$1.243	$519 702.21	$1.420
...	$45 701.60	$0.072	$45 573.75	$0.059	$57 991.86	$0.158
...	73 744.39	0.112	45 212.01	0.059	28 231.46	0.077
...	.....	....	1 402.00	0.002	.....	....
...	2 379.00	0.004	3 488.00	0.005	2 361.00	0.006
le-						
...	.....	....	929.42	0.001	.....	....
...	3 238.24	0.005	4 422.16	0.006	2 394.31	0.006
ce.	.....	....	1 750.00	0.002	.....	....
...	7 713.11	0.012	5 204.46	0.007	1 579.10	0.004
nill	89 091.42	0.136	104 695.35	0.137	41 293.06	0.112
...	139 349.16	0.214	154 532.96	0.202	68 641.49	0.187
ce	$1 287 426.93	$1.98	$1 318 731.13	$1.72	$722 194.49	$1.97
...	8 662.21	0.01	14 688.08	0.02	8 648.06	0.02
ce						
...	$1 278 764.72	$1.97	$1 304 043.05	$1.70	$713 546.43	$1.95
...	61 276.76	0.097	62 199.21	0.081	40 681.39	0.11
ad						
...	127 915.25	0.197	167 549.52	0.219	70 081.32	0.19
...	$1 467 956.73	$2.264	$1 533 791.78	$2.000	$824 309.14	$2.25
...	5 037.57	0.007	2 953.58	0.004	5 115.36	0.01
...	$1 462 919.16	$2.25	$1 530 838.20	$1.996	$819 193.78	$2.24
r..	10.94¢		8.88¢		11.73¢	
...	705 281		804 994		403 089	
...	7.49%		4.9%		9%	
...	652 433		765 306		366 663	
...	22 444 810		28 460 500		12 417 575	
...	58.2 to 1		54 to 1		59 to 1	
ro-						
...	13 373 961		17 225 508		6 980 713	
...	20.5 lb		22.5 lb		19.04 lb	
...	464 ft		429 ft		263 ft	
...	10 547 "		9 343 "		7 746 "	
...	679 "		1 029 "		401 "	
...	1 172 "		1 740 "		1 770 "	
...	12 862 ft		12 541 ft		10 180 ft	

Example 22. Wolverine Copper Mining Co, Lake

Table 90. Summary of Production and Cos

	1911	
	Total amt	Per sta
Tons copper rock mined	400 296	...
Rock sorted out as waste	2.95%	...
Tons rock stamped	388 476	...
Lb mineral produced	12 227 500	...
Lb refined copper produced	9 617 168	...
Yield, lb copper per ton stamped	.....	24
Total cost per lb refined copper	7.542¢	...
Underground costs:		
Sinking	$ 7 468.90	$0.
Drifting	27 474.75	0
Stoping	210 846.60	0
Sundry labor	388.15	0
Timbering	8 550.55	0.
Tramming	69 857.45	0
Bosses and labor	38 886.25	0
Mechanics	10 813.25	0
Hoisting and pumping	23 298.85	0.
Compressor	35 201.72	0.
Teaming, etc	2 803.10	0.
Supplies and fuel	23 120.54	0.
Electric lights	1 913.09	0.
Total underground costs	$160 623.20	$1.
Less profit on supplies to cont'rs	72 665.00	0
Net underground costs	$387 958.20	$0.
Rock-house costs:		
Labor	$13 163.15	$0.
Machinists	1 523.18	0.
Fuel	2 122.00	0
Supplies	2 189.20	0
Teaming, etc	751.75	0.
Electric lights	931.78	0.
Total rock-house costs	$20 681.06	$0
Stamp mill:		
Transport ore to mill	$55 849.63	$0.
Supplies and electric lights	17 323.86	0.
Machinists	8 348.96	0
Fuel and teaming	25 613.10	0.
Labor	32 547.35	0.
Pumping	14 631.55	0.
Total stamp mill expense	$154 314.45	$0.
Surface:		
Superintendence and labor	$22 136.15	$0.
Supplies	5 488.84	0.
Telephones, telegrams, etc	482.69	0.
Taxes and insurance	45 481.00	0
Freight on mineral to smelter	5 894.66	0.
Total surface expense	$79 483.34	$0.
Less rents received from houses	7 383.00	0.
Net surface cost	$72 100.34	$0.
Diamond drilling	$2 399.25	$0.
Construction	2 191.11	0.
Smelting, freights, selling, Boston office, etc	85 669.54	0.
Total Expenditure	$725 313.95	$1.
Development work included in costs	5 153.9 feet	

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23. Miami Copper Co, Miami, Arizona

l Summary of 1912 Operations, as Shown by Annual Report
(See Remarks, preceding Table 103)

....per ton ore		$1.203
.... " " "		0.659
se.. " " "		0.180
and milling, per ton ore treated		$2.042
ing, and general expenses, per lb refined copper		$0.06478
ates to smelter,	" " "	0.00688
nd freight on pig copper,	" " "	0.02321
eight and insurance,	" " "	0.00128
ense,	" " "	0.00122
nses,	" " "	$0.09737
content and rent receipts,	" " "	0.0015
e,	" " "	$0.0958
d off profit and loss, not charges	" " "	0.0052
g depreciation,	" " "	$0.1010

.....	1 040 744 tons, assaying 2.393% Cu
uced.....	46 683 tons, assaying 37.020% Cu
rates.....	34 560 665 lb
ore.....	33.21 lb
.....	69.39%
ns.....	32 832 609 lb refined copper

n the following tables were furnished by J. Parke Channing, Copper Co, and give more detailed costs of some of the items are approximate as to tons per shift.)

le 92. Summary of Mining Costs, 1912
(See also Table 103)

	Tons	Cost per ton
ble 95).....	118 130	$1.17171
Table 93).....	450 209 ($1.01008)	.....
94).....	228 926 (0.16828)	0.47350
.....	244 504	.....
.....	1 041 769 Average	$0.60636
e per ton of ore mined to extinguish development		
o, 101, 102).....		0.31000
).....		0.09909
7).....		0.06327
8).....		0.00852
and expense (Table 99).....		0.04307
d supplies (Table 99).....		0.01586
ting (Table 99).....		0.00439
ampling (Table 99).....		0.02495
ise (Table 99).....		0.02776
g cost.....		$1.20327

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Example 23. Miami Copper Co (C

Table 93. Details of Sub-level Caving or Squar

Total tons mined by sub-level caving, 450 209 tons	
Costs:	
Miners	33.20 tons of ore per shift
Muckers	11.27 " " " " " "
Explosives	0.52 lb of 40% per ton ore
Timbermen, 26.70 tons of ore per shift	
Timber	9.30 bd ft per ton ore
Nails, etc	
Air-drill expense	
Bossing	
Miscellaneous	
Total	
Summary: Labor, 6.10 tons per shift, $0 60328 per ton; su	
Note. — During first 5 months of 1912 a shift consisted of 8 last 7 months, a shift consisted of 8 hr from shaft mouth t about 7.5 hr actual working time.	

Table 94. Details of Shrinkage Stopi

Total tons mined by shrinkage stoping, 228 926 tons	
Costs:	
Miners, 99 tons per shift	
Muckers, 102 tons per shift	
Explosives, 0.25 lb of 40% per ton ore	
Timbermen, 1 584 tons of ore per shift	
Timber, 0.24 bd ft per ton	
Nails, etc	
Air-drill expense	
Drawing chutes, 247 tons per shift	
Bossing	
Miscellaneous	
Total	
Summary: Labor, 38.71 tons per shift, $0.09352 per ton; s	
(See note at bottom of Table 93.)	

Table 95. Details of Stock Pile Ore

Total tons taken from stock pile, 118 130 tons	
Costs:	
Labor, 25.24 tons per shift	
Supplies	
Fixed charges	
Total	

Table 96. Details of Cost of Hauling Ore

Operating:	
Motor, brake, and chutemen, 77 tons per shift	
Track cleaners	
Car oilers	
Bosses	
Power	
Lubricants	
Total Operating	
Maintenance:	
Locomotive repairs	
Car repairs	
Track repairs	
Trolley line repairs	
Chute and drift repairs	
Generator set repairs	
Total maintenance	
Total maintenance and operating	
Labor operating, 59 tons per shift. Tons per motor shift,	

ple 23. Miami Copper Co (*Continued*)

97. Details of Hoisting Costs per Ton Ore

isting		No 4 cage hoisting	
.....	$0.00428	Engineers and oilers.....	$0.00451
npers....	0.01009	Caging and landing.....	0.01141
.....	0.00003	Total labor, 232 tons per	
s per shift	$0.01440	shift.....	$0.01592
.....	0.00089	Lubricants.....	0.00015
.....	0.01339	Power.....	0.00666
.....	$0.02868	Total Operating.....	$0.02273
.....	$0.00620	Cage hoist maintenance....	0.00173
.....	0.00198		
.....	0.00185		
e.....	$0.01003		
g.....	$0.03871	Total Cage Hoisting.....	$0.02446

le 98. Pumping Cost Details per Ton Ore

.....	$0.00298	
.....	0.00087	
.....	0.00004	
.....		$0.00389
ps.....		0.00463
.....		$0.00852

). Details of Sundry Charges per Ton of Ore

d:		Handling men and supplies at No 2 shaft:	
.....	$0.01208	Operating:	
.....	0.00249	Engineers, cagers.....	$0.01246
nen, etc..	0.00557	Power.....	0.00035
p labor...	0.00637	Lubricants.....	0.00011
.....	0.00869	Total.....	$0.01292
.....	0.00350	No 2 shaft hoist maint'ce	0.00169
lies.....	0.00211	No 2 shaft repairs.....	0.00125
ense.....	0.00226	Total.....	$0.01586
.....	$0.04307		
npling:		Mine surface expense:	
.....	$0.01184	Mine clerks.....	$0.00236
.....	0.00953	Janitors and watchmen...	0.00314
.....	0.00198	Miscellaneous surface.....	0.00497
.....	0.00015	Waste dump.....	0.00186
.....	0.00145	Handling supplies.....	0.00915
.....	$0.02495	Assaying.....	0.00433
		Miscellaneous supplies....	0.00195
g:		Total.....	$0.02776
.....	$0.00255		
.....	0.00184		
.....	$0.00439		

Example 23. Miami Copper Co (

Table 100. Drifting, Cost of 5- by 7-ft Drift, Timbered Based on 25 450 linear ft

Miners....3.24 ft per shift	
Muckers....2.28 " " "	
Explosives, 8.42 lb per ft	
Timbering:	
Timbermen, 42 ft per shift	
Timber.....14 bd ft per ft of drift	
Nails	
Track laying:	
Trackmen	
Rails, etc	
Air-drill expense	
Bossing	
Miscellaneous:	
Labor	
Supplies	
Total cost per foot of drift	

Summary: Labor, 1.22 ft of drift per shift, $2.96370 per ft
$2.55602 per ft. (See Note, Table 93.)

Table 101. Drifting, Haulage Drifts, about 10- by 10-f with 9-ft Posts and 8-ft Caps; Costs based o

Miners.....1.58 ft per shift	
Muckers....0.44 " " "	
Explosives, 13.16 lb per ft	
Timbering:	
Timbermen, 1.70 ft per shift	
Timber.....225 bd ft per ft of drift	
Nails	
Temporary tracks:	
Trackmen	
Rails, etc	
Air-drill expense	
Bossing	
Miscellaneous:	
Labor	
Supplies	
Total cost per foot of drift	

Summary: Labor, 27 ft per shift, $13.34685 per ft of drif
$9.67758 per ft. (See Note, Table 93.)

Table 102. Raising, 5- by 5-ft Raise, Costs Based on

Miners....3.92 ft per shift	
Muckers...0.34 " " "	
Explosives, 4.20 lb per ft	
Timbering:	
Timbermen, 23 ft per shift	
Timber.....17 bd ft per ft of drift	
Nails, etc	
Air-drill expense	
Miscellaneous:	
Labor	
Supplies	
Total cost per ft of raise	

Summary: Labor, 3.05 ft per shift, $1.21741 per ft of
plies, $1.55091 per ft. (See Note, Table 93.)

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Months, 1912	Daily product, tons ore handled	Labor, tons per shift			Dynamite, lb per ton ore	Timber			Bd ft timber per ton ore
		Direct	Indirect	Total		Cost per 1 000 bd ft			
						Timber	Surface expense	Total	
January	2 171	5.70	29.6	4.78	0.74	$23.05	$5.12	$28.17	7.36
February	2 540	5.45	33.0	4.68	0.77	23.68	3.48	27.16	8.46
March	2 798	5.40	36.8	4.71	0.79	22.70	3.52	26.22	8.47
April	2 782	5.31	32.5	4.56	0.85	23.92	3.36	27.28	8.76
May	2 840	5.35	33.4	4.76	0.83	23.42	3.46	26.88	8.07
June	2 998	5.77	33.4	4.92	0.82	22.94	4.00	26.94	7.17
July	3 110	5.76	33.2	4.91	0.75	22.28	4.51	26.79	6.77
August	3 155	5.56	30.6	4.70	0.78	23.74	4.50	28.24	6.87
September	3 105	5.77	30.0	4.83	0.72	23.51	4.02	27.53	7.26
October	3 080	5.64	32.9	4.82	0.62	24.07	4.04	28.11	6.79
November	2 859	5.49	29.1	4.62	0.50	23.73	3.81	27.54	7.83
December	2 810	5.14	25.9	4.28	0.47	22.07	3.77	25.84	8.36
Average	2 854	5.54	31.5	4.71	0.72	$23.25	$3.93	$27.18	7.65

Note. — Direct labor comprises all operative underground labor, hoist engineers, and mine foreman, excluding repair labor, surface labor, engineers, samplers, timekeepers, clerks, watchmen, etc. (See Note, Table 93.)

STANFORD LIBRARIES

Example 23. Miami Copper Co (*Concluded*)

Table 104. Milling Costs, Details of Labor, Power, Supplies for Operations

Year, 1912	Labor		Total supplies, cost per ton	Power		New water supply, cost per ton	Total operating cost per ton	Maintenance per ton	Grand total per ton
	Tons per shift	Cost per ton		Kw-hr per ton	Cost per ton				
Coarse crushing	433	$0.00697	$0.00084	0.461	$0.00571	.....	$0.01352	$0.01502	$0.02854
Conveying	351	0.00693	0.00005	0.229	0.00283	.....	0.00981	0.00063	0.01044
Intermediate crushing	263	0.01375	0.00003	1.968	0.02435	.....	0.03813	0.03960	0.07773
Fine crushing	442	0.00830	0.00003	5.273	0.06528	.....	0.07361	0.05383	0.12744
Classifying	953	0.00382	.....	.....	.....	.....	0.00382	0.00896	0.01278
Distributing	.....	.....	.....	.....	.....	.....	.....	0.00099	0.00099
Sand tables	260	0.01397	.....	0.571	0.00707	.....	0.02104	0.01042	0.03146
Slime tables	249	0.01463	.....	0.669	0.00829	.....	0.02292	0.00508	0.02800
Recrushing middlings	4 018	0.00089	.....	0.657	0.00814	.....	0.00903	0.00306	0.01209
Tailing disposal	490	0.00568	0.00020	0.400	0.00493	.....	0.01081	0.01562	0.02643
Stacking and loading concentrates	232	0.01234	0.00461	0.344	0.00425	.....	0.02120	0.00499	0.02619
Reclaimed water	173	0.00681	0.00016	1.817	0.02163	.....	0.02860	0.00280	0.03140

Nevada Consolidated Copper Co, Ely, Nevada

General Summary, 1912 Operating Results as shown in Annual Report

	2 852 515
t direct to smelter	35 216
	2 887 731
opper produced	63 063 261
ned copper per ton ore treated	21.9

		Total	Per lb Cu, cents
roportion of stripping expense		$1 436 366	2.28
luction plants		759 129	1.20
s		1 414 506	2.24
		1 480 089	2.36
t, including depreciation		1 204 630	1.91
		918 151	1.45
		103 359	0.16
s charged		$7 316 230	11.60
nd silver		521 277	0.83
ld and silver		$6 794 953	10.77
ments	$1 459 112		
	50 000		
t	22 883		
eous	9 925	1 541 920	2.44
or copper at Atlantic seaboard		$5 253 033	8.33
ing pits, stripping, tracks, steam he year, deferred to future operations		620 714	0.98

en in report:

ed	2 852 515
	1.692%
copper	68.25%
on	9.09
d content	0.016 oz
er content	0.049 oz
gold	45.84%
silver	50.11%
ver recovered per ton ore	16.48¢
ncentrates	10.49%

Example 24. Nevada Consolidated C

Description of Working Conditions. The major
from steam-shovel pits. The capping, averaging sligh
moved by steam shovels, working on benches averagin
done by churn drills, one drill serving each shovel. D
20 ft from top edge of bank, and carried to a depth of 5
hole is first sprung with dynamite and then charged wi
pending upon the ground. The heights of benches vary
grades to the various overburden dumps. Eight 1
saddle-back locomotives, and 50 18-yd one-way dump
There are two shifts, 9 hr each.

The ore is of the disseminated porphyry type, and
monzonite porphyry intrusive. Analysis of ore:

Au	Ag	Cu	SiO_2	Fe	C
0.015	0.04	1.6	71.0	4.0	

The ratio of concentration as determined in practice
concentrates. Ore is hauled from mines to concentrato
level elevation at mines, 7 250 ft; at concentrator and
crushing is done by gyratories and rolls, and fine crush
Chilian and Huntington mills in others. Concentratio
Concentrates roasted in 16 18-ft McDougall furnaces.
in 5 reverberatory furnaces, with California fuel oil.
Pierce-Smith basic converters. The power house has t
ators. There are also 3 blowing engines of 6 000, 1
respectively, compressing air to 15 lb for the converting
consists in removing the waste overburden from the o
an average of 2 752 000 cu yd of overburden is remov
being 2 850 000 cu yd.

Miscellaneous Data (furnished by officers of the N

Table 106. Average Number of Men Employ

Mining	
Engineers, timekeepers, clerks, etc	14
Superintendents and foremen	18
Steam shovels:	
Engineers	16
Cranesmen	16
Firemen	16
Pit laborers	90
Dumping waste	110
Locomotive operations:	
Engineers	24
Firemen	24
Brakemen	32
Switch tenders	7
Yardmasters	2
Trackmen	81
Blasting:	
Churn drill men	30
Powder men	11
Shops:	
Mechanics, etc	44
Laborers	73
Teamsters	3
Total men at mine	611
Tons ore per man per day	12.6
Tons ore and waste per man per day	40.2

Supt, fore	
Unloading	
Sampling	
Coarse gri	
Fine crush	
Transport	
Laborator	
Water serv	
Misc'l labo	
Repairme	
Total m	
Average t per day	
Calcining:	
Operati	
Repair l	
Reverb'y	
Operati	
Repair l	
Convertin	
Operati	
Repair l	
Power hou	
Laborator	
Shops	
Industrial	
Yard and	
Managem	
Total s	

Total average number employed in all dep

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Nevada Consolidated Copper Co (*Continued*)

-pit mine the steam shovels are under their own power. Bucy- used, of approximately 100 h-p capacity. Shovels are actually f shift time. Fuel consumption averages 2.86 tons of engine e steam-shovel equipment is employed about 40% in mining ore The steam locomotives are of the saddle-tank type, 16 by 24 linders. Fuel consumption averages 2.42 tons of engine coal per tely 26% of motive power equipment is employed in ore service verburden to waste dumps. An average of about 5 500 i h p power is used. The waste heat from the reverberatory furnaces this power and averages about 70% of the total fuel cost, which, ack coal equivalent, gives an average of 9.6 tons slack coal con- year.

ver Distribution and Costs for Milling, Smelting, Etc

tion		Average cost per hp-year	
.....	4 150 h p	Fuel.....	$43.89
.....	1 150 "	Operating labor.....	10 10
.....	250 "	Supplies and operating repairs..	11.89
d.....	5 550 h p	Cost per hp-year	$65.88

oal Consumption for Mining and Stripping, per Dry Ton of and Waste Handled, 10 Months, 1913

	Mining per ton ore	Stripping per cu yd	Total coal used
	Lb	Lb	Lb per ton
.....	4.13	6.10	3.22
.....	3.77	9.72	4.28
.....	7.90	15.82	7.50

Used in Mining and Stripping Operations, 10 Months, 1913

	Mining		Stripping		Total consumption	
	Per cent of total	Per ton ore	Per cent of total	Per cu yd	Per cent of total	Per ton
	Per cent	Lb	Per cent	Lb	Per cent	Lb
.....	73.3	0.21	61.5	0 95	65	0.205
.....	26.7	0.08	38 5	0 59	35	0.111

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Example 24. Nevada Consolidated

Table 110. Data on Keystone Churn-Drilling

Churn-drill holes bored (about 6 in diam)
Total linear feet of holes drilled
Average depth of holes
Ground broken per foot drilled
Average cost per foot drilled
Average depth of hole drilled per 10.5-hr shift
Coal used per 10.5-hr shift
Drill crew consisted of driller, tool-dresser, and o

Table 111. Data on Steam-shovel

Steam shovels
Average material moved in 9 hr
Maximum material handled per day
Maximum material handled in 31 days
Coal used per steam-shovel shift
Material moved per ton of coal consumed

Consumption of steel rails for steam-sh new track or lines for temporary use, amounted This is equivalent to 19.61 tons ore per lb of s fish-plates, tie-plates, spikes and bolts.

Water Used in Milling. Approximately 4. per ton of ore in milling, or 7 200 gal per min of r of ore daily. Of this fresh water, 5 000 gal per m ers and used by gravity in mill circulation. The back into circulation from ponds at concentra makes total water in circulation in mill 20 200 ore daily, of which 7 200 is fresh water added to about 20% solids.

	Iron County	Iron County	Iron County	son County	Baraga County	Mar-quette County	Mar, quette County	quette County	parts of state
Mining	$1.35	$1.30	$0.97	$1.13	$1.28	$1.67	$1.33	$1.58	$0 65
Exploration and development	0.10	0.64	0.04	0.09	0.204	0.10	0.08	0.07	0.08
Construction, shafts, machinery, etc	0.16	0.25	0.23	0.10	0.01	0.19	0.11	0.08	0.15
General expense and administration	0.099	0.09	0.064	0.097	0.065	0.064	0.12	0.08	0.045
Taxes	0.063	0.025	0.02	0.08	0.01	0.12	0.07	0.05	0.015
Total expenditure at mine	$1.772	$2.305	$1.324	$1.497	$1.569	$2.144	$1.71	$1.86	$0.94
Royalties	0.39	0.21	0.24	0.24	0.181	0.14	0.16	0.26	0.12
Rail freights	0.40	0.40	0.40	0.40	0.35	0.40	0.32	0.32	0.40
Lake freights	0.70	0.54	0.57	0.60	0.61	0.60	0.65	0.65	0.60
Commissions	0.045	0.067	0.09	0.07	0.06	0.02	0.01	.....	0.10
Total expenditure, f o b, Cleveland, Ohio	$3.30	$3.52	$2.62	$2.80	$2.77	$3.30	$2.85	$3.09	$2.16
Average loss per ton	.....	0.22	.....	.....	0.02	.....	.....	.....	0.22
Average profit per ton	1.04	.....	0.83	0.84	.....	1.10	0.85	0.33	.....
Avg value per ton, f o b, Cleveland, Ohio	$4.34	$3.30	$3.45	$3.64	$2.75	$4.40	$3.70	$3.42	$1.94

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Example 26. St Lawrence Pyrites (

(Data furnished by F. T. Rubidge, V

Description of Working Conditions. Ore occurs in thickness, and inclined from 30° to 45° from the hori and laminations of gneiss of varying thickness occur i gangue which makes drilling easy, but about 50% of c reality a pyritic gneiss.

Only one mine is worked. Hoisting shaft is single co is unbalanced. Two veins are mined, about 75 ft of ro done by hand, two men to a car. Cars are loaded m though some shoveling is necessary. Average load, 60% of ore is dumped directly into skip; remainder int age hoisting depth, 250 ft vertical. Stoping method go overhand and part underhand. No timbering except f 55-ft centers, are left, making stopes 40 ft long. Hang defined, and hanging stands well.

Table 113. Average Mining Cost per Ton I

Underground		
Development in rock	$0.004	Shipping f
Development in ore	0.183	Breaking
Stoping	0.279	at mine
Timbering	0.015	Engineerin
Tramming by hand	0.146	Administr
Hoisting	0.026	ing supt
Skip tenders	0.010	Warehouse
Pumping	0.033	General su
Underground tracks	0.006	Insurance
Supt, bosses, and timekeepers	0.021	Taxes
General underground expense	0.031	
Total underground mine exp.	$0.754	Total surf
Total mine operating expense per 2 240 lb,		

Note.—These 1912 figures do not include depreciati of home office expense, but do include all maintenance

The charges for development are actual charges for in excess of that required for amount of ore extracted. ft of drilling was necessary, while 2 369 ft was actually c to development, although most of it was incidental to linear ft being 1 575 for the year.

During the year the drilling equipment for stoping w to hammer drills, the latter including air feed and "Ja for January shows 22.3 hammer-drill shifts and 314.8 pi record of 13.6 tons broken per drillman shift (includin cember record shows 257.9 hammer-drill shifts (of wh shifts) and 3.6 piston-drill shifts, with a combined reco (helpers used only on the piston drills).

Of total cost of 88¢ per ton, the total labor on surfa 50¢, excluding administration and mine office expense 19.9¢, and power 11.7¢ per ton. Power is purchased 0.75¢ per kw-hr; hence 14.6 kw-hr were required per drill equipment and the substitution of electrically-dri the power has been reduced to about 8 kw-hr per ton, of 14.5% for hoisting, 9.5% for pumping, 8% for crushi underground hoist.

6. St Lawrence Pyrites Co (*Concluded*)

ures are given for 1912, because the mine has not worked
13. Better results are now obtained than during Dec,
igures taken from stope records show:

rage for May, June, July, Aug, and Sept, 1913

.....		10 542 (2 240 lb per ton)
.....		11 310 " " "
:		
.....	115.0	
.....	127.2	242.2
rilled.....		3 010
drilled.....		18 017
(total hours paid divided		
.....		300.1
.....		207.6
.....		466.3
.....		122.1
.....		35.0
rill shift.....		43.2
" runner shift.....		34.8 (no helper)
an in stope.....		24.8
ot drilled.....		0.58
an (see formula below)....		9.67
ite, fuse, and caps, cost per		
.....		$0.0939
tram (ton-foot basis).....		405 ft
r man shift, muckers, and		
.....		16.78

wing formula is used for determining tons mined per man:

Symbol DR = Drill runners

H = Helpers

CL = Cleaners (stope shovelers)

Tm = Timbermen

$M + Tr$ = Trammers and muckers

$$\text{ined per man} = \frac{\text{Tons broken}}{DR + H + CL + Tm + X}$$

$$\text{r} = \text{Tons broken} + \frac{\text{Tons trammed}}{M + Tr}$$

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Example 27. Coal Mini

Examples of these costs are limited, few data being publication. Tables 115 to 118 present some of the m

Table 115. Data on Anthracite
(Philadelphia & Reading Coal a

Figures are from statements in annual reports, but c as some relate to buying and selling of both anthracit ing receipts from the sales and amounts for purchase o the profit was between 9¢ and 10¢ per long ton of anth 1913. Company apparently paid about $2.50 per ton

Costs per ton 2 240 lb, year end
Tons mined
Mining coal and repairs
Improvements at collieries, etc
Royalty of leased collieries
Improvements and repairs of houses
Taxes on coal lands and improvements
Damages account of coal dirt
Fixed charges
Additional wages, account of sliding scale
Total per ton of 2 240 lb
" " " " 2 000 "

During previous years, company has charged abo of coal lands.

Table 116. Bituminous Coa
(Pittsburg Coal Co, from Sept 1, 189

Tons of coal and coke produced (2 000 lb)
Earnings after deducting expenses for operation, interest on bonds of subsidiaries, and adjustme
Less: Reserve for depletion of coal lands
Depreciation of plant and machinery
Interest on bonds
Total capital charges
Apparent profit per ton, including all income, et
Paid in dividends
Record for year ended Dec 31, 1913
Tons of coal produced
Tons of coke produced
Total tons produced (2 000 lb)
Operating charges: maintenance, repairs, administ ing, and general expenses
Taxes
Interest on bonds, etc, of subsidiary companies
Commercial discounts and interest
Losses, insurance, and pensions
Total operating cost
Total capital charges
Net earnings for year

nple 27. Coal Mining Costs (*Continued*)

verage Tonnage Mined per Day per Man during 1912

n U S Geol Surv Rep, " The Production of Coal in 1912 ")

	Number hours per day	Average tonnage per man per day	Per cent of coal mined by machines (†)	Average price at mines per ton 2000 lb
.....	8 and 9	2.91	23.2	$1.29
.....	8	2.95	3.7	1.71
.....	.....	1.14	1.8	‡2.33
.....	8 and 10	3.72	23.2	1.49
.....	8	3.95	44.9	1.17
.....	8	3.88	54.7	1.14
.....	8	2.37	1.3	1.80
.....	8	2.97	1.1	1.62
.....	8, 9 and 10	3.38	66.4	1.02
.....	10	3.11	2.5	1.18
.....	8	2.11	52.7	1.99
.....	8	2.17	20.7	1.76
.....	8	4.03	32.3	1.82
.....	10	3.28	8.1	1.42
.....	.....	3.46	33.8	1.53
.....	8	3.77	87.0	1.07
.....	8	2.4	7.1	2.14
.....	9	2.1	.....	2.11
.....	8*	3.89	50.8	1.05
.....	9 and 10	2.68	18.6	1.14
.....	.....	1.84	4.8	1.67
.....	8	3.18	3.8	1.67
.....	10	3.6	40.8	0.96
.....	8	2.69	7.7	2.39
.....	8, 9 and 10	3.68	52.3	0.94
.....	8	3.85	32.1	1.58

of employes, 40% about equally divided between 9 and 10 hr.
on per year per machine, 13 763 short tons.

No 4, "Coal Mining Practice in Dist No 7," May, 1914, Illinois Coal
District includes mines operating on bed No 6, west of Duquoin
Bond, Clinton, Fayette, Macoupin, Madison, Marion, Montgomery,
St Clair, Shelby, Washington, and a portion of Perry, Christian,
ies. Bed No 6 varies from 2.5 to 14 ft thick, average 7 ft. There
mines, but shafts are generally used, as bed lies at considerable
ft. General system of mining is unmodified double-entry room-
are worked by panel system. Average recovery of coal, 55%.
l June 30, 1912, 22 454 672 short tons, or 39.1% of production of
ng, 196; shipping, 150; local, 46. Tons mined by machines,
ned per 8-hr shift: 140 by chain, 71 by puncher machines. Aver-
mines, 158. Days work performed, 4 399 826. Employes: sur-
und, 25 493; total, 27 847. Aver number underground men for
3. Tons mined per day per man, 5.1; per surface man, 60.5; per
5; per worker in shipping mines only, 7.3 tons. Tons mined for
788. In one mine 1 keg powder breaks 30 tons coal in shooting
ns after puncher undercutting. Tons broken per keg of powder,
ding on method. Cost of powder, from 1¢ to 10.3¢ per ton coal.
ring year, 426 353 kegs. Timber props range from 1.3 to 7.2 per
ng from 10.5¢ to 70.3¢ per 100 sq ft of roof, or 0.5¢ to 3¢ per ton
locomotive haulage, 78; cable, 3; mules, 101; hand power,

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Example 27. Coal Mining Cos

At one mine a gasolene locomotive hauls to shaft bottom
suming 18 gal gasolene) at cost of 0.17¢ per ton. A
(consuming 15 gal gasolene) at cost of 0.2¢ per ton;
lb each, including cars. Ton mileage in coal of all locos
per shift. Cost of keeping mules, from 75¢ to $1 per da
in favor of loads, 2 mules, weight 1 300 lb each, made
cars, capacity 3 500 lb, weighing 1 000 lb empty, the
about 54.67.

Table 118. Cost of Hydraulic Filli

(U S Bureau of Mines, Bull No 60, to which refe

Based on plants of at least 400 cu yd daily capacity. Items of cost in ct per cu yd		Culm	Mater	
			Culm mixed with crushed breaker and boiler refuse	Local drau sand, gra clay,
Surface transport (mechanical)	Gravity.....	1 to 1.5	4 to 5.5	8 to
	Scraper or conveyer..	4.5 to 6	7.5 to 10	11.5 to
	Pump.....	2.25 to 4	5.25 to 8	9.25 to
Intermediate transport	Shaft or slope	0.25 to 0.5	0.75 to 1.5	0.9 to
	Bore hole....	0.1 to 0.2	0.2 to 0.5	0.3 to
Underground transport	Trough.....	0.1 to 0.3	0.1 to 0.3	0.1 to
	Pipe.....	0.3 to 0.5	0.3 to 0.5	0.3 to
Distribution	In flat workings.....	7.25 to 9	7.25 to 9	7.25 to
	In incline workings..	12.25 to 14.5	12.25 to 14.5	12.25 to
	In pitch workings..	14 to 27.5	14 to 27.5	14 to
Drainage	Incidental...	0.4 to 0.7	0.4 to 0.7	0.4 to
	Special.....	1.25 to 8	1.25 to 8	1.25 to
Total	Minimum...	9.3¢	9.3¢	9.3
	Maximum...	56.25¢	56.25¢	56.2

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SECTION 2

WAGES AND WE

BY

EDWARD K. JUL

ASSISTANT PROFESSOR OF MINING, COLUMBIA SCHOO

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Note. — Numbers in parentheses refer to Bibliograph

WAGES

1. "Day's Pay" or "Comp

Day's pay system, an agreed amount of wages work, is open to the objection that it entails an u to avoid waste of time and supplies. But, for ma engineers, mechanics, foremen, office force, and mi pay system is the only practicable one.

For underground work, day's wages are also preferable in those mines in which the minerals are irregularly dist in the separate handling of ore and waste, or in which th demanding slow and painstaking operation; both of thes by any system which aims to promote speed.

Day's wages compared. Data in Tables 1 and of comparison for estimates of mining cost, since la ponderating factor (power and supplies being the tion. Wages for the same class of labor are natura where supply of workmen is limited, living conditi is expensive.

able 1. Day's Wages in the United States

on	Pa Anthracite, 1912-16	Alabama Coal, 1906	Lake Superior Copper, 1916	Bingham, Utah, 1907	Joplin, Mo, 1909	Telluride, Colo, 1907	Butte, Mont, 1913	Globe, Ariz	Goldfield, Nev, 1913	Griss Valley, Cal, 1907	Minn Iron Ranges, 1914
	$	$	$	$	$	$	$	$	$	$	$
			6 70						4 00	5 00	4 75
			3 85		4 00			5 00	3 10	3 75	3 45
		2 50		3 25	2 75	4 50		3 50	2 40		
		1 50		3 00	2 25	4 00		3 50			
	2 50	2 25	2 75	3 00		3 00	4 00	3 50	2 15	3 00	2 55
	1 90	1 25	2 30	2 75		3 00	4 00	3 50		2 25	2 35
		1 50	2 30	2 75	1 75	3 00	4 00	3 50		2 50	2 40
	1 65	1 60		3 00	2 50	3 25	4 00	3 50	2 40		
	2 30		2 30	3 00		3 50		4 50		3 00	2 55
	2 25	2 25	2 75		2 75				2 40		2 40
	2 50		2 43		3 50	5 00	5 00	5 00	2 65	3 00	
	2 30		2 00		2 75			4 00	2 55	2 50	2 60
	1 90	1 15	1 92		2 75				2 40		2 50
	1 90		1 92					3 50	2 20	2 25	2 40
	2 75	2 35	2 75		3 00	4 00	4 75	3 50	2 45	3 00	3 80
	2 60		3 00		2 75	4 00	4 75	3 50	2 40		3 50
	2 40		3 00				4 75	3 50	2 50		4 00
								4 00	2 45	3 50	2 65
	2 00		2 08		2 50		4 00	3 50	2 25		2 50
	1 10		2 00						2 05		2 35
	1 90	1 25	1 85						1 90		2 10

Day's Wages in Other Countries (U S Dollars)

abor labor	Kolar, India, 1912	Cobalt, Ontario, 1907	Rossland, B C, 1905	Pachuca, Mex, 1912	Cerro de Pasco, Peru	Rand, So Af, 1913, Whites	Rand, So Af, 1908, Kafirs	Queensland, 1914 (a)	Broken Hill, N S W (a) 1911	Chosen, Korea, 1914	Western Australia, 1904
	$	$	$	$	$	$	$	$	$	$	$
	(a) 5 00	4 00	5 50		8 (a)			3 68		0 63	
			4 50	3 75	5 (a)			2 88		0 50	
		3 25	3 50	1 50	1 50	5 20		2 40	2 64	0 38	3 32
		2 50	2 50		0 75		0 40	2 16	2 40		3 00
	(a) 3 50		3 25	0 65	0 75	4 60	0 36	2 04	2 64	0 25	2 80
			2 50	0 55	0 75	3 00	0 30	1 64	2 28	0 20	2 52
	(b) 0 15		3 25	0 55	0 75	3 36	0 32	1 86	2 28	0 20	2 52
					0 50		0 60		2 40		
	(a) 2 62		3 50	1 40	1 50	5 00	0 28	2 40	2 88	0 35	2 80
					2 00	4 46	0 36	1 96		0 35	
	(a) 3 12	3 25	3 50		5 (a)	5 14	0 48	2 88		0 38	3 20
				1 25	2 00	4 50	0 40	2 40		0 36	
	(a) 3 10				0 75	3 78	0 45	2 00	2 88	0 25	2 80
	(b) 0 16				0 75	3 50	0 45	2 00	2 88	0 25	
	(a) 3 90	3 50	4 00	1 40	1 75	4 84	0 45	2 44	3 00	0 30	3 60
	(a) 3 90	3 00	3 50	1 40	2 25	4 80	0 28	3 00	3 00	0 33	3 60
	(a) 4 00		4 00	1 40	4 00	4 90	0 36	3 00	2 75	0 35	3 60
					2 50				2 75		
	(b) 0 15				1 50		0 40	2 16		0 30	
							0 48	1 44		0 18	
	(b) 0 09		2 50	0 55	0 50		0 28	2 00	2 28	0 18	2 40

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2. Sliding Scales for

Sliding scale arrangements aim to establish price of mine product, where latter is subject to vantage of workmen and operator.

Most noteworthy examples in U S have been: PE First award of the Anthracite Strike Commission (1) increase of 10% in mine wages over those in force duri vision as a minimum basis, award No 8, effective Apr crease of 5¢ above a base price of $4.50 per ton domesti harbor points), wages should be advanced 1%. Aver p by an accountant appointed by a U S judge of the 3d contributed by the mining companies in proportion to tinued in force by renewals of the original agreement, but was abolished by mutual consent May 20, 1912. mine managers and organized laborers, June 1, 1912, m workers was set at $4 per day, while copper sold for 17¢ it sold for 15 to 17¢; minimum for surface labor, $3.25

3. Contract, or "Piece-rat

Advantages of contract system rest on the men work more diligently, skillfully, and efficien employer. The system in some form has been ap mine operation. From standpoint of an operat promote speed, reduce unit costs, and diminish e contractor generally obtains better returns and than a day laborer; adoption of contract system most invariably proved mutually beneficial. Ope with certainty on ultimate cost of a given undert to reap reward of his energy, skill, and resourcefu

Drawbacks of contract system, in general, workmen whose interests do not exactly coincid difficulty arises in the enforcement of standards safety. (*b*) Discontent is sometimes engendered a contract, or by refusal of manager to renew a co are unduly lucrative to contractor. Disputes ov work performed are also frequent. (*c*) In metal-m large production is likely to reduce grade of outp checked by levying a fixed charge for hoisting or

Applicability of contract system. For above most successfully applied to operations connected mineral deposits, notably boring, shaft sinking, tu winzes, and raises, and, in general, wherever size of rock fairly uniform or ascertainable, or the c readily measured and inspected.

Examples. COAL is almost universally mined on co on tons or bushels delivered at tipple (Art 11); narro mines is contracted at so much per linear yard, to speci for the coal obtained; "brushing" (taking up bottom s afford necessary head room) is contracted at so much for an agreed width. METAL-MINE STOPING has not factory contract basis, except on the Rand, where rich zone are comparatively uniform over large areas; also at mines, and at other mines characterized by well defined richness. SHOVELING AND TRAMMING are operations w

s because it is often impossible to arrange this work so as to avoid example at Joplin, where 15¢ per ton is paid for shoveling into cars distance of 150 ft. Interesting example of HOLE-DRILLING COS- land, B C (2) for both development and stoping. Machine drills asting crew 1 shift. Cost per ton stoped reduced from $0.865 to , from $11.14 to $8.79. Rate of drifting increased from 50.8 to t sinking, from 27.2 to 58 ft per mo; aver earnings increased by n was later abandoned there, and at other mines where introduced, d extra work of measurement thrown on shift-bosses.

similar work, vary as widely as actual costs; hence not instructive e. For costs, see sections relating to the several operations; in uld be allowed a profit of at least 20% above actual cost, this being ing to company by reason of greater speed. Contractor should furnish his own blasting and other supplies; arrangements as to f machines, tramming facilities, sharpening of drills, bits, repairs, vidual preference.

4. Bonus, or Premium, System

system. From operator's standpoint, two chief draw- ontract system are: difficulty in maintaining standards of that contractor reaps practically the whole benefit of re- ion. Bonus system consists in setting a minimum daily or inimum wages rate based on aver performance); work per- ned) in excess of the minimum is then paid for at an addi- ated that the employer shares the saving in operating cost, n more rapid work. For elaborate comparison of several ibliog (3).

(a) At Franklin mine, IDAHO SPRINGS, COLO (4), 4 by 7-ft drift, tramming, was advancing 3.6 ft per shift, at cost of $11.97 per ft ; helper, $2.75; trammer, $2.50). On establishing bonus system, ance of 90 ft per mo, progress rose to 6.8 ft per shift, at total cost er ft was bonus. Drill runners' wages also rose to $6.58 per shift. v (5), 5 by 7.5-ft drifts were run by two shifts of 2 men each. er, $4; shoveler, $3.75) were paid for aver advance of 1 ft per man nthly). For advance in excess of 4 ft per day, bonus of $1 per ft mbo and Laguna mines; 50¢ in Combination mine. Aver advance al cost of $6.20 per ft, of which $3.77 was for wages. (c) Stoping ne (6) was done by both hand and machine. Miner was charged 45s for gelignite; 2s 9d per shift for native helper. For machine fathom (36 sq ft × thickness of vein), minimum wages of 27s per Of earnings over minimum, one-half was paid to miner, and all e canceled. For hand work, rate was, 63.5s per fathom; guaran- shift; surplus divided as above. Machine drifting was contracted 80 ft per mo; bonus of 5s per ft per mo in excess. At a LAKE (*E & M Jour*, 96, 1177), 2 men, working 24 shifts per mo, at $3.25 were required to break 900 tons per mo (18.75 tons per shift at f 10¢ per ton for output above 900 tons. Supplies not charged to ded 7 or 8¢ per ton, in which case excess was deducted from bonus.

5. Cooperative Systems

s is sometimes arranged by permitting employes to buy ny, their employer; additional incentive to participate in offered in form of a discount from market price of shares, a onus, or an installment plan of purchase.

on, in 1903, instituted a plan by which common or preferred shares oyes upon easy terms, with benefits beyond those enjoyed by ordi- mber of shares which can be subscribed for depends upon the

salary of the employe, but it is relatively greater for th
stock is the market price, or usually a little less. Payme
per mo for preferred, varying each year in accordance
$1.50 per share per month for common; maximum that
earnings. A premium of $5 for preferred and $3.50 for c
paid on each share of stock purchased by employes. N
canceled and the money paid in is refunded to the emplo
are not paid to employes who cancel their subscriptions,
pany's employ, and the forfeited or unpaid premiums ar
rata among the remaining shareholders, under this plan,
In case of death of an employe who has subscribed for s
this plan, his estate receives the unpaid premiums for the
share of the premiums undivided at time of death.

On Dec 31, 1912, more than 32 248 employes were st
aggregate holdings amounting to more than 125 848 share
1913 were: Employes earning less than $800 per year,
$2 500 per year, 36 557 shares; earning over $2 500 per
the corporation for conducting this employes' stock su
$1 000 000 per year.

On the Comstock Lode, in 1908, certain companies
ore produced each month, in excess of $50 000, among all
wages. Another company made similar division of all it

At an Indiana coal mine, a legal corporation of mi
owner then leased his property for an experimental peri
aiming to maintain safety of mine. Owner's royalty w
profits. The arrangement resulted in greatly increased
output during a dull season (8).

6. Leasing, or Tribute, S

Mining lease, as term is used in U S, is a simple
(mine owner) grants to LESSEE ("leaser," or "trib
tions, the exclusive right to do a certain piece of wo
to accept payment in form of a proportion of net
contractor, lessee thus shares risk as well as pro
A I M E, Vol 21, p 916). A clause called BOND AN
by which owner promises to sell his mine and del
lessee for a stipulated price at any time before term

Leases may be granted for exploring and developing
at Goldfield (9); to reopen a seemingly exhausted mine,
during the height of a mine's prosperity, as at Leadville, C
camps (11). In latter case, advantages of leasing are m
exists in form of stringers or pockets of high-grade ore.
profits under leasing system, while unable to operate pro
reasons stated below.

Advantages of leasing system, from OWNER'S
velopment accomplished at small cost, since lessee
a risk in performing dead work which a cautious mi
When developed, ore is produced with small wor
owner are absolutely net, since lessee usually pays f
ing, besides his own working expenses. (*D*) Les
tramming, and sorting, whether by carelessness or
liable for lien. (*F*) Lessee is responsible for dama
to workmen. (*G*) Slight supervision required.
advantages are: (*a*) He frequently has keener per
the particular deposit than even a skilful foreman
sometimes deliberately conceal information leading
(*b*) No heavy initial outlay required for machinery

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office force, interest, insurance (of property), and other cus-
es. (*d*) By more careful personal supervision, loss of time
ves and supplies are reduced, hence affording cheaper mining
usually attain on his own account. (*e*) Even under adverse
s returns are usually larger than ordinary day's wages (as
onsidering his extra diligence and skill); excessive dead work
nsated by an allowance for "footage" (say, $2 per ft), or by
ller royalty.

sually based on a sliding scale which varies with gross value
ount is computed on net returns from smelter or reduction
ng haulage, freight, and treatment charges (sometimes also
ing fees). **Shipments are ordinarily made in name of** mine
ducts his royalty, then his charges (if any) against lessee for
rendered, and finally hands balance to lessee.

first stages, royalty rate was 25% on ore over $50 per ton, and 20%
At one Colo mine (12) rate was numerically equal to ½ value of ore:
e, 50%; special provisions for $300 ore. At a mine in Gilpin county,
33 to 75%, if drift had already been run under leased block; 25 to
run by lessee. At Cripple Creek (14) customary rate is 25% on all
dville: under $10, 10%; $10 to $15, 15%; $15 to $20, 20%; over
matical discussion of a rational basis for levying royalties, see (15).

s of a mining lease. A lessee can void his contract at any
y, while a lessor has no means of ousting a lessee if the latter
he contract. For this reason, and because most lessees are
ng, it is particularly important that mining leases be care-
early understood.

st likely to cause trouble are: (*a*) What constitutes ABANDONMENT
sual stipulation is for a minimum number of days' work (200 man-
Cripple Creek; 60 shifts per mo, at Goldfield), or a minimum amount
00 ft per yr, at Leadville). Forfeiture results from violation of any
not attributable to casualties, strikes, or litigation) and owner may
session without formality of a lawsuit. (*b*) Ownership of PLANT
disposal of ORE-DUMPS remaining on property at expiration or for-
Goldfield, all improvements underground, and all broken ore not
on of lease, become sole property of mine owner. At Cripple Creek,
mps remain property of mine owner, but lessee must remove his
ays. It is advisable to include a definite stipulation as to the length
tion of lease, within which owner must market ore on dumps, or
ee's equity in that ore shall cease; neglect of this provision has been
wsuits. Additional stipulations, not already mentioned, relate to
of leased block. (*d*) Date of expiration of lease. (*e*) Work to be
to yield maximum output with due regard to development or ore-
th miners and workings. (*f*) Adequate timbering to be placed and
imum dimensions of openings: shafts commonly 4 by 8 ft, and
r. (*h*) No underhand stoping; no stoping within 10 ft of shaft; no
one another. (*i*) Workings to be kept clear of ore, rock, rubbish,
r or his agents to be admitted to mine at any time for any purpose.
essee to be settled promptly, and vouchers to be shown to lessor on
-leasing without owner's consent. For verbiage of typical Colo

7. Methods of Paying Wages

n pay-days. MONTHLY pay-days reduce amount of clerical
of pay roll, but often work hardship on individual miners;
commonest at mines which are remote from head office, and
ties. BI-MONTHLY pay-days are the rule in anthracite dis-

trict, and are common elsewhere; involve more cl
considered more equitable by the miners. WEEK
with companies having large clerical force, and si
ample banking facilities.

Wages are paid: (*a*) By check. This is ofte
avoids necessity for filling out individual receipts
cashing checks requires opening of a bank accou
sometimes involving a discount; method not reco
(*b*) In currency. This is preferred by miners, and
to operator is the risk of theft of cash during trans
orders on company store. In an isolated locality,
and when this belongs to mining company, paymer
may not be objectionable; elsewhere, the plan is li
may lead to serious discontent among miners; it l
thracite field, after long protest from the organize

Time-books, pay-rolls, etc. Where day's wa
is supplied with a TIME-BOOK, with columns in w
hours worked each day by each man under his cha
are so arranged that the foreman can distribute ti
to nature of his employment, for purposes of cost
unwise to demand too much clerical work from
once a week, entries in the time-books are transf
where a distribution to different working accoun
periods, PAY-ROLL is compiled from office time-boo
the total hours (shifts, or days) worked, rate per
different kinds of work by same individual), and
In case of contract work, amounts due will be com
veyors, foremen, weighmen, etc. At same time
charges of company against each workman, such
cash advanced, contribution to hospital and doct
check-weighman, and Union dues (in some case
propriate sources and entered. A copy of this p
in cash needed for following pay-day, as well as
several accounts, to balance deductions, is sent t
sends required amount to mine office. In meant
an individual "DUE-BILL" is prepared and deliver
amount he has earned, itemized deductions, an
These statements are preferably distributed a fe
opportunity for rectifying mistakes and for dividi
in case of a contract.

Anthracite agreements (Art 11) require a contractor t
due to each of his laborers, so that they can obtain t
If wages are paid in cash, it is convenient to have a cou
can be signed and delivered to paymaster, as receipt f
quired for above-described book-keeping, it is customar
ceding pay-day by an interval ranging from 2 or 3 days
before pay-roll is made up, his wages may be paid out
covered from wages account on next pay-day (17).

8. Accident Compensation a

Development and purpose. Under the com
sued for damages by an employe injured in his se
defenses of contributory negligence, negligence of

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With progress in the mechanical arts and in transportation, ly evident that these common-law defenses placed an em- ed disadvantage in a suit with his employer. As early as OYERS' LIABILITY LAW (State of Ga, and applicable only to the defense of fellow-servant. Other states have enacted r modifying the defenses of fellow-servant and assumption s were mainly restricted to railroads; in no jurisdiction were es entirely abolished with respect to all employments. Un- n injured employe was still under burden of proving per- part of someone as proximate cause of his injury, a proof ncreasingly difficult with the development of modern indus- hich "trade risks" are so prevalent. An injured employe ssity of incurring both expense of law-suit and enmity of his yer, on other hand, was liable to be held for excessive dam- conjecture of a sympathetic jury, or must insure his liability he rate of which was commonly based upon the worst rather he average conditions surrounding the employer's particular MPENSATION LAWS seek to diminish injustice to both parties gation to pay damages independent of fault (except criminal of injured employe); (*b*) making measure of compensation etermined (based on aver weekly earnings); (*c*) permitting is compensation liability at more equitable rates.

of compensation laws. DIRECT, or simple, compensation law demnity immediately upon person in whose employment an injury tate insurance, law requires employer to pay an annual tax graded his business and proportional to his pay-roll; an injured workman t of this fund by state officials. COMPULSORY compensation law on on employer, either to pay direct indemnity or to contribute to Nev). ELECTIVE law permits employer to choose whether to abide provisions, or to operate under a liability statute which deprives non-law defenses; employes are similarly required to state their ective system is to circumvent the usual constitutional provision of property without due process of law, which the N Y Court of an inherent defect in any compulsory compensation statute.

Existing compensation statutes have adopted practically a uni- sonal injuries from accidents arising out of and in course of employ- toxication or to wilful or intentional misconduct." Necessity of for accident, and degree of fault, is thereby eliminated, except in d wilful misconduct, different forms of latter sometimes being ex- atute.

sation. For TOTAL DISABILITY, 50% of aver earnings (65% in Cal riod of 240 (Cal) to 780 weeks (Wis), or until total reaches 4 to 6 es. In W Va, for permanent total disability, compensation runs TIAL DISABILITY, 50 to 65% of lost earning capacity, running for iods. Most statutes include schedules fixing amount of compen- uries entailing only partial disability. For DEATH, compensation ent may be a lump sum equal to 3 to 4 times annual wages of de- his aver earnings running for specified maximum period; to partial ion. There is wide diversity in statutory provisions on this sub- to make amount of compensation, in any case, mathematically

mpensation liability. In most states, employers are re- r liability, and in some are permitted to choose method of

atutes (Mich, N Y) 4 methods are allowed: (*a*) Self-insurance, on lvency. (*b*) Mutual companies. (*c*) Regularly organized

liability-insurance concerns. (*d*) Organization administ
injured employe usually has direct claim against insurer,
must be less than that stipulated in compensation act.
insurance is optional, and in others (Nev, Ohio, Ore, Wa
fund is compulsory. Objections to last-mentioned form
practice, are numerous (18). An important, but indirect
insurance is the influence that it exerts in promoting s
ployer's premium will properly be in direct proportion to
some states (Ariz, Cal, Minn, N Y, Wis) employer is
arrangement with his employes as to manner of adjusting
vided the statutory amount is not diminished; in others
Wash, W Va) all such arrangements are forbidden.

Other provisions of compensation statutes. As leg
velopment and adjustment, it will not be useful to dig
statutes. Full information as to any particular state ca
tion commission," "industrial board," or analogous office

9. Pensions and Benefit Funds (See also

Pensions are granted by many of the most pro
from a fund to which employes may or may not be a
FUNDS are more frequently maintained by the em
assistance from company's accounting and clerical f
pany also contributes financially.

U S Steel Corporation, in 1910, added $8 000
$4 000 000, to establish an old-age pension system,

(*a*) Compulsory retirement for men at 70 yr of age and
20 years' service. (*b*) Retirement at request of the empl
the age of 60 for men and 50 for women, after 20 yr of se
of permanent total incapacity, after 15 yr of service.
each year of service, 1% of the aver monthly earnings f
Credit for service rendered to any of the plants of the s
poration or to the predecessors of such companies. (*f*)
max pension, $100 per mo. At end of 1912, the corporatio
during that year, $358 781 was paid in pensions (aver,
other assistance. Average at which pensions were gran
yr. Cost of administering the fund was 3.12% of disbu

D L & W RR admits certain outside employ
to participation in the pension fund originally devi
Employes 70 yr old, and in continuous service fo
pensioned. Employes 60 to 70 yr old, and 25 yr i
pensioned, if incapacitated. Cases of incapacitat
alleviated at discretion of pension board. Monthl
ice × 1% of aver monthly earnings during previous
for pension exceeds allotted appropriation, all allo
CLEVELAND-CLIFFS IRON CO (19) has adopted pra
minimum pension of $15 per mo; to widow of a
$12 per mo for 5 yr, and $1 per mo additional for ea

Rogers-Brown Iron Co. Each employe pays
which company's cashier is president, with mine ca
officers. Death benefit, $300; for incapacity, $1 p

Homestake aid fund. Each employe pays $1
tributes $1 000 per mo. For death or total dis
partial disability, $1 per day for max of 6 mo (21).

Copper Queen benefit fund (22). Employe
company pays 4% interest on this fund and also co

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numbers one-half the whole working force, or $25 000 if it ths. Benefit at death from sickness, $100; from accident, as admitted after 45 yr old, latter benefit is only $100. For dent, 0.5 wages, for max of 52 weeks; sick allowance, 0.5 day, for max of 52 weeks.

often maintained at collieries. Miners collect and bring up their company provides storage for them, and periodically sells them to about 8¢ each, the money being used for sick benefits and funeral ex- L & W company pays out, from this source, about $1 700 monthly. see Bibliog (23).

10. Special Problems of Mine Labor

24) supply nearly all the unskilled labor in South African rable workers come from the milder and healthier climates, guese East Africa; those from hot sections of Central Africa, higher altitude of the Rand, develop abnormally high rate ing 164 per 1 000 per year). Kafir labor is recruited by arying with conditions, averaging (1908) $11 for 6 mo; $20 yr; about 0.75 of this labor is recruited by an association of Recruits are subjected to medical examination, contracts ards are issued to them, and their desertions from assigned by government. Max contract is for 1 yr, with permit to BACKS to Kafir labor are: (*a*) incapability of sustained phy- of ambition; bonuses or high wages simply make them will work at mine only long enough to satisfy present needs, ges; (*c*) susceptibility to pulmonary diseases, aggravated ing. At hand drilling, minimum day's work is 24 in; con- n. More experienced "boys" operate air drills under super- 2 drills requiring 4 or 5 "boys" and 3 drills, 6 or 7 "boys." vise 20 to 40 hand drillers. For Kafir wages, see Table 2. Kafir is about 8¢ per shift, cornmeal being his principal food.

and (25) were imported to supply deficiency in Kafir labor at close Coolies were indentured before leaving China; none was allowed act for his services; they were confined exclusively to the Rand dis- upation was strictly defined as unskilled; their residence was rigor- overnment. Contract period was for 3 yr, with option of renewal, tical pressure repudiated this option and all Chinese were sent home. red to house Chinese in sanitary manner, provide free medicine and nd food in following ration per man per day: rice, 1.5 lb; dried or lb; vegetables, 0.5 lb; tea, 0.5 oz; nut oil, 0.5 oz; salt; total cost, inese was about same as that of white men and about 0.5 that of practically immune from pneumonia. Cost of operation with her at first than with Kafir, because of expense in preparing addi- nodations, but towards end of 3-yr period, efficiency of Chinese s.

ormerly the Mexican miner was of a distinct class, familiar nerations; recently, many mines have had to employ agri- ho, during planting and harvesting seasons, abandon the EONAGE is now theoretically abandoned. Principal DRAW- labor are: (*a*) Inefficiency, due to low vitality, unhygienic liquors, warm climate and high altitude in some localities; serving numerous fiestas or saint's days. While a Mexican essive loads on his back or shoulders he is often inefficient as d shoveler, and is usually unskillful as a mechanic. BEST an labor are attained by: (*a*) placing as much work as

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mine officials and diminishing the number of complain
which was felt to be too remote and deliberate a bod

Bituminous interstate joint conference, for
conditions, has been in sustained operation in the
territory (Ohio, Ind, Ill, and Western Pa) since 18
(Mo,Kan, Ark, Ind Ter, Tex) since 1904 (29), duri
ditions have there become firmly established: (*a*)
and authority of union (U M W) to negotiate agre
are recognized by employers. (*b*) Basing point
compensating operators in one state for natural
character of coal seams, etc) enjoyed by those in a
2 000 lb adopted. (*d*) Both screened-coal and ru
authorized. (*e*) In former case, standard screen
spaced 1.25 in apart, is stipulated. (*f*) Different
chine and pick mining, between puncher and chai
thick and thin coal seams. (*g*) Uniform workin
is adopted for all classes of mine employes.

Conferences are held in Jan every 1 or 2 years, at a
tended by delegates from miners' unions (usually nat
and by representatives of coal mine operators selected
in southwestern field and in Ill maintain organizations.
etc, comprise 2 members of each side from each state;
bers of each side from each state. On ballots, each sid
imous vote (32 in central competitive conference) i
Miners' demands, having usually been formulated at
tion of U M W, are presented in form of resolution, w
acceptance. Resolution then goes to scale committ
agreement is seldom reached on any clause, and being
tion is further debated and generally fails to pass. It
mittee, composed of 2 members of each side from ea
strictly secret. At this point, a counter-proposition
and argument then ensues until a compromise is effect
is then approved by scale committee, and is finally ad
signed by 2 representatives of each side from each sta

Consistent TREND OF MINERS' DEMANDS has been t
ment for coal in all states within central competitive
mine basis alone (this provision is already in force in
pick-mining over machine mining. (*d*) Uniform wag
mission of Iowa and Mich to interstate conference o
(opposed by operators of Ohio, Ind, and Pa); and of W

Bituminous state agreements (29) are in fo
the interstate conferences, and also in many oth
mining territory of the U S being governed, as to
by formal contracts between operators and or
ventions follow the interstate conferences and a
they are attended by officials of U M W and by

In those states in which a basing point has been d
principal work of convention consists in establishing
ranging both above and below the basic rate, and so a
trict (those who break down or produce coal) shall r
same returns for a day's work. Factors involved in
of-mine *vs* screened-coal basis of payment; differentia
times determined by practical test on 1.25-in screen) v
machine or "shooting off the solid") and powder empl
Ohio and Ind, present difference is about 25¢ per ton
consistently advocate run-of-mine basis; operators opp

explosives and diminishes proportion of higher-priced lump coal. mining; bonus in favor of former is intended to compensate for ous nature of operation, but miners constantly advocate raising the or 2 reasons: (*a*) To discourage use of machines and thereby add nployed. (*b*) Chain undercutters produce less slack coal than pick dd to operator's profits, to a share of which the miners consider (*C*) Punchers *vs* chain cutting machines. (*D*) Thickness of seam; thin seams, to compensate for smaller tonnage per unit area under- ENTS effected by state conventions relate to: (*a*) Prices for narrow . (*b*) Wages of inside workmen other than miners; in central com- e wages are practically fixed on a uniform minimum basis by the and are accepted, usually unchanged, by state convention in that of outside workmen, over which the interstate conference, as yet, (*d*) Frequency of pay-days, and details as to issuing of wages, Hours of labor; an 8-hr day is generally considered to mean 8 hr exclusive of lunch and traveling time. (*f*) Provisions for timber- sposal of mine refuse, removal of waste from loaded coal, etc. (*g*) shots; generally stipulated that practice must follow legal require- (*h*) Disciplining of miners for sending out inferior coal, for ignoring absenting themselves on working days, etc; in this connection, the s its support and coöperation in the enforcement of discipline. (*i*) nt of disputes; this is closely analogous to the present practice in ee Conciliation Board, and Grievance Committee, above), except a conference between officials of the company and state officials of ll agreement, duties and privileges of "pit committee" are clearly unnecessary officiousness on their part, and interruption to work- "Check-off" system, withholding Union dues or fines from a work- ally accepted by the operators in states where agreements are in quality of powder, oil, tool sharpening, etc.

WELFARE

12. Wash and Change Houses

usually provided with bathing facilities, have become a at all mines employing more than a few men. Some com- ed to incur expense of such building, through apprehension ould be abused or unappreciated; statistics covering several ovided with such facilities indicate that 85% use the wash se of change and wash house is commonly offered gratis, but that the levying of a small charge, 50¢ per mo, induced better benefits afforded.

operating details (30). In designing a composite change lowing points deserve consideration: (*a*) Should be situated enient to mine opening. (*b*) Structure must be fireproof, tile, concrete, or sheet steel on wood frame. (*c*) Floor space, nly, should be 15 to 20 sq ft per man to be accommodated. e walls free from obstructions, the former preferably of con- per ft towards drain, so that entire interior can be washed ckers are used, they should be not less than 12 by 16 in by 3 metal construction and provided with combination instead ould slope steeply, for ease of cleaning, and to prevent piling (*f*) Drying racks standing on floor, or hooks and lines sus- spaced 30 in apart in ceiling, are recommended for hanging uld not be used for street clothes; if lockers are used for mine give every man 2 lockers. (*g*) Ample heat must be pro-

vided for drying mine clothes; usually in steam co or drying racks. (*h*) Adequate ventilation is esse mally oppressive; natural ventilation through cu artificial exhaust has sometimes been installed. be separated from change room, preferably by a s door at each end. (*j*) Shower baths most advant also be provided; tubs and swimming pools are u shower booth, 4 by 5 ft, for every 20 lockers is con all use wash house at same time. (*k*) About 40

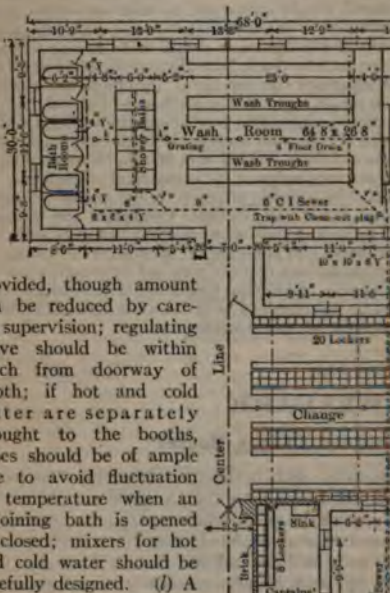

Fig 1. Change and Wa

provided, though amount can be reduced by careful supervision; regulating valve should be within reach from doorway of booth; if hot and cold water are separately brought to the booths, pipes should be of ample size to avoid fluctuation in temperature when an adjoining bath is opened or closed; mixers for hot and cold water should be carefully designed. (*l*) A capable attendant, constantly on duty, will readily justify his wages. TOILETS, specifications for, see Art 15.

Cost of erection. Vulcan mine, Mich: brick, 252 men, cost $6 604. Aurora, Nev: wood and accommodating 200 men, cost $3 819. East Vul 81 by 29 ft, for 250 men, change house mainly, wit built on company time, cost $10 325. Baltic min men, $20 000 complete.

Detection of "high-grading." Stealing of rich ore source of loss at Bendigo, Kalgoorlie, Kimberley, Crip elsewhere; rich ore is brought out, a few lb at a time, in other schemes, and is sold to illegitimate assayers or t small worthless mine simply as a blind. Courts and

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in face of lax moral sense in most miners' communities. Only o require all miners to change clothes as they come off duty, pref- othes in one room and street clothes in another. This procedure pposed by organized labor, and led to a strike at GOLDFIELD in 1907. re effected on following basis: Each miner was allotted 2 adjoining nine, working clothes were removed and locked in one locker; other and street clothes put on; company's time-keeper and other desig- l to be present, but no strangers.

Mine Communities and Miners' Dwellings

dicious planning of mining community, to promote health- and convenience, is as important from the financial as from of view; a healthy and n is both more efficient TOWN SITUATED NEAR llowing advantages (31). valk, entailing less loss of work, a serious factor in ricts, and less exposure in k. Having homes, stores, oximity is convenient for onsible officials are within nes. Where mining com- nsite, proximity to mine economies in cost of haul-

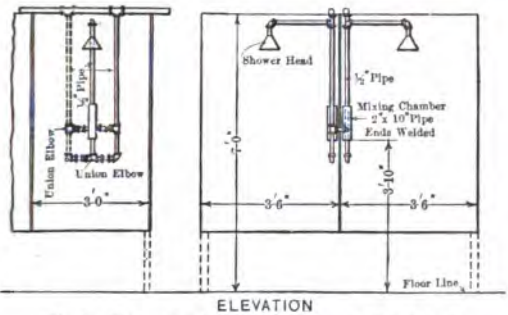

Fig 2. Fittings for Shower Bath, Tenn C, I & RR Co

t is genuine asset. Permissible distance from mine depends nveyance; electric cars in many districts have greatly en- daily travel; some mines operate work trains, and some special cars at reduced fare for mine workmen.

ownsite, the following features deserve attention, though all may) Accurate contour map is indispensable for efficient planning. (*b*) s should be approx parallel to contours, involving less expense for ers, and surface drainage. (*c*) Narrow but well-surfaced street, alks, is more suitable for requirements of ordinary mine town than dition. (*d*) Trapezoidal house lots, fronting on a diagonal street, ght to each house than rectangular lots with houses all at same dis- Deep, narrow lot involves less expense for street improvements ilization of land than a wide shallow lot of same area. (*f*) On

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closely spaced, deep lots, houses should not all be at s
houses should be farther back than single-story. It is
story houses. (*g*) Houses on both sides of a narrow st
one another. (*h*) Along a contour street on a hillside,
down-hill side and should be as narrow, from front to ba
hill side may be graded up to level of street, by setting
instead of at front of lot. (*i*) Houses on steeply slop
usually inconvenient. Steps and verandas of hillside ho
at front or back.

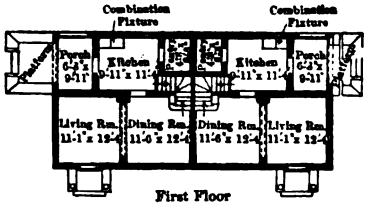

Fig 3. 2-Story, 2-Family, Concrete House

Principles of design. Advisable for a compan
house plans for: variety of appearance, better u
rental. Number of houses of each size can be est
families in neighborhood. Most workmen's house
of foreign workmen in U S use all but 2 rooms o
being most important room in workman's home, s
accepted, and situated on pleasantest (south or e
more than 4 or 5 rooms should be of 2 stories, to r
tion and permit better lighting.

Materials of construction. WOODEN FRAME house
variety in design more easily attained; (*b*) more readil

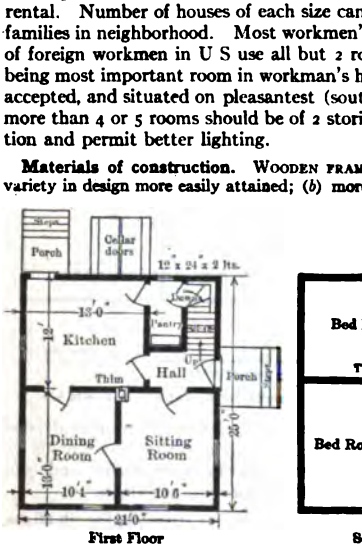

Fig 4. 6-Room House, Trimountain Mine, Mi

Nanticoke, Pa), has advantages of being durable, firep
vermin proof. Unless precaution is taken to leave ai
condense moisture. MOULDED CONCRETE BLOCKS can
tailings. At Mineville, N Y, blocks 8 by 10 by 20 in (
make and 10¢ for hauling and laying. Variety of appe

t coarseness. BRICK excellent for better ply obtainable.

L. WOOD FRAME: , 2-story: 6 rooms, 1 300; cellar, $200 ral, cost of frame f volume measured ar to ridge-pole, is ; costing $800 to 1 ft. Frame dwell- , 5 to 9¢. Frame o to $10 000, 9 to .5 to 3¢. On basis f ordinary dwelling s approx: 1-story, ory, $2.50 per sq ft. ¢ per sq yd. BRICK rooms, 10 to 14¢ E-BLOCK houses at t about 10% more s of same size, or POURED CONCRETE City," Pa: 2-fam- ost $3 600 to $4 000.

ty of equally desirable cable to include here or construction; see

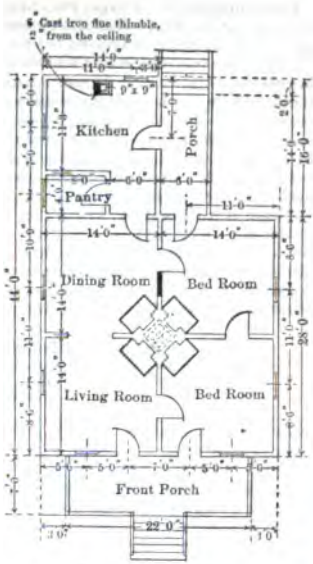

Fig 5. 5-Room, 1-Floor House, Tenn C, I & RR Co

Purification of Domestic Water Supply

mestic water (33) are not all equally objectionable; prob- which are physiologically wholesome, or could easily be made purely æsthetic reasons. Most objectionable features are ste, odor, and pathogenic bacteria; ordinary hardness is ndry and boiler supply, but not hygienically. TURBIDITY, ver water, can be removed by storage, coagulation, or filtra- aracteristic on water-sheds comprising swamp areas; it is composition of vegetable matter in bed of reservoir and by ater surface; it is seldom deleterious, but may become tem- at time of spring and autumn "overturn," when bottom y charged with organic extracts, mixes with whole body of Color may be partly corrected by filtration, and entirely ical means. TASTE results from same causes as color and iron in excess of 1 part per million; iron can be removed by filtration. ODORS arise from same causes as color, and from and industrial wastes; they can be corrected by storage, l methods; except for its offensiveness, water having tastes duce only negligible physiological effects. HARDNESS is y," calcium or magnesium carbonate held in solution by pres- cipitated when latter is eliminated by boiling or by addition permanent," mainly sulphates and chlorides of calcium and more elaborate chemical processes for their removal.

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Contamination by sewage, from hygienic standpoi
of itself, but possibility is always present that domes
germs, notably those of typhoid fever and cholera.
by water, are less tenacious of vitality than BACILLUS
in enormous quantities in human intestines and in t
stantial absence of bac coli is considered a sure indica
present. Presence of CHLORINE, beyond that normally
seashore or salt-bearing springs or strata, is strong pres
by sewage. NITROGEN in various compounds may re
but is generally considered proof of sewage pollution,
indicates the probable length of time elapsing since in
virulence of pathogenic germs can then be deduced,
matter are: albuminoid ammonia, free ammonia, nitr
portion of albuminoid ammonia in a colorless wate
nitrites, free ammonia, and chlorine occurring togethe
pollution by sewage. Following figures, in parts pe
Health, 1913) show difference between pure and pollu

Potable and Polluted Waters of

	Ammonia	
	Album	Fr
Boston; surface origin	0.0145	0.0
Springfield, surface, filtered	0.0080	0.0
Lowell, tubular wells	0.0064	0.0
Blackstone river, at Worcester	0.1286	0.9
Hoosic river, below North Adams	0.0489	0.0
Merrimac river, above Lawrence	0.0224	0.0

Interpretation of nitrogen determinations sh
chemist, and should cover a considerable length
member that principal purpose of these determin
of presence of pathogenic germs.

Total number of BACTERIA is determined by micros
note being made of the number of times bac coli is ob
Bac coli may appear in 1-cc as often as once in 20 time
ing apprehension of typhoid, but the possibility of typ
water must always be considered. In 1912, N Y wa
every ten samples (1 cc) but death rate from typhoid
same year, bac coli was found once in every 100 sam
Bacteria, both pathogenic and harmless, can be nearl
tion, and completely by sterilization with ozone or hy

Storage serves 2 purposes (*a*) equalization of
discussion of storage requirements under differen
and consumption, see Sec 38, Art 20, 21; also (3

Opinions differ as to necessity for stripping all veget
under construction, but this precaution probably dim
odor. Reservoir should preferably be narrow and de
so that fluctuation in water level between spring and
growth of weeds. Supply from reservoir should be
excess water can be wasted, it should be drawn from

Purification is effected by: (*a*) giving sedime
by coagulant) an opportunity to settle, taking m
diminishing bacterial life by action of sunlight a
completely devitalized by storage in open reservoi
by action of wind, and color by action of sunligh

a drawback to extended storage, although their effect is limited to tion of color and odor; algæ can be prevented by treatment with o million parts of water), or, in case of small reservoirs, by covering

olished by fountains, cascades, or flowing over dam, is useful also, when followed by filtration, for removing iron.

valuable help for curing turbidity and diminishing bacteria, orage or filtration. Coagulant commonly used is alum, 1 gr eam of water entering first storage reservoir. Coagulating uires presence of lime, which must be added if not originally .

ss (35) can be reduced by addition of slaked lime in quantity suffi- CO_3, which holds $CaCO_3$ and $MgCO_3$ in solution (say 1 lb per 1 000 lness). Water which is both turbid and hard, requiring an exces- an be more cheaply clarified and softened by lime.

hen properly conducted, will remove all turbidity, practically than one-half of color. Two general methods are: (*a*) slow pressure or mechanical filtration.

truction: a shallow reservoir, with water-tight sides and bottom, re advantageously of concrete, sloping towards one corner. Area to 40 000 sq ft; small bed requires more frequent cleaning, but can cted against freezing; advisable to build at least 2 beds, for alter- ing system of tiles with open joints is laid down, leading to dis- e next covered by 6-in layer of coarsely broken stone; 5 successive and screened stone or gravel, of diminishing coarseness, are then a layer 30 to 40 in deep of sand. Best sand for upper layer is of 3 to 0.4 mm diam, and of nearly uniform size. Water to be filtered voir to depth necessary to cause percolation at the desired rate of descending velocity; the slower the percolation, the more complete . When upper layer of sand becomes clogged, its surface is scraped of 1 or 2 in; when thickness of sand remaining is reduced to 12 or e added or the removed portions may be washed hydraulically and gging of sand filter increases in proportion to amount of turbidity h excess of algæ; in extreme cases, desirable to apply preliminary filtration, or coagulation.

rs are more commonly used after preliminary treatment just nitting water under pressure up to 10 to 12 lb per sq in, per- e rapid, up to 150 in descending veloc per hr, and, if properly efficient. Installation of mechanical filters should be super- n hydraulic engineering, and their operation must be con- eful control.

is most advantageously applied to water which has pre- d by sedimentation, coagulation, or filtration, for purpose of against pathogenic bacteria. Two substances most com- OCHLORITE of calcium ($CaCl_2$, containing about 30% Cl) and ; in either case, usual proportion is 0.3 to 1 part free chlorine The $CaCl_2$ must first be emulsified with water, and kept in utomatically into stream of water to be treated; liquefied ottles, costs 8 to 10¢ per lb, and requires specially designed mployed for disinfection in Europe; mechanical difficulties d its installation requires service of an expert; has not yet with chlorine in U S.

mp service, a useful device (38) is a water-tight canvas bag, 20 in aving loops at top for suspension and one or more nickled f-

at bottom. $CaCl_2$ is put up in sealed glass tubes holdi
chlorine). One tube will kill all pathogenic germs in 3
is safe to drink in 10 to 15 min. Desirable to filter wat

Boiling is last resort to insure sterility of d
germs are killed by boiling for 5 min; some spor
higher temp, but it is not certain that such germs
water can be improved by aeration.

Distilled water, though perfectly safe pathogenic
for continuous use. A still can readily be improvised f
pans, preferably tinned on surfaces which come in cor
can be heated by live steam, where available.

15. Sanitation

Wastes requiring disposal are: garbage and
water. Under favorable conditions, such as:
topography, small opportunity for stagnation o
tion, cool, dry climate, and water supply obtaine
against contamination, no serious difficulty is lik
known and archaic methods. In absence of any
demands attention, and may become an enginee

Garbage should be placed in covered and water-tig
a week in covered-wagons. It may then be buried in
tance from town, or incinerated. Usually, enough
provide fuel for latter purpose. Descent of garbage
retarded by baffle rods, giving it a chance to dry part

Excreta. Only 2 methods of disposal should
munity: dry closets, or a system of sewerage, in
system is expensive to install but cheap in oper
a communal sewerage system is impracticable,
be inadequate for flushing; (*b*) extra liability of
the maintenance of both water mains and sewers
of workmen induces expensive repairs.

Dry-closet, an improved form of privy, may be the
many mining towns. Where water supply must come
N Y (40), the old form of earth vault must be abando
tion: (*a*) Receptacle, galvanized iron, preferably cylin
water tight. (*b*) Seat should be only slightly higher
seat so designed as not to permit it to stay open, to ad
through door in back, so designed as to fall tight
through screened aperture. (*f*) Door and window sc
ville, N Y, $15 for structure, $1 for receptacle.
weather, twice a month in winter, full can is removed,
cleaned, empty can is returned to place. Full cans ar
along alley in rear of houses, and taken to disposal g
or incineration. Cans are emptied, washed, and dried
tor for disposing of excreta from 238 families cost $
averaged $1 per family per mo. Same plan has been
I & RR Co) and elsewhere (41).

Sewage disposal (42). Where water supply i
of laying sewers is not prohibitive, a system of se
age can be disposed of by dilution, irrigation, or

Dilution. If mixed with sufficient volume of
decomposes into solids and gases, under comb
oxygen absorbed from atmosphere.

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eeds 6 cu ft per sec per 1 000 persons discharging sewage, objec- stream are unlikely to result; this refers only to creation of a an that a stream so polluted would be safe for water supply. If e between seasons, or if its water has opportunities to stagnate, may arise. Roughly speaking, a stream will purify 0.02 of its t 0.05.

le sewage has been practiced for centuries in Europe, and n many parts of western U S. Prime requisite is a porous, l in topography; less porous soil requires preliminary re- n sewage, by sedimentation or septic treatment (see below).

underdrained, preferably by tile pipes, 3 to 4 ft deep and 30 to 50 acre will absorb raw sewage from 300 persons, or 30 000 gal per ranged as to take sewage in rotation; it commonly flows for 4 to east an equal period for drying. Berries or salads, eaten with- e raised on sewage farms.

nvolves 2 steps: (*a*) putrefaction, in absence of oxygen, ids are disintegrated and reduced in volume, and most of decomposed into NH_4OH, CH_4, and CO_2; (*b*) oxidation, resence of air, whereby NH_4OH is converted into HNO_3, mbines with alkalies, normally present in sewage, to form

upon bacteria, and considerable delay is sometimes experienced comes well established. Putrefaction is stimulated by temp of of acidity; nitrification proceeds best under following conditions: forded by intermittent flow through porous bed; free ammonia esence of alkaline base, preferably lime and magnesia, but not in final products; temperature, the warmer the better. Putrefac- re conducted in "septic tank," the overflow (effluent) of which ification on natural soil, or in artificial filter or contact beds.

nall system (43) is shown in Fig 6. Capacity of sedimen- be equal to a minimum of 6 to 8 hr flow of sewage, while city is bac- led by mposi- should amber nce of possi- ntains er, as pre- nber is

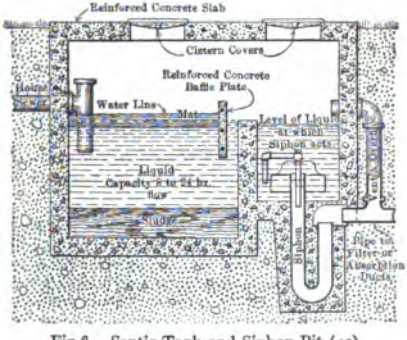

Fig 6. Septic Tank and Siphon Pit (43)

t most ic solid ewage, o tons er) per s removed, at long intervals, by bailing (or through blowout, in e installations), and disposed of by drying, burying, or burning. all. Second compartment of tank contains some form of auto- discharge entire contents of that compartment at intervals of 4

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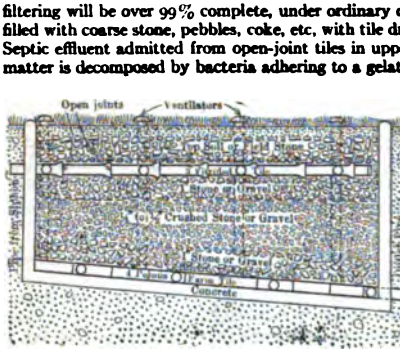
Nitrification. Effluent from septic tank may be on porous sandy soil. SAND FILTER BEDS, where n trenching (4 to 6 ft deep, 50 ft apart) and tiling an ar contributing sewage. Surface requires raking from bution of effluent over whole area. Where soil is sembling water filter (Art 14) can be made, preferably radiating open-joint tiles for discharging filtered se coarse gravel, 12 in fine gravel, 12 in coarse sand. 400 000 gal per day. Average cost of maintenance treated. With sand filters, intermittent treatment obtain their necessary oxygen. In cold climates, filt covered with boards and earth, with ventilators. filtering will be over 99% complete, under ordinary filled with coarse stone, pebbles, coke, etc, with tile d Septic effluent admitted from open-joint tiles in upp matter is decomposed by bacteria adhering to a gelat



LONGITUDINAL SECTION AT CENTER

Fig 7. Percolating Nitrific

depth is 3 to 4 ft; volume 1 to 1.5 times daily flow. tion, bed is allowed to fill for 1 hr, stand full for 2 h 4 hr, making 3 cycles per day. PERCOLATING BED bed, but flow of septic effluent is continuous; in smal joint tiles (Fig 7); in larger ones through variety of 4-in broken slag, coal, or cinders; depth 4 to 5 ft. than contact bed, and will dispose of over twice as m is more efficient than other types in removal of bac wells are closely adjacent, concrete walls and botto made in form of a long trench, with a line of open-j

Toilets. Following standard specifications age systems have been adopted by the U S S sanitation:

(*a*) Closets should be located as close as possible number of small installations rather than a few lar open to outside light and air. Minimum window s fixture must be 4 sq ft, and for each additional fixtur structed that they can be opened. Each toilet roo floor space and not less than 100 cu ft air space fo should be separated from lockers and wash rooms by wash rooms are in close proximity to closet, each cl one wash basin. (*f*) Adequate number of urinals sh Number of seats, one to every 15 persons, based upo shift in departments using the unit. (*h*) Closets sh made of porcelain or vitreous china, not of enamel separately vented and trapped. (*j*) The seat of e

absorbing material and finished with varnish or other substance ervious to water. In no circumstances should seats made of enlain, or other similar heat absorbing substance be allowed. (*k*) ast 7 in wide and 11 in long. (*l*) There should be partitions beof enameled steel, not porcelain, 6 ft high and 12 in off the floor. artitions should be not less than 30 in, and from front of seat to (*n*) Partitions and bowls should be so arranged that entire space at can be easily cleaned. (*o*) Doors should be not less than 20 in on, lower edge of door should be 24 in above floor. (*p*) Clothes d. (*q*) Toilets should be adequately heated in cold weather. (*r*) l be so arranged as to permit proper cleaning of floor and walls. nstructed of glazed tile or concrete, with a smooth surface; should Walls should also be made of non-absorbent material with a ly glazed, not only to facilitate cleaning but to avoid defacement; rs. (*u*) Walls and partitions should preferably be light colored. ould be provided. (*w*) Toilet paper with proper holders should ny. (*x*) Regular and thorough cleaning should be adopted; dislied upon.

ets, see Sec 23, Art 8.

seases Encountered in Mining Profession

an intestinal disease, is transmitted by germs contained in survive for long periods in mud and water underground. ohibit indiscriminate pollution of mine workings; (*b*) wash before eating. Symptoms and treatment, see Sec 23, Art 7.

ion of the lungs, most strikingly developed in gold mines of uced by breathing dust from quartz vein matter; dust of ctically all rocks not having a vitreous quartz nature, has effect. Serious danger arises through increased liability to onary diseases, even where acute silicosis may be absent. lopt wet drilling methods, or (*b*) require workmen to wear ms and treatment, see Sec 23, Art 7.

valent in tropics, especially in low and damp localities, is of mosquito (genus anopheles) which can be recognized xperience.

umulative, and a sufferer usually has sufficient warning to enable er climate. Regularity in recurrence of chills is not so characteris- s of other types. Recognized preventive is quinine, which should oing into a malarial district; sulphate of quinine is more insoluble ive than bisulphate or hydrochloride.

nine before each meal; avoid constipation, by fruit diet or laxaild houses on highest ground available, remove foliage in immediefully, and drain mosquito-breeding pools if possible. SYMPTOMS: s accompanied by violent vomiting; in former case, induce active monade, acetanilid or other antipyretic; in latter case, empty the cac and much tepid water, followed by 5 gr sod bicarb. TREATne 3 times a day; more quinine than sufficient to induce dizziness ote movement of bowels by 10 gr calomel mixed with equal weight of mixture every 20 min. Six hours after calomel, take 1 oz ater, followed 2 hr later by 20 gr quinine. Patient should remain aking 30 to 35 gr quinine per day, in 2 or 3 installments, and conss without return of fever.

ntestinal disease, is transmitted by a bacillus in food and hus) probably not by contact.

ubtful water should be boiled, or distilled, and milk sterilized on fuse should be incinerated, especially excreta from a typho

patient, all of whose clothing, bedding and utensils r
individual cleanliness, particularly in connection with
important. SYMPTOMS: headache (frontal), chilliness
followed by dry hot skin and great thirst; tongue dry,
pulse weak and rapid; later, pale red isolated spots,
abdomen; fever 102° to 104° F, usually entailing cor
MENT unknown. Liquid diet, principally milk, follow
second week; stimulants if necessary. Cold water or te
Cool baths or sponging with alcohol, or wrapping in we
except in case of complications or weakness.

Yellow fever (45), usually an epidemic, is an
likely to attack newcomers into countries where
temp, low barometer, lack of cleanliness, and imp
ally brings immunity.

Symptoms: invasion usually sudden, denoted by
headache, pain in back, constipation, and vomiting.
last 2 to 6 days, when convalescence may begin, or t
marked by vomiting of disorganized blood (black vom
urine and may exude from mouth and nose. Stupor
Recovery from third stage is rare. TREATMENT: diseas
tine and thorough disinfection are necessary. At begi
tard footbaths advisable. For fever, cold sponging an
in small quantities at frequent intervals. Black vo
stomach, counter-irritation of abdomen and fixing musc

Cholera (45) is an epidemic caused by bacillus i
enter with food or drink. Contributing causes: g
heat, and lack of cleanliness.

Symptoms: invasion is always sudden, consisting
muscles of abdomen and calves of legs, with coldness
Diarrhœa is pronounced, becoming more frequent and
fluid. Temp sub-normal, pulse weak and rapid, draw
tially or wholly suppressed. Duration of disease may l
cases death usually ensues on second or third day. T
of opium and tannin, with stimulants; wrap in hot bla
disinfectant solution. Diet, thin porridge exclusively
nursing necessary. Isolation and disinfection of deject
area should use extreme care about cleanliness and mod

Scurvy (45) is a derangement of the blood, most co
plorations, supposed to be caused by lack of fruit and
Indicated by blood clots under skin, ulcerations on s
swollen and ulcerated. Hemorrhages frequent, fever
be modified by increasing ration of fresh vegetables o
particularly beneficial.

Tropical complaints, in general, may be alleviated
gestions (46): (*a*) Before making a first visit to tropics,
with chronic weakness of stomach or liver are most li
(*b*) On arriving in a malarial district, begin taking qui
and prophylactic effects (Warburg's Tincture, liquid
inquire as to quickest route to higher and healthier loca
available spot; raise a few feet from ground; cut grass
vicinity, and screen houses against all insects. Aroun
and other breeding places of mosquitoes. (*d*) Diet sho
of fruit and vegetables, fresh if possible; meat may be
killed meat should be secured at any reasonable cost; s
fiable expense where practicable. Variety of food and
ries are important, because appetite is likely to be weal
distilled; better be a live crank than dead by neglect
liquors entirely; this is particularly important in mal

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ic action of the blood. (g) Protect top of head and back of neck ; arrange working hours to permit siesta during middle of day. t-weight clothing to permit frequent changing, and do not allow he body. (i) In short, the care of one's health, under conditions unsalubrious, and where failure of an entire undertaking may re- one individual, deserves more serious attention than it usually e zone.

ature as quickly as possible around limb above the bite. Cut flow of venous blood from the bitten spot, aided by sucking, if done safely unless mouth has an open sore. Apply strong solution nate, or bichromate. Purpose of alcoholic liquor, usually recom- heart action and counteract narcotic effect of the poison.

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SECTION 23

R, HYGIENE, EXPLOSIONS, AND ACCIDENTS

BY

GEORGE S. RICE

EF MINING ENGINEER, U S BUREAU OF MINES

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parentheses refer to Bibliography at end of this section.

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1. Atmospheric

Table 1. Composition of Pure, Dry,

	Symbol	By volume	By
Oxygen.....	O_2	20.93	2
Carbon dioxide....	CO_2	0.03†	
Nitrogen*.....	N_2	78.10	7
Argon.....	A	0.94‡	
		100.00	1

* Includes small amounts of rare gases behaving narily grouped under N.

† Haldane (*Jour of Hygiene*, Vol 2, p 241) state above the ground, as varying from 0.025 by day to In large towns there is more CO_2. In a London fo

‡ According to Sir William Ramsay.

Weight of dry air. At sea-level press (14 760 mm or approx 30 in merc barom; weight and 0.070914 lb at 100° F (Sec 39, Art 9). laws of gases. At constant temp its density

Humidity. Aqueous vapor is always prese tity depending on climatic and local weather saturation, except in arid districts, is rarely les

Presence of moisture in a given space is indepen that space. Therefore it is not technically correct this is the common mode of expression, and in mi the ventilating current or the mine air has a certai tain wt of water vapor per 100 000 cu ft of air in t ture present is chiefly dependent on temp; press an the temp the greater the quantity of water vapor,

Absolute humidity is the quantity or wt of aque Aqueous vapor is saturated when, at the vapor pre quantity. If vapor is partly saturated, the percent HUMIDITY, which is the ratio of the wt of vapor pre It may be measured by hygrometers (psychrometer termed "dry bulb," the other "wet bulb"; latter h kept wet. Evaporation by air current or by whirlin thermometer in geometric proportion to decrease of

Dew-point is the temp at which drops of visible

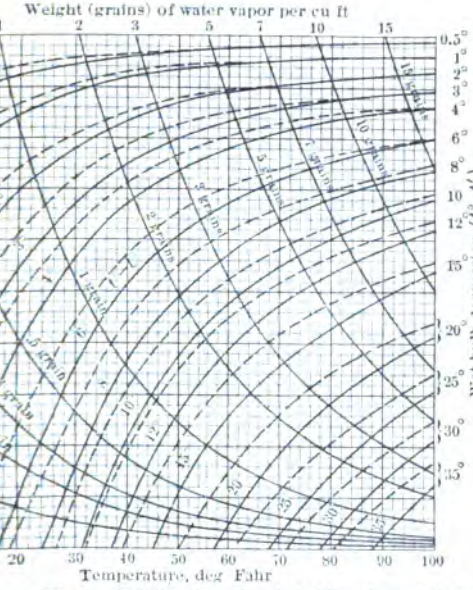
Psychrometric formula employed by the U S W tive humidity, and the temp of dew-points, is:

$$e = e' - 0.000367\,P\,(t - t')\left($$

where t and t' are the temps of the dry and wet-bul barom press, in; e' is the max or saturation press of bulb; and e is the press of aqueous vapor corresp Weather Bureau tables (3) are too extensive for in equal wts of water vapor per cu ft and curves of e varying temp and relative humidities.

Example of use of chart. Required relative h 30 in, and the corresponding wt of aqueous vapor. and wet bulb t' (if sling psychrometer is used, rea $t = 58°$ and $t' = 50°$, $t - t' = 8°$ = wet-bulb depres depression curve of 8° from the right until it intersect

ad in left-hand margin opposite point of intersection. Wt of vapor
r noting the curve (or interpolated curve of equal wt) which passes
ersection; in this case, 3 grains. At a high altitude, where barom
temp 58°, and wet-bulb depression 8°, the broken line curve of 8°



etric Chart. (Full lines for 30-in; dashed lines for 23-in barom)

tersects temp curve 58°. Relative humidity in this case is 61%;
rains. For other barom pressures it is necessary to interpolate, or
chrometric tables (3). Barom readings for different altitudes are

For wt
r vapor,
for wt of
Sec 14,

To de-
ce $t-t'$,
ry-bulb
whirled
e most
t. In
'eather
r, it is
he wet
ot less

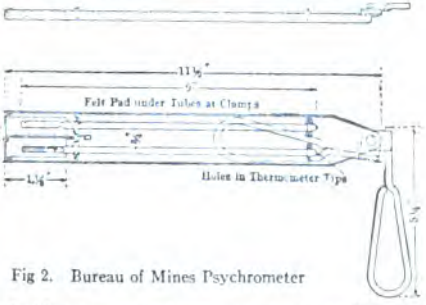


Fig 2. Bureau of Mines Psychrometer

It is obtainable at many instrument stores, is easily used,
erefore less suitable for mines than those having protect

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A form designed by Bureau of Mines (Fig 2) has minum frame, with the pivoted handle stiffly connect with aluminum liner and a shoulder strap. In wh should stand sideways to the air current, and if in from strong sunshine. The whirling and reading ar lowest reading of wet bulb has been obtained. Whe must be continued for a considerable period after wet as vapor from ice is given off slowly.

Stationary psychrometers (hygrometers), while type, are useful, and those having a wet bulb are fai of constant velocity. There are several recording changes in length of a hair corresponding to changes differential expansion of large metal bulbs, wet and scribing pen. Stationary hygrometers give good re chrometer observations and adjusted from time to ti lating current is artificially humidified to moisten co mine the hygrometric condition of the current.

2. Constituents of Mi

Mine air differs from normal atmospheric a contaminated by mine gases, or the normal pe is absorbed by breathing, by rapid and slow absorption by the mine walls or minerals. Gaseo tions of men and animals; from blasting, fires, ical action, and from gases given off by strata (

Quality of mine air may vary widely in different p artificially ventilated, variations in temp, and influx cause air currents. Ordinarily the composition of systematic ventilation (Sec 14). Recent laws in Gre certain minimum percentage of O, and limit the ma air (Art 6), as opposed to the older and prevalent o certain quantities of fresh air per min to be circulate

Normal mine air is air considered not inju and which does not contain dangerous proportion Its composition varies but little from that of th

Oxygen (O) is vitally important for animal the atmos (dry basis) and is slightly heavier th and 760 mm weighs 1.429 gm. At −118° C of 50 atmos (735 lb) per sq in. The liquid O mm. O has strong affinity for most elements. is called oxidation; rapid oxidation due to flame The self-generation of flame by rapid oxidation tion. When air is mixed in certain proportion inflammable dusts or gases, and a spark, glowin introduced, an explosion will probably take plac

O has neither color, taste, nor odor, and can not be dis from air. In well-ventilated mines O rarely falls bel cially in the interior coal fields of the U S, the coal when pieces are left in the gob, oxidation is sometim result. The pyrite or marcasite, generally present in assist, but spontaneous combustion may occur in a carbonaceous shale absorb O without formation of a CO_2. This action is more rapid with some kinds of c mines in coal fields especially subject to "gob fires." is slow the O content frequently falls to 18%. For deficiency, see Art 4.

Components of Normal and of Impure Mine Air

when O = 16; d = sp gr with air as unity, at 0° C (32° F) and mercury; e = boiling point at 760 mm.)

(a) Chem symbol	(b) Molec wt	(c) Gm per liter	(d) Sp gr	(e) Normal boiling temp, deg F	Usual source in mines
O_2	32.0	1.4292	1.1055	−183	In atmosphere
N_2	28.02	1.2514	0.9673	−195	"
CO_2	44.00	1.9768	1.5291	− 78	In atmos, blasting, oxidation, mine fires, emanations
CO	28.00	1.2505	0.9672	−190	Blasting, mine fires
NO_2	46.01	2.0549	1.5895	.....	Burning of nitro explosives
CH_4	16.03	0.7159	0.5545	−161	Emanations from coal measures
C_2H_6	30.05	1.3567	1.0494	− 89.3	Component natural gas
C_2H_2	26.02	1.1621	0.9056	.....	From carbide lamps
H_2	2.016	0.0898	0.0695	−253	Mine fires, explosions
H_2S	33.98	1.5392	1.1906	− 10	Explosions and rarely from springs
SO_2	64.07	2.9266	2.2638	− 10	Combustion of pyritiferous ores or native sulphur
H_2O	18.02	....	.....	100	In atmos and from evaporation

nent elements of some of the gases named above

(a)	(b)		(d)	(e)	
S	32.07	....	1.98 (x)	440	From pyrite in ore or coal
C	12.00	....	2.1–2.3	(y) (x)	From coal, methane or timber

erence to water at unity; y = sp gr of graphite with reference to rbon is known only in the solid form. Wt of 1 liter of dry air at , as determined by Regnault, is 1.2932 gm.

olume forms nearly ⅘ of the earth's atmos. It is colorless, odor- as little chemical affinity for other elements. At ordinary temp it upport life; it acts as a diluent of other atmospheric or mine gases. is its faculty of promoting the growth of certain bacteria, and thus oils at −195° C, whereas O boils at −183° C; hence, when liquid atmos, the first portions that evaporate are richer in N than in O. taken into account in the use of liquid air rescue apparatus (Art 19).

% of the atmos. It is heavier than N (sp gr = 1.38), and behaves ng grouped with it in ordinary analysis.

, or carbonic acid gas (CO_2), is found in the earth's atmos stant proportions, except in and near large cities and manu- Its proportions are kept constant by plant life, which takes and gives off free O. It is a product of both slow oxidation ion of carbonaceous substances. Hence in closed places and ntages of CO_2 are variable. It is colorless, odorless, and will

not support life or combustion. (For physiolo
variable constituent of blackdamp, which somet
decay of timbers and to fungous growth. BLA
and to absorption of O by coal, causing an exce
chiefly of N, with 1 to 20% CO_2, though rarely
by mine water, never CO_2 alone as sometimes a
tant constituent of AFTERDAMP, produced by fi
(For composition of afterdamp, see Art 15, 17.

CO_2 is a stable gas, but in presence of burning or gl
or burned to CO. In a closed mine the percentage o
through absorption of CO_2 by percolating water, by
reactions between O and coal, or timber, rarely by i
districts, by supplanting of original atmosphere, by in
by absorption of CO_2 by coal. It is much more solub
absorbs 1.71 of its volume of CO_2. Sp gr, 1.529; boili
fied at 0° C by a pressure of 35 atmos. Critical tem
If the liquid is allowed to escape freely, evaporation
passes directly from solid to gaseous state.

Carbon monoxide, or carbonic oxide (CO
produced by imperfect combustion. CO_2 is re
in appreciable quantities at temp above 500 to
in presence of excess of C the more rapidly is C
made in gas producers and incidentally formed
firing. With bad firing or wrongly designed fu
gases and sometimes may be observed at nig
the top of stacks. It is the important constitue
gas, and of blast-furnace gas.

CO is colorless and inodorous. Mixed with air
mixture will explode if ignited. Combustion formu
TEMP is about 600° C. CO burns in air with blue
separated from other gaseous products of combusti
ensue after a few respiratio ns when over 0.5% is pr
protected by oxygen breathing apparatus could dete
a safety lamp, is fallacious. Probably its associa
smoke in fumes from mine fires led to its being pop
may be present in quantities dangerous to life, whe
from a fire, or in afterdamp of an explosion, the term

Nitrous fumes (NO_2, N_2O_3) are not normally fou
duced when nitroglycerin or nitro-cellulose explosive
onating; then reddish fumes are produced which are
have the smell of fuming nitric acid, even when grea

Methane or marsh gas (CH_4) is the most
liery gases. By miners it is usually termed
as these terms are used whether or not other
are not synonymous with the term methane.
also is frequently found in tunneling through c
ally infiltrates into metal mines at contacts o
3). It is colorless and odorless; but its comm
collieries, where the air may be musty from de
the idea to many mining men that it has an od

CH_4 in burning has a bluish flame. Even
small as 1% it is visible in flame of a safety
down, as a faint conical blue cap (Art 9); wit
becomes larger and more distinct. Effect of
to cause it to "spire," or lengthen. When its

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han 14.8%, the mixture will explode if ignited. Maximum
d when the CH_4 is 9.4%.

es (4) a mixture of air with 7.5% of CH_4, which explodes violently at
n not be exploded if the press is reduced to 200 mm (8 in mercury).
hat increasing the press to 5 atmos did not affect the low limit, 5.5%;
re of CH_4 and air be heated to 500° C, the limit is lowered to between
lowering of explosive limit may be an important factor in the propa-
p, or coal dust and firedamp, explosion. Formula of combustion:
$+ 2 H_2O$. Volumetrically, 1 vol CH_4 requires 2 vols O or about
when ignited 1 vol CO_2, 2 vols aqueous vapor, and the unchanged
of ignition (Dixon) is from 650° to 750° C. If the burning gas is
n temp, the flame will go out. On this principle is based the con-
amps.

ropane (C_3H_8), **butane** (C_4H_{10}), and other heavier gases belonging to
components of "natural" gas, and are not normally found in col-
a pronounced crude oil or natural gas odor, which, apart from the
on, gives warning that there is probable leakage from a neighboring
the Bruceton (Pa) experimental mine it has been observed that the
on entering a place containing 1% "natural" gas, but a person who
ome time no longer notices the odor. Ignition temp is 520° to 630° C;
of CH_4.

is not found in mines under natural conditions, but is sometimes
uantities from unlighted or extinguished acetylene lamps, or from
n wet ground of calcium carbide (from which it is produced by the
When C_2H_2 is mixed with air in percentages between 3 and 73%,
sive. It ignites at a temp of 483° C (900° F), and can be ignited by
, or cigarette. 1 lb of pure carbide yields 5.5 cu ft C_2H_2; commercial
4.6 cu ft. It is colorless and tasteless, but has a peculiar smell not
g. Combustion formula: $C_2H_2 + 3 O_2 = 2 CO_2 + H_2O$.

not found in mines under normal conditions. It occurs in after-
t explosions, as a distillation product of the hot excess dust, after
has passed, and from coal fires as a product of destructive distilla-
est gas, 1 liter weighing 0.0895 gm. It ignites at 550° C, and burns
me. Mixed with air it is explosive between about 9 and 65%. Com-
$H_2 + O_2 = 2 H_2O$.

ide or sulphuretted hydrogen (H_2S) has been found, though very
CH_4 from gas "blowers" or "feeders" in coal mines. Being readily
s sometimes given off by stagnant water. It is occasionally detected
small quantities in afterdamp from coal-dust explosions, and in fumes
rous coal. It has the odor of rotten eggs, is very poisonous, and
pale blue flame.

or sulphurous anhydride (SO_2) is found in coal mines only as a
afterdamp, but in sulphur mines and in mines having rich pyrite,
which fires have started, SO_2 gas becomes a serious factor. It is very
orless, and has a suffocating sulphur smell.

ases is a fundamental principle which makes ventilation of
Gases diffuse into air at rates in inverse ratio to the square
ies.

about 8 and of CO_2, 22; rate of diffusion of CH_4 as compared with
3, or nearly twice as rapid. Prussian Firedamp Commission made
ng methane through a top inlet into a compartment of a gallery in
ding to 3, 4, and 5%; diffusion proceeded slowly, was nearly complete
irely complete until 4 hr. When diffusion is complete gases do not
ffusion is greatly assisted by mechanical air currents, but it is diffi-
on rates of diffusion for mining conditions. If gases pass through a
underground fan, the mixing is immediate, judging from tests at the
hen CH_4 is given off at the face, it immediately rises to the roof

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owing to its lightness, and if the outflow continues, high places, especially pot holes, unless a strong curre case, while gas is diffusing into the air along a botto the rate of diffusion the high place or pot hole will c A somewhat similar condition exists in pillar working emanates firedamp (Sec 14, Art 2, 3), and the presen constitutes a menace when the atmos press suddenly

Solid impurities in mine air. (See Art 3.)

3. Sources of Impu

Blasting. Gaseous products of explosives v sive, kind of wrapper, and whether used in c Table 3 shows results of tests by U S Bureau n Bichel gage, using 200-gm charges with thi analysis when an ordinary dynamite is burned i

Table 3. Gaseous Products of Explosives, in Bureau of Mines

(See Sec 4, Art 2)	CO_2	CO	NO_2
30% "straight" nitroglyc dyn	22.9	28.4	.....
40% "	27.3	26.9	.....
50% "	24.4	31.2	.....
60% "	22.2	34.6	.....
60% low-freezing dyn	8.9	47.4	.....
40% ammonia dyn	41.4	3.8	.....
40% gel dyn	50.8	3.0	.....
40% gel dyn, when *burned*	19.4	13.7	11.9 (a)
5% granulated nitroglyc powder	51.3	2.7	.....
FFF black blasting powder, in gage	49.7	10.8	.....
FFF black powder, when used in coal	19.2	28.2	.....

(a) Incl 0.6% NO_2. (b) Not determinable under cond

The table shows that the NO_2 red fumes, wh dangerous to breathe, are produced when dyn onated. Also, when black powder is used in from those in the testing gage; that is, more presence of carbon and hydro-carbons of coal. and H_2S totaled 45.4% by volume. If gases by CH_4 in the air and coal dust, they may be Smoke from black-powder, in blasting in coal, an open light. 1 lb of black powder produces a ing coal gives about 2.3 cu ft of inflammable ga about 7 times its volume of air, the 40 cu ft of if there is any coal dust or firedamp in the air. and Na_2CO_3 are always present in form of sm 60% dynamite produces 10 to 13 cu ft of gas, percentage of CO, CO_2, and N. CO is produce and volumes dependent on kind and quantity it is properly detonated. It is the principal and, being dangerous to breathe (Art 2, 5), the a working place after blasting may cause perm

oxygen depletion are important factors in the condition only in collieries, but also in metal mines containing carbo- in some Michigan iron mines, or those in which large amounts .

idize slightly, some, like those of central Illinois and Iowa (including hale) oxidize rapidly when broken into small pieces. Hence, sponntl/ occur in old gobs. Furthermore, oxygen is absorbed by coal . Parr, Univ of Ill, Dr. H. C. Porter, of Bureau of Mines and others). ses the depletion of O in the air circulating through old workings ver 2%. If a mine room is tightly sealed up, the O is rapidly ab- few weeks or months, practically all of it may have disappeared. s in extinguishing thoroughly sealed mine fires.

bacteriological action requiring O; sterilized timber does not decay. ed by hot, humid air, and probably by the crushing of timber, as in loyed in the block-caving systems, in the Lake Superior iron mines, s, and elsewhere. Adjacent to crushed masses of timber the deple- ction of CO_2, accompanied by heating, is so marked as to cause dis- s.

men and animals in well-ventilated mines are of small importance rces of gaseous impurity, but are a large factor in metal and other cial ventilation, especially in headings or stopes unconnected near workings. An aver man under different conditions, from complete l work, consumes 0.25 to 4 liters of O per min and exhales, together O, 10 to 20% less CO_2 than the O consumed, or 0.2 to 3 liters. For ions in mines the aver exhalation per man is between 2 and 3 liters of inhales 30 to 60 liters of air per min. If the CO_2 content of adja- re is to be kept to 1% or less, then 7 to 10 cu ft of pure air per man , and about 3 times this amount for each draft animal (Art 5).

f lights, other than electric, has important effect in inner s where ventilation is slow, particularly in metal mines with

. Paul (Bureau of Mines tests): a sperm candle, burning 1 gm, con- per hr; a miner's brass lamp, burning 4 to 6 gm oil, consumes 0.8 r's lamp, burning 8 gm oil, 2.16 cu ft; a driver's lamp, burning 6 gm ft; ordinary type of miner's carbide lamp, using 12 to 14 gm carbide. These tests show that a lamp consuming 1 cu ft O per hr uses up all r; and, to keep the depletion of O within the safe limit of 1%, such a 1 ft air per hr. The amount of gaseous impurity given off by large places is so considerable that, where air currents are feeble, the use of s is advisable.

om strata. CH_4 (Art 2) is the most important and dreaded n collieries. It is probably derived from coal-forming mate- n due to heat and long-continued pressure. There is no between the amount of gas retained in the strata and the mation; the quantity seems to depend more on geologic struc- boniferous to Eocene the coal measures when opened are CH_4.

ons, especially the contorted Appalachian anthracite folds, produce lat practically undisturbed beds of the middle interior fields. Also, l beds of the Rocky and Cascade Mts, more gas is found than in the f central Wyoming. But this rule is not invariable, and seems to be depth of impervious cover, giving vertical pressure and less chance Sec 14, Art 2).

H_4 in the strata appears to bear a rough relation to the press equal to depth of bed below water level, but this is subject s according to permeability of strata. CH_4 is held

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strata in the pores of the coal, from which i
coal is exposed in mining, and more rapidly a
crevices, fault zones, and porous strata abov
liberate gas as "blowers" or "feeders," lasting

Sometimes, as in Belgium and in a few instances
of men have smothered by instantaneous liberation
at the Agrappe mine, Belgium, 131 men were kille
versed the ventilation and took fire from a stove or
the gas found in crevices and reservoirs has been sl
tities given off by different kinds of broken coal hav
Most of the earlier tests tend to be misleading as gu
conditions, because the coals were heated to 100° C.
coals at ordinary temp, the samples yielded on suc
the remainder being chiefly N and air. Volume was
crushed. Coal from the Saarbrücken district, Gerr
2.5 its volume in gas. Coals apparently vary wi
crushing, but differences in laboratory methods n
tween results reported by various investigators.

Tests on American coals by Chamberlin (6), Pa
the total volume of gas from crushed coals at ordin
the volume of coal. Most of the gas is given off
rapidly. Monongah coal (W Va), crushed to 30 me
0.86% of the volume of coal, while that crushed
By allowing the gas to drain off in a vacuum over
coal gave off 1.83 volumes. In tests of American
gas given off is CH_4; the amount of CO_2 is from 5

Quantity of CH_4 liberated from the pores
liery is probably too small to be a serious fa
menace is the gas stored in crevices and porou
current of many mines in U S and abroad show b
is idle than when working, and in many others
This indicates that the quantity of gas escapi
mining is small compared with that issuing fr
tiguous strata, through crevices, joints, or fiss
pilation of figures of total steady inflow of C
obtained by multiplying volume of current by
by analysis, and reduced to a common basis o
spective of volume of ventilating current (8).

The figures in Table 4 are not of average c
to obtain; those for the Liévin mines for di
variations may be in the same mine. As an
itself, the highly gaseous conditions prevailin
cite mines are not so serious as in less gaseou
pure coal dust is very explosive (Art 14). A
cu ft of pure CH_4 per min, even if ventilatio
in the class of gaseous mines, in which safety
concentrations of gas are likely to be found
workings. In many mines not over 25% of
throughs (Sec 14, Art 6), the rest being lost by
making over 500 cu ft CH_4 per min may be
the U S make as much as 1 000 cu ft per mi
of the table for pure CH_4 gives the volume o
explosive limit, 5.5% (Sec 14, Art 2).

Effect of barometric pressure on issuance of
investigations in every mining country show contra

of Methane into Gaseous Collieries in Europe and the U S

	District	Country	CH_4, cu ft per min	Authority
..	Westfalia	Germany	623	Prussian Firedamp Comm
..	"	"	350	"
..	Saarbrücken	"	337	"
..	Lower Silesia	"	178	"
..	Pas-de-Calais	France	169	Léon Morin
..	"	"	320	"
..	"	"	507	"
..	"	"	391	"
..	"	"	292	"
03	"	"	62	"
05	"	"	248	"
08	"	"	523	"
ns				
..	.....	England	494	Royal Comm on Mines '07
	.....	"	522	"
m	.....	"	983	"
..	.....	"	406	"

	Kind of coal	State	CH_4, cu ft per min	Authority for sampling
..	Anthracite	Penn	3 701	Darton, N. H. (8)
e.	"	"	1 226	"
	"	"	3 308	"
I				
n.	"	"	3 212	"
..	Anthracite	Penn	2 037	Darton, N. H.
..	"	"	2 500	"
..	Bituminous	Ill	181	"
..	"	"	204	"
..	"	"	200	"
le	"	"	227	"
..	"	Wash	73	Wolflin, H. M.
..	"	Penn	450	Burrell, G. A.
..	"	"	700	Rice, G. S.
..	"	Ala	193	Morrow, S. L.
..	"	"	282	"
..	"	"	200	"
..	"	"	268	"

disastrous explosions have occurred, with firedamp probably a con-

eprecated "colliery warnings" of low barometer, on the ground
generally caused by carelessness or other conditions, rather than by
f firedamp, and that between such warnings care might be relaxed.
ervations made by Bureau of Mines, in a gaseous Pennsylvania
Alabama, indicated that a falling barometer was generally followed
CH_4 content of the return current, but the correspondence was not
the graphic curves were reversed. When a gaseous mine has ex-
poorly ventilated, especially if the roof be broken, much gas may
the atmos press falls the gas issues and diffuses into the air currents.
elieved that gas inclosed in the pores of coal, or that stored under
g in blowers, is affected by atmos press changes, since these rarely
s 0.5 lb press per sq in, as compared with the far greater press under
in the coal.

Emanations and outburst of CO_2 from the strata in certain small coal basins of Central France; miners of coal and dust dislodged, and even blown to the sur liable to result from blasting, shots are sometimes fi refuge chambers are also provided. Various theories outbursts. They are probably due to acid waters atta ures. This seems true of CO_2 sometimes found in open S W Wisconsin lead and zinc district. Many Cripple in contending with CO_2 filling fractures of volcanic r found that by bulkheading and blowing in air unde the CO_2 could be kept back. Possibly CO_2 may som blasting and from exhalations of miners, but generally

Nitrogen occasionally emanates from the strata. almost pure N. The natural gas used in Fort Worth contains about 40% N. Generally an excess of N in the O by the exposed strata.

Gases from coal mine fires. Fires in most mit men, other than fire-fighters, to work; but central Illinois, Iowa, and Missouri, slow-burning shut off by fire walls are apt to be disregarde escape through or around the walls. These gas accompanied by CO which would cause serious not properly managed. Similar conditions preva districts, where fires in abandoned mines threate

Gases from fires in metal mines, as in the carbon of some Michigan iron mines, may be serious when th ordinary means. Such gases consist of CO_2, SO_2, and sulphide ores of Arizona and Montana, fires have so old stopes, producing much SO_2 from burning ores, an bustion. (For gases from explosions of firedamp and

Hydrogen sulphide (H_2S) may be produced by the mine fire, by action of acid waters on sulphide ore, or on sulphates in stagnant water. It may also occur in powder. H_2S is very soluble, but may be liberated in up stagnant water in traversing old mine workings.

Gases from internal-combustion engines, as gasol Fatalities have been caused by the exhaust gases when t out sufficient circulation of air. CO_2 and CO are produ buretor adjustment and size of engine. O. P. Hood (9) of Mines on 2 types of gasolene locomotive, that the CO conditions is about 13.5%, while under normal running For safety and health of employes the maximum should of air required to dilute the gases to a safe point; or co limited to the maximum air current possible to maintai should not be used underground unless a definite volume the engine is running, so that, for the maximum amoun under worst conditions, its percentage in the air will no

Solid impurities in mine air. ROCK DUST dry holes or "uppers," with ordinary machine dri compressed air; (*c*) squibbing, or enlarging the charges; (*d*) adobe shots (Sec 4, Art 9), or shall ing dry ore by shoveling or from chutes. Causes dust, especially in wide, open stopes, as in the Jo and, if the dust is siliceous and made up of shar most favorable for causing pulmonary diseases (nary light dust clouds, is not considered injurious thrown into the air in a dense cloud, in presence

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dust out of the air (referring to above items): (*a*) Use lls, or a direct spray into hole. Drillmen dislike this method wet; remedy is to wear oil-cloth suits. The alternative is one type is a nose guard, with wet sponge to breathe through), not altogether satisfactory as they tend partly to throttle ain CO_2 from exhaled air. (*b*) Do not blow out dry drill . (*c*), (*d*) Squibbing and adobe and blockhole shooting e while men are working nearby; do it either at shift end, after wetting down the vicinity and cleaning the dry dust case of adobe shots, the dynamite should be covered with r dirt, not with dry material, nor without covering; putting vering material causes inefficient and dust-raising shots. (*e*) g or loading is easily remedied by sprinkling.

4. Temperature and Humidity

f air in large mines approximates the temp of adjacent ntinued lower or higher temp of air current, even if rapid, e walls only a few degrees, and if the current travels in con- more than 1 000 ft, its temp approaches that of the wall.

ges of temp affect the temp of the strata for a compara- e below surface (rarely more than 20 or 30 ft, Art 2), unless t air currents or streams of water. The Alaska gold-bear- frozen the year round, below a few ft in depth, though ner is abundant on the surface.

to locality. Though frost goes to great depths in the polar conditions have much greater effect on temp of the strata, at depths matic conditions. Thus, in the Lake Superior copper mines, the t great depths, as compared with the temp of the strata in most 2).

nperature with depth. Rate of increase of temp varies localities.

10) the temp gradient (depth for each degree rise in temp) varies ; ft; in Belgium, 56 to 93 ft (isolated case, 22 ft); in Germany, 63 tunnels the temp gradients, as measured from the mountain crests, St Gothard, 84.7 ft; Mont Cenis, 80.4 ft. In the Calumet & Hecla gassiz reported the temp at 4 580 ft as 72° F, and the mean temp at he very low gradient of 223.7 ft; whereas, in the adjacent Tama- nt is said to be 100 ft. In the Comstock silver mines, Nev, the only 29 ft per deg, due to presence of hot springs; in Ohio-Tono- per deg; Grass Valley mines, Cal, 107 ft; Rand mines, Transvaal, . Evidently, deposits due to action of thermal waters, in certain ction is still going on, will have correspondingly higher and more an in areas where such changes have long since ceased, or in un- strata where increase in temp is due to thickness of covering.

mine atmosphere may locally rise above the wall temp flow of hot waters; (*b*) active fires of broken sulphurous rbonaceous shales, or timber; (*c*) oxidation of sulphurous too slow to raise the temp to point of ignition of the ma- nd oxidation of timber, due to bacteriological action, which in crushed timber, as in the block-caving system, and where humid.

sumes the temp of the walls. Tests in the Monongah colliery, a traversing 4 000 ft of double entry, 30 000 B t u per min were ntilating current. In a number of tests the intake air averaged

27° F, aver velocity 453 ft, volume, 66 432 cu ft p
the aver temp of the air was 45° F, and at 3 800
return current (1).

Seasonal temperatures do not appreci
places in large mines. Where the ventilatin
the return current is practically constant fro

A series of tests lasting 1 year, in 20 typical Illin
max fluctuation in temp of return current of only
the relative tightness of stoppings and doors (2
workings deeper than 50 ft under solid strata, and
sensibly affected by seasonal changes, unless there

Table 5. Temperature and Humidity of Intak
Coal Mines (Highest and lowest weekly

Name of colliery	Depth of shaft, ft	Week ending	T Ou sid air
Oglesby (a)	464	Mar 3	15
		Sept 8	80
LaSalle (a)	440	Mar 3	15
		Sept 8	80
Big Muddy No 9	114	Feb 9	16
		Sept 8	79
Sherrard No 2	210	Feb 9	16
		July 14	78
Assumption No 1 (a)	1 004	Feb 9	15
		Sept 8	82
Empire No 2	105	Mar 3	15
		Apr 27	57
Peerless	230	Mar 2	19
		July 28	76
Woodside	245	Feb 9	15
		Oct 13	61
M & C	560	Mar 3	21
		Oct 13	61
Saline Co No 3	100	Dec 15	27
		Sept 8	76
Equality	90	Feb 9	18
		Oct 13	65
Hart & Williams No 1	632	Feb 9	12
		July 28	78
Big Muddy	100	Feb 9	18
		Oct 13	65
Superior No 3	340	Feb 2	20
		July 17	80
Madison No 2	90	Feb 9	21
		Sept 8	85
Lumaghi No 2	193	Feb 9	21
		Sept 8	83
Centralia No 5	676	Mar 3	25
		Sept 8	85

(a) = longwall mines; others a

Hot metal mines, Comstock Lode, N
air entering at 46° attained a temp of 71.6°
entrance, 91.9° at 10 000 feet, and 95° at 13 0

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g the tunnel was 94 to 95° F, which presumably represented that level. Locally the water temps were much higher.

ap of mine air can be effected locally by spraying hot surfaces with disadvantage of increasing the humidity), or by active circulation cool air. At long distances from the intake the latter plan will re- orking places only a few degrees, rarely as much as 10°. But, if ine are heated by fires or by crushing and pressure, as in the caving working place can be improved by diverting a current of cooler air evel or crosscut. This may be done by brattices or pipes, some- ementary blowers or "booster fans" (Sec 14). The return air from eferably pass direct into the main return current. This method is ining practice, but has not been so extensively employed in metal e.

lity in mine air is generally higher than that of the open averages higher in mines where there is seepage of water.

at 8 typical Illinois collieries (2) show 95% aver relative humidity , that of the outside air for same period being only 72% (Table 5). rence a quantity of water ranging from 0 to nearly 20 000 gal per m the mines per unit air current of 100 000 cu ft per min.

r in metal mines is more variable than in collieries, and is nflow of water. The absolute humidity (wt of water vapor) of an intake air current cooler than the mine temp never nless already saturated is increased by presence of water. ne walls and the contiguous air increases with depth, relative unless water is encountered.

ty in relative humidity possible in a metal mine is shown by inves- stock mines, Nev (11). In one tunnel, where intake air did not drainage water, for first 6 800 ft from portal (mostly untimbered), y came mainly from walls. Rate of evaporation of moisture was sq ft of exposed surface. From 6 800 to 10 000 ft, the air current drainage water and became almost saturated. On basis of 6 ft d water surface, the accession of water vapor was 798.5 grains per re humidity of working places varied from 22 to 100%.

ive humidity. A relative humidity approaching saturation is to obtain. In coal mines this is desired for preventing the dry- ble coal dust. HUMIDIFYING THE VENTILATING CURRENT may g systems, separately or in combination with artificial heating steam pipe coils, or, most easily, by steam jets (1).

the U S are comparatively shallow, and the temp of working places if this temp is not exceeded, saturation is neither uncomfortable t 5). In mines other than coal, lignite, and gilsonite, a high humid- nd the lower the better, especially in deep mines where the temp On the Comstock lode the temp in some drifts has reached 130° F tive humidity is difficult, and in most cases the only practicable ir to the working places through metal pipes, so it may not absorb

MINE HYGIENE

Health of Impurities and Conditions of Mine Air

uality of air for human breathing is variable in compo- it relates to immediate preservation of life. But for good ges of O, N, and CO_2 should not vary widely from normal, tter of wt or density of air, the lung action adapts itself is produced by changes in press from 0.5 atmosphere at high os in caissons.

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In deep diving a press of 6 atmos has been attaine
is less O in a unit of vol, the lung action must be quic
the same vol and wt is exhaled that was inhaled, wh
temp. O is the life-supporting element in air, and
food, through agency of lungs and blood, maintains b

**Table 6. Quantity of Oxygen Consumed and
Working, and Number and Volume of Respiratio**

(J. S. Haldane's report to Doncaster Coal Owner's C

	O_2 consumed per min, liters	CO_2 exhaled per min, liters	$CO_2 \div O_2$, %	b p
Rest in bed.....	0.237	0.197	0.83	
Rest, standing.....	0.328	0.264	0.81	
Walking 2 miles per hr...	0.780	0.662	0.85	
" 3 " ...	1.065	0.922	0.87	
" 4 " ...	1.595	1.395	0.88	
" 4.5 " ...	2.005	1.788	0.89	
" 5 " ...	2.543	2.386	0.94	

U S Bureau of Mines has recently made
with foregoing figures. It will be noted that t
that inhaled. Bureau tests of breathing nearl
tions of work showed a reduction of 2.11 to 5.
of O is not affected by enriching the air with
haled to O consumed (called the respiratory qu
person at rest to 0.95 for a person at hard wor
rise above 1.00. Ordinarily a person uses only
but during violent exertion the full lung capa
respiratory quotient. Though Table 6 indicat
per min is generally required, tests with O resc
care of especial exertion 3 liters of O per min sho
alent to 14.5 liters of air, or about 0.5 cu ft pe
completely consumed, but this is never the cas
closed space, like a mine, the O is to be maintai
1%, the air supply per man must average abou
for draft animals, about 30 cu ft per min.

Classification of conditions and impurities, a

(a) Conditions due to altitude, high temp and hu
gerous unless extreme.

(b) Impure air, not toxic, as air depleted of O; not da

(c) Gaseous impurities having toxic effect; not se
large quantities.

(d) Gaseous impurities acutely poisonous, and diffi

(e) Solid impurities leading to lung diseases.

Effect of high altitude: deep and rapid br
number of red corpuscles. As much coal and m
at elevations between 5 000 and 11 000 ft, and s
ft, the consequent decrease of O per unit of volu
level reduces working efficiency. Men should
heart and lungs are sound before giving them e

Effect of high temp and high humidity
Either condition alone is bearable, but when

results. At moderate temp, say under 70° F, relative humid-
s unobjectionable, and a nearly saturated atmos is not serious
in a good current of air; otherwise it is oppressive and a day's
inishes with a further rise in temp and saturation.

4): "At temp of 93° in still, saturated air I found that, though I was
ist and doing practically no work, my temp rose 5° in 2 hr, and was
when I found it necessary to come out." Church (monograph on
880) reports that "the immediate results of high temp and hot vapors
ness, dizziness, fainting, vomiting, and as graver results, insanity and
s in the Comstock mines at that time positively traceable to heat
hole. Probably heat increases the bad effects of powder fumes and
rd, referring to the same mines (1883), states that the ultimate effect
not easily noted. Power of recuperation appears to be extraordinary;
in is intense and frequent, there may be no lasting injury. Young
"Local physicians say that the average working life of Comstock
es 25 years, and that they show no greater susceptibility to disease
lents." Miners working stripped to the waist are careful not to expose
drafts; on passing from a hot to a cold place a heavy coat is used.
noved before going to surface, or trousers and coat are worn over them.
nd cold showers after coming off shift. Frequent drinking of, and
sts, arms, and head in ice water are resorted to. Frequent rests are
oling rooms, so situated as to receive the freshest and coolest air. In
, cold water from a hose is turned upon the miner while at work.
F, with 50 to 70% relative humidity, and air moving at 200 to 300 ft
prevent efficient work, nor is it particularly uncomfortable. Temp of
above conditions, considerably reduces efficiency. Same temp, with
moderate air velocity, greatly impairs efficiency; higher air velocity
nore bearable, but miners can not work long at one time. Temp of
turated atmos with no current is bad, and prolonged exposure to it,
, is dangerous. Temp of 90 to 98°, and saturated currents of 400 to
ery trying. Vitiated air impairs labor efficiency more than high temp.

urities without toxic effect, but causing oxygen deficiency, in-
EFFECT OF OXYGEN DEFICIENCY near sea-level or below it is the
n of O due to lower atmos press; as in mines at high altitudes.

⅙ less than at sea-level, as at Denver (5 000 ft), is not noticeable;
Leadville (10 000 ft), violent muscular exertion causes rapid breath-
rt action; continuous exertion causes faintness. The O present in a
eadville is the same as that in a sea-level mine, where the O has been
to 13.8%. At top of Pike's Peak (14 147 ft) the barometer registers
present corresponds with but 12% O in air at sea-level. People be-
owever, and live at these and higher altitudes. In Peru, Bolivia, and
ere are many inhabitants above 15 000 ft.
4) that a person not exerting himself will rarely notice anything un-
entage of O falls to 10%; breathing then becomes deeper and quicker,
he face somewhat dusky. At 7% there is usually distinct panting,
nd the face assumes a leaden blue color; the mind becomes confused,
ed, though the subject may be quite unaware of the fact. IN AIR CON-
of consciousness occurs within 40 sec or less, without warning symp-
dentally putting their heads up into a roof pocket of CH_4 have almost
nconscious. In a test made at the Pittsburg station of U S Bureau
f the rescue corps, WHEN O WAS REDUCED TO 7%, in a mixture of air
t lost consciousness without much warning. The test being imme-
e subject at once recovered consciousness in the open atmosphere,
tress until the next day, when he was decidedly unwell. Haldane
sciousness in air deprived of O is quicker than in drowning; not only
cut off, but that previously in the lungs is rapidly removed; loss of
ickly followed by convulsions, then by cessation of respiration." In
dogs, the heart continues to beat from 2 to 8 min; in men this period
onger. SO LONG AS THE HEART BEATS, HOWEVER FEEBLY, AN'

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MAY BE RESTORED BY ARTIFICIAL RESPIRATION. T
as the after effects of deprivation of O are serious,
soon recover. Whereas CO_2 in excess carries its ov
danger is imminent, O may be so deficient as to imp

Since an ordinary oil-fed lamp flame is ext
diminishing and flickering of the flame is a g
is below normal. An acetylene flame is not a
local atmos dangerous to life, as it will burn wi

Gaseous impurities having toxic effect,
exposure. Excess of CO_2 is of frequent occu
cause slight distress when it exceeds 1 or 2%
until over 3%. WITH 3 TO 4% CO_2 BREATHI
is marked panting and increased frequency
accompanied by throbbing head and flushed
noticeable on return to fresh air. With 8 to 1

Haldane states (4) that AT 11% CO_2 UNCONSCIO
result unless there is exposure to this amount for
from old workings, owing to its being heavier than
diffusing but slowly in weak air currents; hence a p
held up until he can be taken to fresh air. But, si
include much CO_2, and being heated and containing
the roof, persons not protected with breathing appa
CO_2 in itself has little effect on a lamp flame, but
the deficiency of O which always accompanies pres
CO_2 is prompt removal of the person affected to fre
O, or first aid treatment or both (Art 20).

Gaseous impurities acutely poisonous.
see Art 2 and 3; their hygienic effects are as fo
is the commonest of the deadly mine gases.
having weak ventilation, when blasting is don
mine fires, and in return air currents from im
latter conditions are more prevalent in coal
It always exists in dangerous amounts in the

Oxygen necessary for maintenance of life is absor
of HÆMOGLOBIN (red-colored matter of the blood);
pound with hæmoglobin, and is thus carried to t
greater affinity for CO than for O (250 times grea
taken up CO can no longer take up O. It becom
proportionate to the percentages of CO and O pres
affinities for these gases. When the atmos contains o
½ saturated; with 0.08%, ½ saturated; with 0.16%
about 3 liters of blood, which is capable of combining
tains 0.2% CO, and the man is at rest, he breathes
5 liters reach the lung cells. As he absorbs not mor
it will be 0.5 hr before his blood becomes half sat
0.2% CO was present the hæmoglobin did not reac

A percentage of CO far too small to form a "
noticeable in any way (Art 9), *may be very dan*
unconscious without warning.

In recovery work following an explosion in Alaba
was nearly overcome, and was barely rescued by div
overcome. Though the safety lamps were burning
the effect on himself, several were finally seen to sin
uration there is tendency to dizziness and shortness
is scarcely possible to stand, and slight exertion c
resemble those caused by alcohol. If death appro

aturated with CO. The percentage of CO that is fatal if breathed s with individuals. Anything above 0.15% is dangerous, and 0.4% eath. For continued exposure 0.03% or over is unsafe, as the effects tent cumulative because of the greater affinity of hæmoglobin for

s rescued after becoming unconscious, complete recovery is often o the tissues, and serious ailments sometimes ensue. A person may een exposed until he reaches fresh cool air, when he may suddenly a of hæmoglobin with CO gives the blood a pink color, the greater eper the color, and this continues after death; so that a person killed bright red blood instead of the dark colored blood of one dying from ce.

O poisoning is prompt removal to fresh air, in a warm place, of pure O. If patient is breathing faintly or not at all, give and administer O.

ions of small percentages of CO are given by the behavior of cana- als. Canaries are so sensitive and certain that they have been offi- escue work in Great Britain, and in the U S by the Bureau of Mines. anaries are much more sensitive than white mice (Table 7).

Summary of Burrell's CO Tests with Canaries (12)

s, %	Effect
	Very slight distress at end of 1 hr
	Weaker at end of 1 hr than after exposure to 0.09%
	Distress in 3 min. Fell from perch in 18 min
	Distress in 1.5 min. Fell from perch in 5 min
	Fell from perch in 2.5 min

p from its perch in an atmos in which a man could remain for some ious effect, though such warning should cause immediate withdrawal ss breathing apparatus is used. In a chamber containing 0.25% of ed distress in 1 min and fell from its perch in 3 min. Burrell re- s 20 min, which caused a slight headache. After withdrawal to fresh and headache for several hours. For symptoms caused by serious g the fire disaster in Snaefell lead mine, Isle of Man, see (4).

drogen (H_2S) is more acutely poisonous than CO, but is fortunately es (2, 3). According to Lehmann, air containing 0.07% will cause nals and to man after 1 hr exposure; in an experiment on a man, were produced within a few minutes in breathing air containing 0.2% would kill dogs and cats in 1.5 min. Haldane says it acts t, and after absorption affects the brain and other parts. It makes ange in the blood. With very small percentages the first symptoms eyes and air passages; with more, eye inflammation results, accom- pain, particularly the next night; with 0.05%, giddiness, vomiting ath are soon caused. Men are quickly stricken down on entering an uch H_2S, as in sewers (4).

s or "red fumes" (NO_2) must be carefully guarded or mines using large quantities of dynamite, since it may re- detonation of a blasting cap, causing the dynamite to burn.

nel, 13 men exposed to NO_2 by a change in the ventilating current choking, nausea, profuse perspiration and headache, but revived on But they soon began to cough up bloody mucus, and showed NO_2 rs 9 men died. The others, including the rescuers, were ill for months 20 lives have been lost at one time from NO_2 fumes produced by the dynamite (4). Air containing enough NO_2 to produce irritation in sages is very dangerous. The after-effect is frequently an intensely Hence the need to avoid going back too soon into dynamite smoke,

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if there is any characteristic smell of NO_2. Men wh
or H_2S, should, after removal to good air, at once
protected from inclement weather, and prevented fr
taken to their homes, or preferably to a hospital.

Siliceous dust (Art 3). The seriousness of
tion by investigations into the causes of ph
SILICOSIS, in the Transvaal (1903 report), in C
later commissions in South Africa and Austr
in mines containing siliceous dust is far greater
Recent investigations by the U S Health and
of Mines prove that similar serious condition
U S, where quartz, phonolite, rhyolite, or othe
as at Butte and Cripple Creek; also that esp
in the cherty (flinty) "sheet ground" of Joplin,
many years, DUST OF ORDINARY METALLIC ORE
DOES NOT CAUSE PHTHISIS. Though the use
other precautions, will bring improvement, a
directly or indirectly attributable to breathing

Drillmen in certain mines are incapacitated for
when drilling dry holes, though they may be able t
tasks before dying of silicosis or consumption. Po
any lung disease. Absence of change and bath-ho
miner to go out into the cold air in damp or inadequ
condition of the lungs, hasten the end through acqu

Quantities of rock dust found in mine air.
mine, Cornwall, 1904) found the amount of dust i
holes, to be from 0.05 to 1.14 mg per liter, averaging
0.22 mg; in hand-drilled dry holes, 0.01 to 0.41 mg;
0.34; at a rock breaker, when sprinkler was not
within 3 to 5 min after blasting, 0.06 to 0.34 mg.
it is not dissolved; the sharp particles become emb
cells, impairing vitality of the tissue and rendering
and acute pulmonary diseases (4, 16).

Remedies for siliceous dust. Free use o
ever dust is likely to be made, and not usi
shots, and adobe shots, without first wetting
until the dust settled, are practical methods of

Noseguards with wet sponges are efficient, but if
obstruct breathing, and it is difficult to enforce their
ciated with much siliceous material appear to give
shows this to be the case in iron mining in Englan
Franklin Furnace, N J.

Poisonous metallic dusts require special
must be taken in case of $PbCO_3$ dust, to prev
halation or through the mouth. In mining cin
taken to protect workmen from poisoning.
cancer of the lungs (4), must also be guarded a

Tests for siliceous or poisonous dusts in air.
through a cotton filter in a tube, the dust collected
and after taking sample. Volume of air is determi
or pump. The pump is convenient, but likely to
valves or piston. Damp sugar as a filtering medium
has been adopted by U S Bureau of Mines, as bette
gators determine the volume by drawing the air thr
and exhaling into a liter bag, utilizing the mouth-b
ratus for connections to filter tube and bag. This

ns of breathing and location of the various workers. Due allowance ce in volume of inhaled and exhaled air, which, unless the subject is mself, are in the ratio of about 1 to 0.9.

. Permissible Percentages of Impurities

circulated. Since about 1905 the question of definitely limit- the different gaseous impurities has been considered, but there sus of opinion as to what the limits should be. The older ral, is to increase the ventilating current by artificial means orking places seems good. In coal mines in the U S, practice ulation of a definite volume of air, based on the number of imals. Under most state laws the air may be measured near in other states the more intelligent requirement prevails of e crosscut or breakthrough nearest the face. Under different he volume is generally from 100 to 150 cu ft per min per under- and 500 cu ft for each animal. In the anthracite region, e amount of firedamp, 200 cu ft per man are required. In ese limits are exceeded at the intake, but even in best prac- kage of stoppings or doors, only 65 to 70% reaches the last mines, not over 15 or 20%. The stipulated volumes exceed ments, which are about 10 cu ft per min per man, and 30 cu ft se (Art 5). The added amounts are considered necessary or eeping away firedamp, and to provide for leakage and slow and shale. When firedamp is present, enough air must be ent accumulations of gas.

sified as gaseous when a "firedamp cap" (see Art 9, Testing be observed at a number of faces or in the air current. This presence of 2 to 3% CH_4, although with a modern safety lamp ed by a skilled observer using a drawn-down flame.

in European coal mining countries has been to analyze the air in return current. This has been done in the Ronchamp collieries, (4), and has been extensively used by the French Mine Inspection W FRENCH MINING REGULATIONS (adopted 1911), no minimum volume . The practice is about 3 cu m (106 cu ft) per min per man, but advance heading must not contain more than 1.5%, and any other

nits of Impurities in Mine Air, Established by Analysis

	Minimum % O	Maximum %		Maximum temp	
		CO_2	CH_4(*g*)	Dry bulb	Wet bulb
ulations of 1911..	.....	.....	2.00(*c*)	95° F(*d*)	86° F(*d*)
of 1911.....	19	1.25	2.50(*f*)	.....	.....
l.....	.....	0.20	.....	.....	.....
h Wales(*a*).....	19	1.00	.....	.....	.....
.....	20	0.20	.....	87°	80°
tralia.....	.....	0.25(*e*)	.....	.....	.....
and(*b*).....	19	1.00	.....	.....	80°

(*b*) All operations in Thames and Waihi gold fields below 800 ft, 200 er man; 500 cu ft per horse. In other districts, 150 cu ft per man; (*c*) Except for 30 min after blasting. (*d*) Men forbidden to work bulb readings are higher, except in case of necessity. (*e*) Return air must not contain over 1.5% CH_4 for advance headings, and not over rrents. (*f*) Open lights must not be used in any seam (of any one urn air contains over 0.5% CH_4. (*g*) Causing withdrawal of m

return, 1% of CH_4. In Great Britain (last mining
specified, but certain maximum limits for impurities
determination. No lamp or light other than a locke
portable electric lamps) can be used when the air ha
gas. If the percentage is 2.5 or over in the general bo
men shall be withdrawn from that place. Other lim
deemed to be in a fit state . . . if the air contains les
CO_2." In the U S many collieries regularly analyze
cal limits are set in state regulations either for coal or
Australia, and New Zealand, mining commissions
ment of certain maximum limits of impurities, temp,

7. Special Miners' D

Ankylostomiasis, or miner's anemia (Germ
practically the same as the "hook-worm disease
sucking worm. It attaches itself to the walls o
sucking discs on its head. Length varies in
growth; hookworm of southern U S (Uncinaria
0.3 to 0.6 in (14).

The larva comes from a microscopic egg, hatching
ditions of air, heat, and moisture, into a microscop
and ready to enter the body as opportunity offers.
live nearly a year in soil with suitable environment.
the body through the skin. If a person treads on
gets particles of it on his hands or other parts of the
bore through the skin, especially in tender places,
causes local inflammation, called "ground itch." Th
swept through the heart into the lungs; but, being
blood vessels, bursts out into the air spaces. It wri
pipe to the throat; then it is swallowed, entering the
with infected food. From the stomach it passes into
and becomes a hookworm. A person seriously af
worms clinging to the intestines. They bite into th
is lost by bleeding. A poisonous fluid given off by t
of the bowels. Fortunately, it does not multiply in
eggs pass out with the stool. If a person is not reinf
but their lifetime in the body is 10 to 15 years.

Symptoms: pain in stomach, capricious ap
eating), obstinate constipation followed by di
unsteady pulse, and in cases running a fatal c
and dropsy. A person seriously affected look
and body appears starved. Distended abdom
with bloating of face, body, or legs, are especi
dren, on whom the effects of the disease are very
an old-appearing head, and other abnormalities;
cating the disease, whether in a mining or othe
more common and show less definite signs; p
only by examining stool with a microscope for

Treatment. Epsom salts and thymol, tak
of a physician, are commonly used remedies.

Origin. The disease is an ancient one; was evid
scribed by Dubini in 1838, and definitely identified
in 1854, as due to presence of a nematoid worm. S
termined the widespread prevalence of the hookwo
(14). The Rockefeller Sanitary Commission is carry
tional campaign in Southern agricultural districts, al
ination and treatment. In mining, it was first rec

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d tunnel, hence was called tunnel disease. In 1896, 107 cases were an miners; it spread rapidly, especially in Westphalian coal mines, mines are kept damp to prevent coal-dust explosions, and as many not, conditions favored propagation. It is said over one-half the , seriously reducing the coal output. Prussian Gov't adopted autions, and made the operators responsible for hospital care of ss than 2% of the miners were affected, by 1910 only 1%. The nic among Belgian miners in Liège district in 1908, when 26% were nes, 54%. Belgian Gov't took stringent measures, and by 1910, ed was reduced to 4%, in some collieries only 2%. In Italy, at 38% of the workers were affected, but after spreading lime on floor ber reported in 1910 was only 5.2%. In England the disease ap- nine, Cornwall, 1902, which led to an investigation for the Home e. In mines of the U S, the disease has been noted in a joint pre- by Marine Health and Hospital Service and the Bureau of Mines. orted its presence among some southern coal mine workers, but cts of the U S have been but little affected. Coal mines in this cool, and the acidity of their drainage water is no doubt a pre- is also true in many deep and hot metal mines, though in 2 deep es 60 and 70% of the miners are reported as affected mildly. In re used, to which may stick infected earth brought by boots, there for spread of the disease.

ns: use of shoes or boots without cracks or holes and leather equently with antiseptic solutions; rigid sanitary precau- mines and homes; neither children nor grown persons to go lsory use underground of portable sanitary latrines, which ned and disinfected on the surface (Art 8). When there is the disease being present at a mine, there should be bacte- of defecations. Many German collieries have medical staffs pplicants for employment as miners, and periodically inspect efern, santonine, thymol, and other vermifuges are prescribed. easily to treatment and mild cases are quickly curable.

culiarly a miner's disease, affects the optic nerves, and causes eballs. In severe cases the victim is unable to walk straight, ent rotary motion. Except in very severe cases it is con-). ALLEGED CAUSES: insufficient light, and work of bottom- ting, while lying and working in a strained position, with eyes Another cause is said to be working in thin seams, where lent of throwing the head on one side, and looking diagon- serve the roof as the miner walks along. The consensus of inion is that the chief, if not sole, cause is poor lighting, as particularly the older types, the light of which is less than an investigation showed that 91% of those affected used eas in 2 mines using open lights only 0.35% were affected.

n's Compensation Act became effective in Great Britain, nystag- t forcibly to the attention of the British mining industry. In 1909 owed in 631 cases; 1910, in 956 new cases. General use of the pick , and necessity in most European mines of regular use of safety ease serious in the past, but the improvement in portable electric remedy within reach. It has been stated by some British physi- ted with nystagmus could not see a "gas cap" in a safety lamp, nace in a gaseous mine. Dr. J. Court tested 106 miners; though l but 4 could detect 2.5 to 4% of CH_4 (17).

illumination, as indicated by tests at Cockerill colliery, Belgium. each given 2 benzine safety lamps, instead of one, and examina- nth showed general decrease of nystagmus. To decrease danger of a broken safety lamp, it is preferable, instead of providing 2 safety

lamps, to provide 1 lamp and a permissible sto
that give a conical angle of lighting of at least 130
greater than 1 c p (Sec 42, Art 15).

In U S few thin coal seams (say less than 40 in th
wall workings in Northern Illinois little hand unde
used in comparatively few collieries, except by "fir
lights are the rule. Even in gaseous mines permis
placing safety lamps. Open lights, oil and acetyl
give 3 to 10 times the illumination of a safety lamp.
tagmus in U S has been made, but if there were man
by mine physicians who are constantly examining
and Ellis, of State Board of Health, examined 500

Color-blindness, though not specifically a
portance in gaseous mines. The faint blue fl
when the fuel flame is drawn down, is indi
afflicted with color-blindness; hence the adv
defect all applicants for positions intrusted wit
bosses," mine examiners, underground manag
rejecting those who cannot clearly detect the s

8. Underground Sa

Requisites (*a* and *b* being summarized from Art
(*a*) Good air, containing not less than 19% O, not
poisonous gases. Air temp should be below 80° F, i
rather than sanitation, the body of air should contai
tions are usually attainable by having good ventilati
(*b*) In mines producing siliceous dust the air shou
should be done with water-injection drills, or accompa
be cleaned with scrapers and not blown out with com
covered with wet stemming material, and dry ore w
(*c*) Sanitary latrines should be provided (15), espe
so located that one may be reached in not over a
working place of each man; where possible the statio
lights. The latrines should be taken daily to th
fumigated. An excellent latrine, used in certain
body mounted on a mine truck for haulage so that
the mines on cages. Where skips are used tub latri
lain with tight-fitting covers. The Butte latrine whe
with water to keep the excrement fluid; it is arrange
thoroughly by connecting it to a water main. Aft
with an antiseptic solution. WATER-FLUSHED CLOSE
shaft stations or landings, the excrement being flus
with the mine water. If the quantity of water be la
near habitations, the plan is usually unobjectionable
from water closets serving 150 men runs into an un
water discharges from this into the sump. In some
used (wooden boxes about 10 by 18 by 20 in deep);
dry dirt and a scoop, also disinfectant for sprinkling
nearly full, the cover is nailed on and it is sent out
dynamite boxes can be similarly used. In coal mi
haulage sidings or on main levels, latrines are bad
lution in the vicinity, but at the "face" in most
or clay refuse, which is stowed in the "gob." In thi
deeply without trouble, and thus be safely cared for.
(*d*) Underground cabins, pump rooms, engine ro
men may gather, should be frequently whitewashed
(*e*) Miners should be required to take their lunc
dinner buckets; in some cases they may be permit
waste rock or refuse to be dumped on the surface.

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re ankylostomiasis (Art 7) is found, lime should occasionally be ways, gangways, or entries, as a disinfectant, and the resting places eaten should be whitewashed. In collieries calcium chloride will pose, and be of assistance in compacting coal dust.

is an aid to sanitation, and so far as possible haulageways, manways, andings should be whitewashed. Electric lights should be installed, missible in gaseous collieries.

be provided for heated and perspiring men, while waiting at shift own-cast shafts, and cages for hoisting men in such shafts should be hot mines (where the law permits) there should be an inclosed but n surface landing of shaft to bath or change-house.

r must be of good quality, and should be provided under sanitary y by drinking fountains not requiring the touch of the lips, as in ipe-tap. As it is rarely feasible to provide running water for drink- hout a mine (unless in conjunction with pipe lines for sprinkling), be kept in sanitary condition should be provided; each man should his own drinking cup, and the faucet should be so placed in a recess se his lips, but is compelled to use his cup.

LAMPS, AND APPARATUS FOR INING IMPURITIES IN MINE AIR

9. Safety Lamps

in gaseous mines to provide means of illumination which, d, will not ignite firedamp; (*b*) a quick means of testing for in "gaseous" or in "open-light" mines. It is possible to flame (as was formerly done), but it is a dangerous practice. ous safety lamps the following are taken as representative.

), invented in 1815 by Sir Humphrey Davy, lamp, and is used to the present day. The : *a*, brass hood; *b*, brass ring; *c*, gauze cap; *e*, 3 iron standards; *f*, wick snuffer (pricker); ick holder; *i*, brass ring; *j*, brass oil fount; ic feature, separation of the flame from the by wire gauze, is common to all fuel-burning conductivity wire gauze cools an impinging w that at which CH_4 ignites. The "Davy," feties, is a small brass lamp with round wick, ntle with 28 wires per linear in, or 784 aper- e mantle is cylindrical, about 6 in high by s a double gauze cap fitting inside of stand- lamp body. A wire, called "snuffer" or ough the oil vessel from the bottom, and has evel, so the wick can be cleaned, raised or oving the gauze. To prevent opening a safety ine every lamp should be locked or sealed. ty lamp in a firedamp mixture depends chiefly f air inlets and of outlets for escape of gases he "Davy" air may enter through any part burned gases escape from any other part. t air, the spent gases go out through the top e double, to resist wear and corrosion. Free as, and rapid discharge of combustion pro- convenient for a quick test. But these very

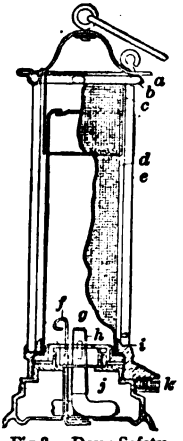

Fig 3. Davy Safety Lamp

ent of danger, since if put quickly into an explosive atmos a strong within the lamp, which might pass flame to the outside. Also, the a current of an explosive mixture, having a velocity of over 300 feet of 450 ft per min of explosive mixture will cause the flame to pass less than 2 min; at a velocity of 600 ft, in 10 or 15 sec. In a quiet

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atmos if gas is flaming in the lamp, and it is dropped, if it falls sidewise. It is therefore not a safe lamp exce countries it is not permitted in gaseous mines. The P only large coal district which extensively uses the "Da by the ordinary Davy is small, 0.16 to 0.20 c p.

Stephenson or Geordie lamp was invented by th son, soon after Davy's invention. Within the gauze a perforated copper diaphragm. The air-feed is th below the gauze and glass. Th cylinder and under air-feed are

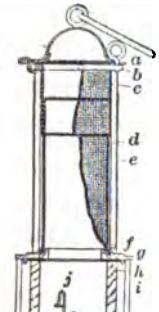

Fig 4. Clanny Safety Lamp

Clanny lamp (Fig 4) c Geordie. Around the flam ney, surmounted by a gauze the lower part of the gauze, This causes baffling, which t lamp gives much more light

The parts of the lamp are: *c*, gauze cap; *d*, main wire ga washer; *g*, brass ring, middle; *j*, wick snuffer; *k*, cotton wick; *n*, brass ring, lower; *o*, brass c neted single-gauze Clanny is n but addition of a bonnet or pi in a strong current of explosive Clanny is poor, but modified guided, have better combustion

Mueseler lamp is superio ter combustion by means of which extends a little below nular space for air-feed betwe is held in place by gauze ring at top of glass. An u an unbonneted Clanny. BONNETED MUESELER

The parts of the lamp are: *a*, iron hood; *b*, iron bo gauze; *d*, metal chimney; *e*, brass ring; *f*, glass globe; ring; *h*, 5 brass standards; *i*, wick adjuster; *j*, wick fount; *l*, key lock; *m*, circle of ventilating holes. B combustion chimney it gives much more light than lamps. Objection to the chimney is that it conceal flame if the latter gets too high on account of fireda remedied in recent English modifications by making combustion chimney of glass.

Marsaut lamp differs from the Clanny chief 2 or 3 concentric gauzes above the glass cylinder air spaces between them, which widen toward reducing the diam of the inner gauzes. It is muc the single-gauze unbonneted Clanny, due to ret the 2 gauzes of the escape of the burned gases, a blanket of non-explosive gas restricting an int sion. Fig 6 shows a BONNETED MARSAUT.

The parts of the lamp are: *a*, steel hood; *b*, steel bon iron gauze; *d*, inner iron gauze; *e*, upper brass bushi rings attached to gauze; *g*, asbestos washer; *h*, 5 bras *i*, glass globe; *j*, flat cotton wick; *k*, copper wick adjus brass ring; *n*, brass ring; *o*, brass ring; *p*, brass oil ring for washer; *s*, key lock; *t*, vertical openings fo Marsaut concluded that: (*a*) a small-diam lamp, like

me susceptible to explosion is small; (*b*) a lamp without a glass nal explosions, because the glass confines the gases and acts like a st to reduce diam and height of glass); (*c*) a conical mantle is safer f internal explosions than a cylindrical mantle of same capacity; f combustion plays a part in preventing in- is not advisable to guide them by a chimney eseler). Marsaut also found that an iron rass, a brass lamp giving only 70% of the iron lamp of same design. Brass has higher ot, and makes oil viscous. Brass is easier to nerally used, though some lamps have cast-

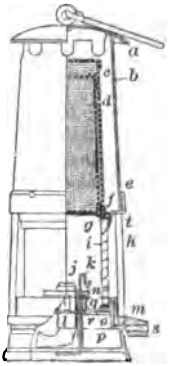

Fig 6. Bonneted Marsaut Lamp

ut lamps. The original Marsaut has a baffling air-feed which is very marked hence the glass may be smoked, giving But it is a good lamp for testing gas.

mination have been improved by various Marsauts are today the most favored safety Among them are the DEFLECTOR and PRO- are similar in principle, the air feed being es of combustion. In the earlier modifica- he gauze is shielded, and an angle diaphragm d extended to the gauze. There is a perfo- the glass. The entering air is deflected up- ring, and is thrown downward through the ss cylinder. Burned gases pass out through upper part of gauze. ications use the full bonnet with holes around the top, for escape of er 1 or 2 gauzes; 2 being considered safest. In some kinds a dia- used to separate upper and lower part of the gauze for outlet and ntering under the bonnet through slotted holes in the ring to which ed.

is an improvement on the Clanny. It has an inner glass chimney with an inlet under the glass, the air entering the lamp through the glass, passing down between the 2 glasses, and up inside the inner one to the flame. This design promotes good combustion. The gauze is also protected by a metal bonnet. Though it has good points, this lamp is not in general use.

Ashworth-Hepplewhite-Gray lamp introduced several new features (Fig 7). It is bonneted, with bottom feed, the air entering through 4 hollow standards. Air is drawn into these close to bottom through holes closed with slides, or at the top, so that tests may be made for a stratum of firedamp close to the roof. One of the standards may have an extension for testing for gas in a roof crevice or among timbers. Air passes from the standards into a circular groove within the brass fount, thence up through small holes to the flame (in this respect returning to one of the features of the Geordie).

The parts of the lamp are: *a*, aluminum hood; *b*, brass bonnet; *c*, brass gauze; *d*, copper ring on gauze; *e*, asbestos washer; *f*, brass-tube standards; *g*, glass globe; *h*, round cotton wick; *i*, wick adjuster; *j*, tin burner; *k*, insulated tube; *l*, asbestos washer; *m*, brass ring; *n*, oil fount; *o*, locking screw; *p*, brass gauze; *r*, lower air inlet; *s*, upper air inlet; *t*, adjusting ring for air admission. As usually made it has a flat wick to increase illumination. It is a good working lamp, though the large standards cast objectionable a frustum of a cone, which assists in throwing light upward. The and the bonnet has no opening except through the top, the burned ugh radial holes above a restricted opening. Thus the spent ga:

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collect above and around the single gauze, making t
strong gas mixtures, even those considerably below

Hailwood combustion-feed lamp combi
the Evan Thomas and Mueseler.

It has an iron tube above with glass tube below,
which there is a combustion aperture. Into this pas
combustion which is completed in the inner glass ch
1 c p, measured horizontally. This and other safe
which are used extensively in Great Britain, have
easily kept clean, and reflect upward and outward
by the lamp's base.

Wolf lamp (Fig 8) was invented, 1883, b
volatile oil, as benzine, naphtha, or gasolene,
screw, instead of the prick
require frequent removal

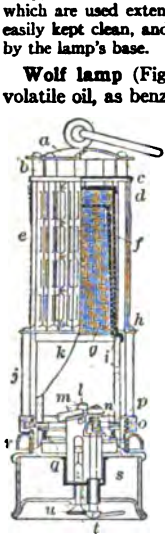

Fig 8. Wolf Safety Lamp, Type 1 B

The parts of the lamp are:
standards; *c*, steel ring, upper
corrugated; *f*, iron gauze, out
for standards, middle ring; *i*,
k, glass globe; *l*, cotton wick
o, brass ring, lower; *p*, asbes
brass baffle ring; *s*, brass naph
adjuster. Air enters through
vessel, passing thence throug
The wick, formerly round, no
give more light, is in a metal
base, with an adjacent tube
lowering a clip which holds top
ment of the wick by the screw
in testing for firedamp, as the
the yellow has disappeared.
tage for relighting, as internal
relighter is placed adjacent to
of paper matches, with clasp
rod projecting through the bo
ment is a flint rubbed by a mi
the wheels, sometimes provid
contain cerium or other subs
make lighting easier, because tests have shown that
wheels are thrown off, pass through the gauze, and m
vessel is packed loosely with wool to prevent the gas
on a flexible ring to permit expansion, with asbesto
as in other safety lamps. There are 2 copper, brass
a plain or corrugated bonnet. The miners' lamp is
made of aluminum, it weighs 2 lb 3 oz. The round
estimated at 0.60 to 0.90 c p; the flat wick, 0.75 to

The Wolf type is excellent both as a working
flame a trained observer can detect 1% firedam
easily detect 1.5 to 2%. This lamp has not been
naphtha is disliked, and individual relighters in
It is commoner in U S than any other. Severa
in U S, among them the Shenk, Seipel, and Ko

Kind and quantity of oil for safety la
lard oils were used; then vegetable oils, fro
finally mineral oils have been increasingly use
volatile oils, benzine, naphtha, and gasolene.
oils which do not vaporize at temp to which th

Characteristics of Representative Safety Lamps

Oil used	Air entering with ref to flame	Combustion chimney	No of gauzes	Bonnet orig'l lamp	Candle power(*b*)
Heavy oil	Below or above	None	One	No	0.15 to 0.25
"	Below	"	"	No	0.1 to 0.2
"	Above	"	"	No(*a*)	0.3 to 0.4
Heavy oil or naphtha	"	Chimney	"	No(*a*)	0.4 to 0.5
"	"	None	2 or 3	No(*a*)	0.3 to 0.5
Heavy oil	Essentially below	"	One	Yes	0.4 to 0.5
Light mineral	Below	Essentially	"	Yes	0.5 to 0.6
Naphtha or gasolene	Below	None	2	Yes	0.6 to 1.0

provided with bonnets. (*b*) Candle power horizontally without improved types. The great variation in candle power obtained tors is due partly to non-standardization of photometric methods, ces in size of lamp, burner, height of flame, and kind of oil used.

vy, Clanny, and other British lamps, railroad signal oil is mineral oil, with little incrusting residue). For the Wolf to 76° is usually employed. QUANTITY OF OIL BURNED in hr generally ranges from 500 to 700 gm. A lamp of the wick uses about 738 gm benzene.

lamps should be used in all gaseous mines. Best type is which can be opened only in a special magnet holder. In d seal is preferred by many, passing through lugs which and which can be stamped with a date. In "open-light ers commonly use a lamp locked with a key.

ty lamps are a great precaution in mines having rapid cur- vored in U S, Great Britain, and France, but in Germany, an mines generally, air velocities are kept low by splitting is often omitted, so that if gas flames up in the lamp it may . It is easy to see, also, if the gauze has become smoked.

uzes are best for most safety lamps. If there are holes in reserve; also the space between the gauzes tends to hold combustion. An unbonneted single-gauze lamp is unsafe of over 300 ft per min. At 450 ft, flame will pass through an 2 min.

s a rule, safety lamps are supplied and kept in order by the They should be systematically cared for in a fireproof ed by windows, and by protected incandescent electric house should be near the mine entrance, but at least 50 ft from explosions. No one but the lamp tenders should be he lamp room. The miners, on coming from the mine, must in this building, where they are taken apart, each part well he wire gauze), the oil vessel filled, and new wicks, asbestos parts supplied when necessary. Then the lamps are reas- breath or by a compressed-air blowing device, and are hung est plan is that each man shall be furnished the same num- ay. Lamps are delivered to men entering the mine by

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checking system, which, in case of accident, i
by his lamp number or check number.

Relighting safety lamps. A gaseous mi
A lamp may be extinguished by an accidenta
low flame, or from blackdamp in old working
the lamp and relight with a match. No one i
session of a match or smoking material.

There are 3 ways of relighting safety lamps:
itself (see Wolf lamp). This is preferred in Germ
LIGHTER, by use of an electric spark. It is a steel
cells, or a magneto, operated by an external crank
ment of the box, and the door is closed. A lighti
of the lamp nearly to the edge of the wick tube; the
into contact with one pole of an induction coil, and
oil vessel. A secondary current is produced in the
mary current from the magneto, causing sparks to
to top of wick tube. Batteries which may generat
even in the smallest amounts are objectionable, as
internal explosion might be serious. Though the
instances of internal explosions having ruptured t
tight it might be better to have suitable explosion-
faces or gauze, similar to those on explosion-proo
lighters in the mine to be convenient to the workin
in charge of a "fireboss," may be located on the
must necessarily be few in number, men may have

Internal lamp igniters are not favored in Great Bri
to relight a lamp while in an explosive atmospher
might be communicated to the outside body of fi
ferred, when properly safeguarded as to location.
have also approved certain magneto relighters, as c

Testing for firedamp (21). Firedamp, w
height of a flame to an extent varying with the
is termed testing with FULL FLAME. It is a qu
bosses and examiners; but is not reliable be
down due to the temp of oil or condition of w
in air containing several per cent of CH_4, the
is clearly wrong. CH_4 burns with a pale blue
just below the explosive point, which is about
from below, or when it contains ethane (as
limit of which is 5%. If the wick of an oil-l
yellow flame shows, and 1% or over of firedam
pale blue CAP forms above, though not separa
to the oil vapor. The primary flame should
well cleaned and the gasolene not too light.
with the firedamp cap, allowance must be m
The firedamp cap is a more or less well-def
pendent on percentage of gas and kind of lam
it takes some minutes for the cap to form, as
ture. To see the cap other nearby lights mus
height of cap is then estimated with the eye,
Scales for measuring the height of flame have
ally used for reasons given later With Wol
firedamp can be estimated, the cap of which
some prefer to leave a point of yellow in the
more difficult to discern the pale blue firedamp
the cap increases in height, and becomes more

lly along an entry or gangway, where there is reason to suspect ngth of flame should be observed from time to time. If the height adjusted for 0.5 hr prior to leaving an intake current, any increase y if it shows SPIRING) is due to firedamp; this should then be tested n flame. Spiring is the elongation of a normal flame into a high rican collieries near gas and oil districts, there is possible danger into the mine. Firedamp is normally nearly pure CH_4, but natural to 20% ethane and heavier hydrocarbon gases, the remainder latural gas behaves in a safety lamp similarly to CH_4, but its lower t 5.0% instead of 5.5% for CH_4 when ignited from the side.

mp caps can not be given precisely, as it varies with differ-

f Firedamp Caps from Photographs (Whalley and Tweedie, 1910)

Wick and oil	Height of cap, in		
	1% gas	2% gas	3% gas
ick, 0.23 in diam, paraffin oil	0.20	0.30	0.40
0.65 in wide, naphtha (boiling pt, ...	0.52	0.67	1.10
0.55 in wide, colza oil	0.17	0.50	1.00

oil tests were not very reliable, as the photographs were taken ide. Other tests indicate that the end view of cap gives a greater gh less than 10% greater.

g flame. The importance of this is shown by tests by G. H. ing a high flash-point petroleum and a bonneted Marsaut -fold burner. Fig 9 shows n 3% gas, the growth of in size of testing flame is percentages less than 3%. es that the density or vis- ore reliable indication than at to estimate percentages e familiar with the varia- egree of visibility for differ- gas and sizes of flame.

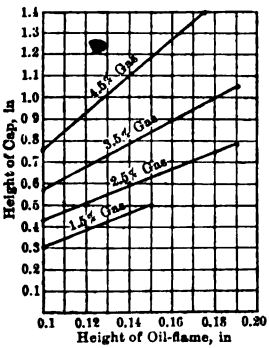

Fig 9. Increase in Height of Cap with Increase in Size of Testing Flame

miners' tests for firedamp Scotch mine by Whalley (19) mount of firedamp is detected eboss's ordinary inspection. le this test in the ordinary raised his lamp too quickly, it) noted that the flame was til they had raised their lamps ng (by analysis) 3.19 to even at is the least amount of fire- e be detected by a fireboss. Thirty four took this test. the percent- to 4.42, averaging 2.65%. Most of the tests were with a non-lumi- lest quantity detected by any of the men using a luminous flame was ount escapes detection by firebosses. Ten failed to find firedamp aried from 1.31 to 3.05 CH_4; average, 2.16%.

ncy of O and presence of CO_2 on safety-lamp flame dane and Llewellyn, by tests in a closed chamber until th ctive, while CO_2 was also formed (4). Other experime

Table 11. Effect of Deficiency of Oxygen a Lamp Flame

Time	Light, in c p	Percentage of		
		O	CO_2	D
11.50	0.405	20.90	0.05	
12.13	0.370	20.66	0.25	
12.28	0.315	20.34	0.52	
12.43	0.270	19.88	0.88	
1.3	0.170	19.34	1.26	
1.23	0.11	18.92	1.71	
1.45	0.045	18.28	2.17	
2.3	0.00	18.01	2.40	

These tests show clearly the importance of keepi 20% to get good illumination from fuel-burning lam

Fig 10. Clowes' Hydrogen Lamp

have shown that, i tuted for an equal appreciable effect Table 11 are due to with an ordinary sa 260° F flashpoint, a

Special lamps for officials and investiga what dangerous as wo (Fig 10), one of the le ploys compressed hyd tube at the side. The as the handle of what oil safety lamp of the plewhite-Gray type. lamp are: *a*, brass ho net; *c*, inner brass b gauze; *e*, glass globe standards; *g*, wick; *i*, hydrogen tube; *j*, *k*, iron gauze; *l*, bras brass oil fount; *n*, key lock; *o*, hydrogen tank; *p*, lower opening for air inlet; *r*, upper opening for air adjusting ring for air admission. In using, the hy by the flame, which is then turned off. Clowes' la where O is as low as 10 or 11%. Pieler's alcohol la sembles a large Davy, but uses alcohol, and has a parts of the lamp are: *a*, brass hood; *b*, steel bonne *d*, base ring; *e*, wick; *f*, wick adjuster; *g*, igniter ba *i*, brass oil fount; *j*, mica window; *k*, 5 brass stand The bonneted form of this lamp, with a narrow w from top to bottom, is safest. The alcohol flame i an iron shield, and the flame is adjusted, to standa tip is just below edge of shield. This lamp is sensit gives a flame 1.25 in high; 1%, 3.50 in; 1.5%, 4.75 the flame fills the lamp. Chesneau lamp (Fig 12 France, resembles the Pieler. The parts of the la hood; *b*, iron bonnet; *c*, copper gauze; *d*, sliding *g*, double iron gauze; *h*, 6 brass standards; *i*, tub dow; *l*, wick adjuster; *m*, graduated scale for mea ing shield and scale for measuring height of cap,

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uze. Wick is tubular, cylinder is iron instead of ed hands firedamp may be determined within 0.5%.

Electric Lamps and Lights

ric **lamps** in form of Geissler vacuum tubes, bichromate battery, were proposed about Dumas for use in mine rescue work, in con- pressed air breathing apparatus, where ordi- s would not burn.

hle electric lamp, for use in gaseous mines, was 95 in Belgium. In 1899 there were reported to be n storage-battery lamps in use in the Bracquegnies Later this lamp was introduced extensively in Eng- ric lamps are a necessary adjunct of rescue breath- evelopment of the two has been contemporaneous. rrières mine disaster, France, the rescue party, athing apparatus, used electric lights, and they recovery operations of every important disaster en the U S government first began mining investi- who took part in rescue operations and investiga- with storage-battery hand lamps; at the Marianna 3) dry-cell flash lights were also used. Soon after- lectric lights were tried in this country (22).

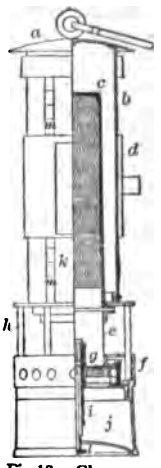

Fig 12. Chesneau Lamp

lectric lamps. In 1913, the U S Bureau of for PERMISSIBILITY OF PORTABLE ELECTRIC MINES. Previous experiments had shown no danger of outside oltage and amperage used) of sufficient intensity to ignite fire- ING FILAMENT due to breakage of the bulb WOULD CAUSE IGNI- e requirements are that the moment the outer glass dome or n the circuit must be opened or cut (22). The CEAG HAND LAMP, nd England, the HIRSCH LAMP (battery being hung from a belt ICO CAP LAMP passed the tests for safety. All had storage bat- urther requirements were made respecting durability, efficiency minimum illumination within an arc of 130° to be equal to that mp of Wolf type, burning naphtha of 0.7 sp gr, with a flame). The EDISON, WICO and MANNESMANN CAP LAMPS, also the lamp, have been placed on the revised list. Other lamps are

permissible electric mine lamps, chiefly of the cap type, have been ts of the U S. In Europe electric hand lamps have to some extent ps. According to W. Hargreaves (21), 80 Ceag electric lamps in howed an average of 1.03 c p at beginning of a shift, and 0.76 at end. week for 4 275 Ceag lamps was 3.1¢ for a 1-year period. British re- fective Jan 1, 1916, is 1 c p at beginning and 0.6 c p at end of day.

f miner's electric lamps compared with safety lamps are: mination. In case of cap lamps the illumination is where he direction towards which the person looks, whether up or ide. (*b*) No danger of igniting firedamp through a defect or , as in case of a safety lamp, or through breakage by drop- something fall on it. (*c*) Using a cap lamp, both hands are) No gases are given off in burning. (*e*) In an atmos contain- O, the efficiency does not fall off as with safety lamps. DISAD- ans of testing for firedamp. But this is not generally serious p, as the foremen and firebosses should be given safety lamps m from time to time.

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Dry-cell electric lights (flash lights) now on t
of permissibility for efficiency, or for safety in the n
the circuit if the bull's eye be broken. But, this is u
use, and they are otherwise safe, except those havi
potential, the use of which, where electric shot-firin
premature firing by accidental contact with the wi
rescue work, for which they are very useful. As th
they can not compete with storage-battery lamps for

Electric lighting from circuits. Incandescent
cuits, are widely used in metal mines, open-light c
the intake gangways of gaseous mines. They are
haulage entries, slopes, in stables, at shaft bottoms,
purposes on electric trolley haulageways. They are
mines, since if a bulb is broken the filament will ig
the feed lines may be short-circuited, due to wear, de
thus caused would ignite firedamp.

11. Apparatus for Detection of Gase

Safety lamps (fuel-burning) for detecting fi
pliances for detecting or analyzing mine gases a

Firedamp detectors are of a number of typ

(*a*) Those depending on diffusion of gases, or *osm*
indicator (23), consisting of a porous inverted pot, t
phragm, movement of which deflects a pointer. Po
ducting and drying material; outside of this are g
which will absorb CO_2. A tight cover surrounds t
taken off for making a test. Due to its lighter dens
the porous material faster than the inclosed pure air
diaphragm and deflects the pointer. To make anoth
out with pure air, unless more CH_4 is present than in fi
CH_4 can be determined, but the delicacy of adjustme
one test at a time, makes this type of instrument of d

(*b*) **Absorption detectors** employ a CH_4 absorben
device on this principle has appeared (23), the nat
An automatic valve admits a definite volume of the ai
the absorbent, or catalytic agent. The reduction in
by the position of liquid in a graduated U-tube co
end being open to atmosphere. Absorbent must be r

(*c*) **Haber firedamp whistle** (24), brought out i
of CH_4 by the difference in density of air and CH_4 m
sound waves. There are two whistles having the s
tainer, the other is exposed to the entrance of the
simultaneously by mechanical means. If CH_4 is pr
the higher note. It is claimed 2% CH_4 can readily
neutralize; otherwise, though the device is not ver
make it promising for special uses.

(*d*) **Shaw gas-testing machine** (25) is not portab
It has 2 cylinders, one containing pure illuminating ga
Measured quantities, obtainable from either cylind
and passed into an exploding chamber, where the mi
tion causes a gong to strike. Usually a number of
tains some CH_4, less of the gas is added to make
between the quantities of pure gas actually and the
is the amount of CH_4 in the sample. It is a labor
bersome and inefficient, compared with the more ac

(*e*) **Detectors employing catalytic agents,** as
are brought into presence of gases like CH_4, the s
slight rise of temp on their surfaces; this effect is

ium. It is utilized to heat adjacent substances which give indica-
ays. In the SUSSMAN ELECTRIC LAMP DETECTOR (27) by heating a
eter the rising column of mercury touches sealed terminals, thus
ric circuit that lights a red bulb. In the HOLMES-ALDERSON FIRE-
the heating of a compound strip of two metals of different expansion
it to curve and complete an electric circuit. This energizes a mag-
opens a switch cutting out the power circuit beyond. In the HOLMES-
LECTRIC LAMP ATTACHMENT (27), by the self-heating of a platinum wire
CH_4 a change in electric resistance is caused, indicated by a galvanom-
amount of heating by catalytic action in these devices is so small
ts are very delicate; and as the degree of catalytic action decreases
hese detectors has yet been found practicable for everyday use.

edamp indicator (26, 28, 29) was one of the first of the portable
"hot wire." It does not depend on catalytic action, but on added
urning on the surface of a glowing coil. There are 2 platinum coils,
ses a current generated by a hand-operated magneto. One coil is
ght tube with glass at one end; the other is exposed to the atmosphere,
der also with a glass at one end. The cylinders and coils have a com-
ends being opposed with a photometric screen between them, on a
he coils are as nearly equal in resistance as possible, to give in pure air
mination. The zero on the scale is adjusted in pure air. When ex-
ixture and a current passes through the coils, the coil in the gauze
brightly due to the burning of CH_4. The photometric screen is then
equal lighting, and the departure from zero, indicated by the scale
ed by tests in known percentages of CH_4 mixtures), gives the per-
The apparatus is sensitive, but the adjustments are too delicate for
culty of adjustment is added to by increase in resistance of the coils
burning of CH_4 on the platinum.

ethane detector (27, 30), brought out in 1902, utilizes the changes
corresponding differences in temp of platinum wire. Any CH_4 pres-
ntact with an exposed hot wire, increasing the temp of the wire, and
. Two wires of a Wheatstone bridge are isolated from contact with
wo are exposed to it within gauze cylinders. According to Goupil-
t's sensitiveness presented certain defects which prevented its use
1), and it is too large and heavy; but these defects might be remedied,
luable CH_4 detector.

.) **portable electrical gas detector** (27) resembles the Léon in
harged wire for surface burning of the CH_4, with a Wheatstone bridge
ncreased resistance of the hot wire. To reduce size of instrument, a
d, and, as its deflection is small, the circuit is so arranged that after
ved it may be brought back to zero by a sliding contact on a wire
circuit. The magnitude of the sliding contact movement required
acts as a multiplier of the length of the pointer.
A is a storage battery; *D*, *E*, *G*, *F* are 4 arms of
e; *H* is a wire of low resistance compared with
, connecting with free ends of *D* and *E*; *J* is a
platinum wire exposed to atmosphere, but in a
mber; *E* is a platinum wire in a tight container
witch for connecting the battery to take a read-
l contact-arm, sliding on wire *H* and connecting
h one terminal of the voltmeter.
may be attached to a portable electric light, or
keeping the hot wires at a temp not above that
out 500° C), the inventor believes the wire resist-
lly change. Also, by keeping the temp of the
ignition point of CH_4, there is little danger of an
out if explosion does occur proper protection by
the instrument as safe as an ordinary safety lamp. It is stated the
cate as low as 0.5% CH_4; and although not thoroughly proved it seems
dvance as a portable indicator.

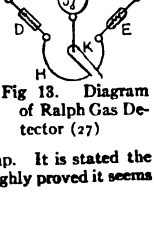

Fig 13. Diagram of Ralph Gas Detector (27)

(*i*) **Hot-wire, fixed, underground station indi** wire, inserted in a safety lamp, is heated by the turned in electrical resistance of the circuit may be indicate as in the foreman's office. The objection to leaving safety lamp, makes it unwise to use the device unde dicators are advantageous for testing explosion-proof

Blackdamp detectors. The behavior of a approx idea of the proportion of blackdamp pr flame is extinguished, by a 3 to 4% deficiency o blackdamp (Art. 2).

This property has been utilized in the HALDANE permits close field determinations of blackdamp. It 7 in long, 0.75 in internal diam, and a wax taper 1/4 When the lighted taper is inserted in the tube, and produced by the heat; the lower the taper in the larger the proportion of blackdamp present, the mo hence, if the taper be slowly lowered in the tube a po tinguished if more than 1% blackdamp is present presence of amounts of CO dangerous to health, no relative proportions of CO_2 and N. But, it is valuab particularly in metal mines where ventilation in pl midity diminishes the flame. The tube is graduate tion of saturation at 64° F, at which the water vapor vapor, or 2% by volume at 760 mm press.

Gas analysis apparatus. A device for det uring the proportion of certain mine gases may Thus, a good safety lamp, in experienced hands and show the flame change due to each added p damp be present, with corresponding deficien deficiency of O), CH_4 indications are either m extinction of the light, even when the CH_4 is DANGEROUS CONDITIONS OF MINE AIR CAN BE (33, 34, 35). Analyses of adjacent atmos are mine fires, especially when these are being se areas, when it is not known whether the fi Analyses of air from a fire area have sometim plosion; so that the men were withdrawn from of danger, and the workings sealed to prevent e

Control of ventilation in gaseous mines currents from each ventilating area has been companies in the U S (32). This has been do France, since 1891 (4), and other French co analyze the air from the various working pla collieries, control by analysis is widely employed lished regulations requiring that CH_4 and CO centages, thus practically compelling analysis. from time to time in other countries (Art. 6).

Methods of gas analysis. For general method but exact determinations require services of experi laboratories. In recent years quick volumetric met determinations of ordinary gases can be made by me little instruction, to within 0.2 or 0.3% of the exa accurate in practice, except in case of CO. When th CO_2 accompanying it to be readily determined by fie dangerous conditions. CO in dangerous amounts is afterdamp of explosions and mine fires, when it is n

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ities. In these conditions, particularly if there is much CO_2, the pres- l always be suspected, and a canary bird, which is very sensitive to CO, in a cage by men entering such atmos (12) (Art 14, 15, 17, 18, 19).

s-analysis apparatus is of the volumetric-determination type. for mines in the U S was the Orsat, which is widely employed ue gases. Similar apparatus has been used in European col- rrell (Bureau of Mines) designed a modified Orsat (35) which sively adopted for mine rescue and fire-fighting equipment.

s pipette, containing potassium hydroxide solution for removal of CO_2; ing alkaline pyrogallate solution for removal of O; *c*, pipette containing ous chloride solution for removal of CO; *d*, n pipette for burning the CH_4 in sample, inum wire, connected through sealed wires tric light storage or dry-cell battery, that inum wire to redness. Combustion pipette is filled with distilled water, slightly acidi- *e*, burette containing 50 cc, graduated to pparatus it contains 100 cc); *f* and *g*, level- ro mark; *i*, stopcock for admission of mine ube connection for admission of mine air rette is connected by a glass tube to *k*, and e pipettes. *a*, *b*, and *c* contain glass rods ncrease absorbent surface of the reagents. contained in a box 12 in high by 6.5 in wide h leather carrying handle. Front and back lide covers, which when removed leave pi- ed.

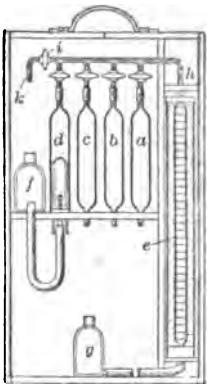

Fig 14. Modified Orsat Apparatus for Complete Analysis of Mine Air

eration. Solutions are leveled to marks on e connections by manipulating the leveling pipette in turn being adjusted while stop- ich is closed after adjustment. Mine air awn into *e* by first raising the water level level bottle *g*, stopcock *i* being open to rubber tube (not shown) connected to inlet e sample bottle, after the latter has been inverted in a pail of water and . By sufficiently lowering *g*, *e* may be filled with the air or gas sample to d, usually to full capacity. The sample is then successively passed into nd *c*, by opening the respective stopcocks and by raising and lowering *g*; each time into *e* to determine the reduction of volume caused by absorp- tive gas by the reagents. The absorbed volumes of CO_2, O_2, and CO are uccessively. Remaining gas is then passed into *d*, where the CH_4, if any, nal residue is assumed to be N.

eadings:

	cc		cc
analysis.....	50.0	Volume taken for combustion.....	40.4
potash absorption	48.7	Oxygen added.....	9.0
.....	1.3	Total volume.....	49.4
aline pyrogallate		Volume after combustion.....	45.3
.....	40.8	Contraction due to combustion....	4.1
.....	7.9	Volume after caustic potash	
rous chloride		absorption.....	43.4
.....	40.4	CO_2 produced by combustion, CH_4.	1.9
.....	0.4	Hydrogen $= \frac{2}{3}(4.1 - 2(1.9) = 0.2$ c c.	

alysis: CO_2, 2.6%; O_2, 15.8%; CO, 0.8%; CH_4, 3.8%; H_2, 0.4%; 76.6%.

pparatus specified can be used within the mine, provided the here the air is good and the safety lamp shows no dangerous lamp. A larger apparatus, of 100 cc capacity, is preferable for

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close determinations, but is more bulky to carry
U S Bureau of Mines' rescue stations and c
apparatus for determining only CO_2 and CH_4
Burrell has recently devised a very small ap
lamp. These are of special value in gaseous m
(See later publications of Bureau of Mines.)

MINE ACCIDENTS AND TH

12. Accidents in Co

Proper precautions greatly reduce the nu
comparing the rate of accidents in mines wher
average rate in same district. Such data is ra
coal districts, as in Pennsylvania, and compa
or countries are unsatisfactory because of vari

Comparisons may be based on: (*a*) number of
1 000 employed; (*b*) number killed or injured ann
Basis *b* is sometimes preferred, but neglects the inf
put per employe, and the question of compulsory
complete filling of excavations, as required in Fra
filling operations are exposed to mining hazards, b
ties believe that, while basis *a* is not ideal, it affords
over, basis *b* is not practicable for comparisons with
give the statistics by both methods.

**Table 12. Production, Number of Employe
Mines in the U S, 1896 to 1**

Year	Production (short tons) (*b*)	Number of employes (*b*)	Aver days active	Total
1896	191 986 357	393 162	185	1 089
1897	200 229 199	397 701	179	975
1898	219 976 267	401 221	190	1 064
1899	253 741 192	410 635	214	1 216
1900	263 684 027	448 581	212	1 492
1901	293 299 816	485 544	216	1 549
1902	301 590 439	518 197	197	1 895
1903	357 356 416	566 260	220	1 752
1904	351 816 643	593 693	202	2 004
1905	392 722 635	626 035	212	2 232
1906	414 157 278	640 780	209	2 116
1907	480 363 424	680 492	231	3 197
1908	415 842 638	690 438	195	2 449
1909	460 814 616	666 555	.....	2 668
1910	501 596 378	725 030	220	2 840
1911	496 221 168	728 348	220	2 729
1912	534 466 580	722 662	225	2 360
1913	570 048 125	747 644	238	2 785
1914 (*c*)	510 000 000	742 868 (*d*)	.....	2 451

(*a*) Figures for production and number of men emp
the United States," U S Geol Surv, except for numbe
compiled by Bureau of Mines. (*b*) These figures rep
ber of employes in entire coal-mining industry of the U
Subject to revision. (*d*) Subject to revision. (*e*) F
ber of men employed in States where records of fata

ber Killed at Coal Mines of the Principal Coal-producing Coun- to Production and to Number Employes (U S Bureau of Mines)

	Production (short tons)	Number of employes	Number killed			Production per death (short tons)
			Total	per 1 000 employes	per 1 000 000 short tons mined	
			1910			
...	501 5.6 000	725 030	2 840	3.92	5.66	177 000
...	296 148 000	1 032 702	1 754	1.70	5.92	169 000
...	158 694 000	577 263	1 140	1.97	7.18	139 000
...	42 274 000	196 786	213	1.08	5.04	198 000
...	26 364 000	143 701	136	0 95	5.16	194 000
...	17 285 000	137 467	307	2.23	17.76	56 000
...	15 183 000	63 969	65	0.93	4.28	234 000
...	12 755 000	105 285	160	1.52	12.54	80 000
es..	9 155 000	17 618	21	1.19	2.29	436 000
			1911			
...	496 000 000	728 348	2 719	3.73	5.48	183 000
...	304 503 000	1 049 897	1 232	1.17	4.05	247 000
...	167 022 000	586 538	1 176	2.00	7.04	142 000
...	.....	.....	.....	.....	.....	.....
...	25 412 000	144 054	165	1.15	6.49	154 000
...	.....	.....	.....	.....	.....	.....
...	15 851 000	69 827	88	1.26	5.55	180 000
...	13 495 000	106 598	148	1.39	10.97	91 000
es.	9 735 000	17 375	15	0.86	1.54	649 000

for bituminous and anthracite mines; lignite mines not included.

ification of Fatalities According to Cause (U S Bureau of Mines)

use	U S, 1911	Japan, 1910	Germany, 1911	India, 1911	Austria, 1911	Gr Britain, 1911	Belgium, 1911	France, 1910
al:								
.....	1 321	198	442	84	....	599	56	81
employed.....	1.81	1.44	0.75	0.79	....	0.57	0.39	0.41
s, und'g'd:								
.....	393	37	181	16	....	256	27	28
employed.....	0.54	0.27	0.31	0.15	....	0.24	0.19	0.14
t explosions:								
.....	379	8	32	16	....	34	1	9
employed.....	0.52	0.06	0.06	0.15	....	0.03	0.01	0.04
'd:								
.....	134	1	43	0	....	16	8	3
employed.....	0.18	0.01	0.07	0 00	....	0.02	0.06	0.02
d'g'd:								
.....	246	31	219	6	(a)63	81	23	16
employed.....	0.34	0.22	0.37	0.06	(a)0.90	0.08	0.16	0.08
.....	63	18	102	12	12	99	29	37
employed.....	0.09	0.13	0.17	0.11	0.17	0.09	0.20	0.19
:								
.....	183	14	157	14	13.	147	21	39
employed.....	0.25	0.10	0.27	0.13	0.19	0.14	0 14	0.20
.....	2 719	307	1 176	148	88	1 232	165	213
00 employed ...	3.73	2.23	2 00	1.39	1.26	1.17	1.15	1.08

(a) Includes all fatalities underground.

Non-fatal accident statistics in the U S a different states that compilations are inaccurate. for 1911 (36), show that 9 106 persons were seri days' loss of time; and 22 228 persons were sli time of over 1 day, and less than 20 days. As th in U S, in 1911 (Table 13), the number of seriou and of slight injuries over 8 times, the number of

Mine accident statistics for different states ar states having small production show widely varying n rates in different years. Pennsylvania, Illinois, and Oh

Table 15. Production, Number of Employes, Mines in U S, 1914 (U S Burea

State	No employes und'g'd and on surface	Und'
Alabama	(a) 24 552	12
Alaska and California	(a)40	
Arkansas	(a)4 652	1
Colorado	(b)10 550	7
Georgia and North Carolina	350	
Idaho and Nevada	6	
Illinois	(a)79 529	18
Indiana	(b)22 110	4
Iowa	(a)15 757	3
Kansas	(b)12 500	3
Kentucky	(a)26 332	5
Maryland	(b)5 675	1
Michigan	(b)2 300	
Missouri	(a)10 418	1
Montana	(b)3 660	
New Mexico	(b)4 021	1
North Dakota	(b)650	
Ohio	(a)45 815	6
Oklahoma	(b)8 948	3
Oregon	(a)203	
Pennsylvania (anthracite)	(a)175 745	53
Pennsylvania (bituminous)	(a)172 196	37
Tennessee	(b)9 580	1
Texas	(b)4 900	1
Utah	(b)4 348	1
Virginia	(a)9 162	1
Washington	(b)5 730	1
West Virginia	(a)74 786	52
Wyoming	(b)8 353	4
Total, 1914	742 868	2 28
Total, 1913	747 644	2 62

(a) 1913 figures. (b) Est for 1914 by State mine in tons) est by U S Geol Surv: 1913, 570 048 125; 1914,

Prevention of Coal Mine

Probably 90% of mine accidents are preventa recklessness, negligence, absent-mindedness, or i remainder are due to unintentional, careless acts ficiencies in equipment and organization. Follow dents in 1911–14 and preventive means is in the

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umber of Men Killed at Coal Mines in U S, 1911 to 1914, with fatalities classified (U S Bureau of Mines)

use	Number killed				Percentage of grand total			
	1911	1912	1913	1914	1911	1912	1913	1914
nd by:								
oal, rock, etc)...	1 173	972	1 060	985	43.14	41.19	38.06	40.19
pillar coal.....	148	179	204	145	5.44	7.58	7.33	5.92
locomotives.....	393	362	424	380	14.45	15.34	15.22	15.50
and burning gas	99	164	91	331	3.64	6.95	3.27	13.50
osions.....	280	137	423	17	10.30	5.80	15.19	0.69
emature blasts, uffocation by xplosives, flying lasts, etc).....	134	133	138	146	4.93	5.64	4.96	5.96
m mine gases...	8	10	33	14	0.29	0.42	1.18	0.57
ock or burns)...	87	76	80	89	3.20	3.22	2.87	3.63
.....	7	7	9	7	0.26	0.30	0.32	0.29
es.....	15	14	24	20	0.56	0.59	0.86	0.82
rned, suffocated, ...	80	11	4	7	2.94	0.47	0.14	0.29
.....	49	54	72	56	1.80	2 29	2.59	2.28
inside of mines	2 473	2 119	2 562	2 197	90.95	89.79	91.99	89.64
y:								
hafts or slopes..	42	28	50	70	1.54	1.19	1.80	2.86
down shafts or	4	5	1	5	0.15	0.22	0.04	0.20
.....	4	4	9	12	0.15	0.16	0.32	0.49
.....	13	17	2	1	0.48	0.72	0.07	0.04
by shaft acci-	63	54	62	88	2.32	2.29	2.23	3.59
face by:								
nine locomotives	45	68	78	68	1.65	2.88	2.80	2.77
ock or burns)...	7	9	8	11	0.26	0.38	0.29	0.45
.....	33	30	26	26	1.21	1.27	0.93	1.06
ons or bursting	3	1	1	5	0.11	0.04	0.04	0.20
nd locomotives.	32	14	8	13	1.18	0.59	0.29	0.53
.....	63	65	40	43	2.32	2.76	1.43	1.76
on the surface	183	187	161	166	6.73	7.92	5.78	6.77
.....	2 719	2 360	2 785	2 451	100.00	100.00	100.00	100.00

caused 38 to 43% of the total fatalities. Most of these acci- o insufficient propping near the face. (In collieries posts are) The number would be greatly reduced by SYSTEMATIC, LOSE SPACING OF TIMBERS, without regard to the apparently safe of. This is general practice in European collieries, and the stified by their low accident rate from falls of roof, particularly ance and Belgium. ROOF should be FREQUENTLY TESTED by METHOD (striking roof with bar or sledge with one hand and ers of other hand to see if portion struck vibrates). This super- uncertain method of judging by the sound. Important factors falls are: SYSTEMATIC DELIVERY OF TIMBERS at each working constant supply of the right kind; employment of ENOUGH pect and instruct the miners, to see that orders to put up props carried out, and to withdraw the men when necessary (37).

Falls of face or pillar coal (akin to roof fal
total fatalities in 1911–14. These accidents are
dipping seams, as in Penn anthracite mines, wh
of the overhanging coal face and the PICKING A
SLABS OF COAL. They also occur in the longwall
in undercutting by hand (pick), the miner lyin
"SPRAGGING" or "BLOCKING" OF UNDERCUT COA
is especial danger unless all loose coal is removed,
can not be reached from the floor, strong ladders o

Mine cars and locomotives caused 14.4 t
1911–14. In level workings men may be run ov
walls on main haulage roads. These accidents a
ing GOOD MANWAYS and compelling their use; AM
along one side; REFUGE HOLES PROPERLY WHITE
of track, and at regular intervals not exceeding 5

Where haulage ways must be used as manways, it i
to have incandescent LIGHTS at regular intervals, in cas
Locomotives or trains should have good HEADLIGHTS, a
system (easily installed for electric haulage). Many
ging" cars, which would be prevented by providing B
mule haulage by riding in front and standing on the t
the train; if necessary to ride in front, have a MOVABL
Accidents while coupling cars, due to the short bumpe
project at least 10 in, to provide 20 in clear space betwe
ings, in which the cars are dragged to the face by mul
HINGED CAR-BLOCKS should be used to prevent runaw
ROADWAYS are important safeguards.

Gas explosions and burning gas caused in
total fatalities. These figures are related to t
(Art 14), which show percentages varying from
in 1913. Statistics of the different states taken
Federal Bureau, but the inspection department
figures and classifications. In some states if an
damp, it is termed a "gas explosion," even thou
coal dust (Art 14). No conclusions regarding t
drawn from the individual figures. In 1914, ou
thracite district, 44 men were killed by gas explos
7.4%, or about the aver annual rate. As anth
and as the anthracite district is much more gaseo
tricts of the U S, the true proportion of gas fatali
country is probably less than 7%.

Firedamp accidents are mostly preventible by care
panies. The chief factors are: FREQUENT INSPECTION
men and GOOD VENTILATING METHODS (including large
room breakthroughs or crosscuts; use of line brattices,
in place of doors; use of double and triple doors; and
doned but open workings should be well ventilated,
securely walled off. Walled areas giving off firedamp
by drill-holes from surface. In gaseous mines, regular
valuable in controlling ventilation. In mines where
classified as gaseous or not) SAFETY LAMPS or PERMI
should be used (Art 9, 10). Mixed lights in the same
safety. PERMISSIBLE EXPLOSIVES (Sec 4, Art 9) should b
sives (black powder or dynamite). Machine-mining in
by EXPLOSION-PROOF MOTORS. Electric-trolley locomot
ing places or return ventilation entries of gaseous mi

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l at all. Explosion-proof storage-battery and gasolene locomotives used in Germany, but this protection in using such locomotives has duced in the U S. (For U S Bureau of Mines publications on the ubjects, apply to Director Bureau of Mines, Wash, D C.)

m explosives, other than those leading to gas or dust explo- 1911–14 4.9 to 6% of the total fatalities. They are chiefly premature blasts, due to accidental ignition of black powder utting fuse too short to allow time to get out of danger, forget- st fuse while lighting others, or getting tools out of the way; tamping bar. Only wooden bars should be permitted. Even rs are unsafe; (*c*) returning too soon to investigate apparent electric firing, a lead wire may come in contact with a coal- under power, or a power line, or with an unprotected dry cell y.

s safer than fuse, provided that, where electric coal cutters are em- the leads are kept in contact and covered with insulating tape until the machine and its leads. If electric firing is by battery, the latter ge) should be so constructed that the binding posts are not in circuit ring-opening button or lever has been pressed to close the circuit. AGNETO BLASTING MACHINES are preferable for safety. Use of power s to trolley lines is dangerous. For electric firiag from the surface, parate insulated circuits, and switch cutouts at mouth of every entry to prevent accidental stray currents entering the firing circuit (38). tically prevents accidents from misfires, if a delay of not less than 5

sfires may be necessary in coal mining, because if another hole y and shot, the first charge might be set off by concussion or g less burden blow out and ignite coal dust. It requires care, p is in outer end of charge. If another hole is drilled, every- outbye should be wet down before shooting.

gases from explosives in a well-ventilated colliery is rare, and nless coal or a pocket of firedamp is ignited, thus adding afterdamp or unless explosives are burned instead of detonated, by using too Art 10). REMEDY is to delay entering the working place until the t.

flying pieces of rock or coal from blasting are due to care- part of the victim in not getting to a place of safety, or the adjacent working without notification, or in not guarding the orking place after lighting a fuse, or while connecting up the ther point. Where workings have several entrances it is well iners with PAINTED WARNING BOARDS, to be placed across or trances not guarded by the shot-firer.

n mine gases caused 8 to 33 fatalities annually in 1911–14, or only he total. Except in case of explosions and fires, which are not in- tle excuse for such accidents. They are generally due to entering nnecessarily. To prevent careless miners from so doing, such places ventilated or permanently wall off.

m electricity (shock or burns) caused 76 to 89 fatalities -14, or 2.87 to 3.63% of the total. Some were from touching trolley or power lines, some from improper insulation and in- es and motors.

accidents might be prevented by safeguards, such as: trolley guard- wire is less than 6.5 ft above the roadway; guarding by boards or er cables or preferably by rubber covering and sometimes armoring; o prevent grounding of wires. In working places or traveling w

trolley and power-line potentials should be limited
cutters should be so designed and installed that a stro
the frame. Cable leads should be of the single, con
being outermost to avoid dangerous shocks from gro
AND USE OF ELECTRIC EQUIPMENT in mines, as in fo
and as formulated by the Bureau of Mines (in press),

Accidents from animals killed 7 to 9 men annu
underground stables, with passageway in front of as
prevent some of these accidents; but above all, PROP
ABUSE would prevent the animals from becoming vici
one made so by abuse, should not be used in a mine.

Accidents from mining machines (coal cu
24 men annually in 1911–14, are due to men
machines, or in handling them. The first-men
using machines in which as much as possible of
latter are casual accidents, to be guarded agains

Fatalities from mine fires, by burning an
1911, but only 11, 4 and 7 in years 1912 to 1914.
is the primary cause of death, because mine f
slowness in restricted passages.

Mine-fire disasters (Art 15) are fortunately rare.
the face, and are generally due to recklessness or ignor
or afterdamp from fires. In colliery workings su
flame explosives, which may set fire to the coal, or
sives, properly used, will rarely cause fires. In a lar
formerly employed to extinguish fires after black-pow
introducing permissible explosives.

Falling down shafts or slopes caused 42 to
Better gates and fences around shaft mouths, a
are the remedy. Gates should be self-closing,

Accidents in shafts from miscellaneous cause
any one year (1911–14) was 12, indicating creditable
catches and other hoisting equipment.

Surface accidents caused from 161 to 183 fatalit
7.92% of the total. The largest item is accidents by
45 to 78 deaths each year. These occur in drift or
number are in the anthracite district, 18 in 1911, 3
Practically all such surface accidents are preventable, e
installation and safety guards.

12. Accidents in Metal and Miscellan

Statistics in following tables were compiled
of Mines, from reports of mine operators (exc
came from the inspection departments). Tab
fatal accidents in metal mines, coal mines an
the more directly comparable. Fatalities in met
usually higher than in coal, but as metal miner
mining risk covers longer working time. In
average of 285 days, coal miners, 238 days; b
ters rarely kill large numbers, as in coal mine ex

Table 18 shows the percentages of fatalities f
of roof, etc, caused the largest number of death
but the proportion of these to the total is greate
hand, fatalities in metal mines from use of explo

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umber of Men Employed and Number Killed in and About nes and Quarries in the U S, in 1911, 1912 and 1913

Metal mines			Coal mines			Quarries			Total, mines and quarries		
	No killed			No killed			No killed			No killed	
No employed	Total	Per 1 000 employed	No employed	Total	Per 1 000 employed	No employed	Total	Per 1 000 employed	No employed	Total	Per 1 000 employed
165 979	695	4.19	728 348	2 719	3.73	110 954	188	1.69	1 005 281	3 602	3.58
169 199	661	3.91	722 662	2 360	3.27	113 105	213	1.88	1 004 966	3 234	3.22
193 088	683	3.54	747 644	2 785	3.73	106 278	183	1.72	1 047 010	3 651	3.49
176 089	680	3.86	732 885	2 621	3.58	110 112	195	1.77	1 901 086	3 496	3.43

ıtalities from Different Causes in and About Metal Mines, nes and Quarries in the U S, in 1911, 1912 and 1913

ıe	Cause of death (percentages of total)							
	Falls of overburden, roof or mineral	Explosives	Haulage and handling	Falls of person	Electricity	Mach'y	Gas and dust explosions	Other causes
s.....	32.52	11.23	7.63	15.39	2.45	3.16	....	27.62
.....	48.58	4.93	17.54	1.54	3.46	1.77	13.94	8.24
.....	25.53	26.60	20 21	8.51	1.06	9.04	....	9.05
s.....	34.82	14.21	9.99	13.91	3.78	3 78	....	19.51
.....	48.77	5.64	19.11	1.19	3.60	1.86	12.75	7.08
.....	23.94	22.07	22.07	8.91	1.88	10.80	....	10.33
s.....	36.16	13.04	11.42	12.31	3.80	3.94	....	19.33
.....	45.39	4.96	18.63	1.80	3.16	1.79	18.46	5.81
.....	20.77	24.04	18.04	9.29	2.73	10.92	....	14.21

ng and transportation are in far greater proportion in coal those from "falls of person" in metal mines are large enough es in coal mines from explosions.

n-fatal accidents. Table 19 shows a much wider variation han in the fatal accidents in different kinds of mining. Prob- of reporting the seriously and slightly injured, and the custom er for slight injuries, which prevails in some districts, are each e for apparent inconsistencies. The death rates are reason- ith one another for the 2 years reported.

nts in placer mining are naturally rare. In Alaska and 3, the death rate per 1 000 employed was 2.24. Number of ka, 3 500; Cal, 1 742. Aver number days worked: Alaska, Number killed: dredging, 7; hydraulicking, 1; underground, 7.

the metal mines of the principal countries of the world ate either more carelessness in the U S and the Transvaal untries, or more dangerous natural conditions. Probably the of inexperienced miners, and of those who do not understand ither the foreman or their fellow workmen, partly accoun

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Table 19. Number of Employes and Number Killed and Injured in and About All Metal and Miscellaneous Mines (except Coal Mines) in the U S, in 1913 and 1912

Kind of mine	Active mines	No of employes			No killed					No seriously injured. Time loss 20 days or more					No slightly injured. Time loss less than 20, but more than 1 day				
		Underground	Surface	Total	Underground(a)	Rate per 1 000	Surface	Rate per 1 000	Total per 1 000	Underground(a)	Rate per 1 000	Surface	Rate per 1 000	Total per 1 000	Underground(a)	Rate per 1 000	Surface	Rate per 1 000	Total per 1 000
1913																			
Copper	694	39 917	16 222	56 139	203	5.09	33	2.03	4.20	1 833	45.92	362	22.32	39.10	9 105	228.10	2 063	127.17	198.93
Gold and misc metal	4 749	46 143	15 170	61 313	180	3.90	30	1.98	3.43	508	11.01	124	8.17	10.31	2 817	61.05	406	26.76	52.57
Iron	206	29 384	21 749	51 133	122	4.15	43	1.98	3.23	2 009	68.37	811	37.29	55.15	7 855	267.32	2 798	128.65	208.34
Lead and zinc (Miss Valley																			

ber of Men Employed, Number of Days Worked and Number ooo Employed in and About all Metal and Miscellaneous Mines in 1912

te	No active mines	No employed	Aver no days worked	Number killed					
				Underg'd (a)	Per 1 000	Surface	Per 1 000	Total	Per 1 000
.....	20	4 827	270	26	9.39	7	3.40	33	6 84
.....	30	1 914	341	11	7.04	10	28.41	21	10 97
.....	479	15 591	302	63	5.44	4	1.00	67	4 30
.....	11	404	257	0	.....	1	40.00	1	2.48
.....	1 048	10 312	276	27	4.38	13	3.14	40	3 88
.....	624	8 892	299	44	6.49	4	1.89	48	5.40
.....	5	259	297	0	.....	0	.....	0	.....
.....	24	4 461	296	0	.....	6	1.42	6	1.34
.....	29	515	218	0	.....	0	.....	0	.....
.....	639	6 229	272	26	5.87	3	1.47	29	4 66
.....	8	440	301	1	3.07	0	.....	1	2.27
ana and West									
.....	2	591	362	0	.....	0	.....	0	.....
.....	12	200	244	0	.....	0	.....	0	.....
.....	50	831	234	2	3.26	0	....	2	2 41
.....	14	460	187	1	9.43	0	.....	1	2.17
.....	4	72	225	0	.....	0	.....	0	.....
.....	19	329	271	0	.....	0	.....	0	....
.....	4	58	260	0	.....	0	.....	0	.....
.....	79	29 469	304	88	4.11	8	0.99	96	3 26
.....	43	16 559	287	28	3.28	22	2.74	50	3 02
.....	318	11 403	267	32	3.75	4	1.39	36	3 16
.....	405	13 340	298	44	4.23	6	2 04	50	3 75
.....	554	7 547	295	16	3.11	18	7.49	34	4.51
e.....	7	90	187	1	58.82	0	.....	1	11.11
.....	10	1 633	282	11	8 69	2	5.45	13	7.96
.....	163	2 502	268	10	7.76	3	2.47	13	5.20
.....	39	2 710	292	16	8.32	1	1.27	17	6 27
.....	70	1 068	201	1	1.37	0	.....	1	0.94
.....	7	310	246	0	.....	1	20.83	1	3.23
.....	30	541	209	1	3 17	1	4.42	2	3 70
.....	260	1 052	189	1	1 68	0	.....	1	0 95
.....	38	762	251	2	19.23	0	.....	2	2 62
.....	7	1 041	275	0	.....	0	.....	0	.....
.....	63	1 963	300	3	1.83	3	9.32	6	3 06
.....	36	3 480	253	5	4.40	7	2.99	12	3.45
.....	8	386	277	0	....	0	....	0	....
.....	336	8 458	302	32	5.52	9	3.38	41	4 85
.....	11	181	250	2	22.22	0	.....	2	11.05
.....	33	2 516	274	2	1.87	2	1.38	4	1 49
.....	277	1 307	151	2	2.33	2	4 45	4	3 06
.....	77	3 834	271	20	7.22	2	1 88	22	5 74
.....	74	662	236	4	14 93	0	.....	4	6 04
.....	5 967	169 199	287	522	4.74	139	2.35	661	3 91
.....	5 232	165 979	282	542	5.18	153	2.49	695	4.19

ose killed in shaft accidents. (*b*) Placer mines not included. Snow
. (*c*) Fatalities and employes reported by State mine inspector. Al-
egation of underground and surface employes estimated from returns

the high rates in the U S, both in metal and coal m
miners are largely untrained negroes. In the U S
inspection by state authorities has undoubtedly
safety in mining, but there are indications that

**Table 21. Number of Men Employed in M
Countries, Showing Fatality Rate p**

Country	1910			1911		
	Employed	Killed	No killed per 1 000	Employed	Killed	No killed
Australasia:						
New South Wales	19 369	29	1.50	19 360	35	1.
New Zealand	8 121	15	1.85	7 400	5	0.
Queensland	12 342	18	1.46	11 031	10	0.
Tasmania(*b*)	5 770	8	1.39	5 247	4	0.
Victoria	16 553	12	0.72	14 051	19	1.
Western Australia	16 279	27	1.66	15 428	36	2.
Austria	19 621	16	0.82	19 934	10	0.
France	27 183	86	3.16	29 998	85	2.
Germany	93 645	141	1.51	92 964	148	1.
Great Britain(*c*)	28 676	43	1.50	29 025	43	1.
Italy	49 995	59	1.18	...	...	
Japan	81 728	118	1.39	80 896	142	1.
Peru	20 410	118	5.78	...	...	
Portugal	6 554	11	1.68	7 484	10	1.
Spain(*d*)	155 182	256	1.65	143 863	180	1.
Transvaal	205 687	882	1 29	225 538	934	4.
United States	....	....	....	165 979	695	4.

(*a*) Compiled from official reports. (*b*) Figures
coal mines and smelting works. High rate for 191
(*c*) Cover only mines coming under the "Metalliferou
cover coal mines in which about 25 000 men are emp
ploying about 25 000 men annually.

Prevention of accidents in metal and mi
in the order of causes in Table 22, and referrin

Falls of rock or ore from roof or wall caused 3
than from any other source. Principal remedies: p
of ore or hanging wall, after blasting, and timberin
sumed. Occurrence of accidents from falls of groun
mining. Also, if the deposit is extensive laterally, t
tioned to prevent collapse of workings.

Falls of rock or ore while loading at working
in the iron mines, and a general average of 1.46%; e
or forepoling in weak ground, before mucking or dril

Timber or hand tools. Carelessness in handlin
Many such accidents are preventable by selecting ca

Explosives accidents, 11.28%, are much comme
16.67% of the total fatalities in the lead and zinc
from this cause. Most of them are readily preven
dling explosives and detonators on surface and unde
thawers (39) (Sec 4, Art 6, 7); using wooden tampin
sinking shafts or winzes; not returning to face too s
sives producing a minimum quantity of poisonous ga

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ntages of Deaths in and About All Metal and Miscellaneous xcept Coal Mines) in the U S in 1911, 1912 and 1913

eath	Killed							
	Copper mines	Gold and misc metal mines	Iron mines	Lead & zinc mines (Miss Valley)	Misc mines	Aver for all mines 1911	Aver for all mines 1912	Aver for all mines 1913
from roof	32.20	32.38	31.52	38.10	26.67	27.48	32.10	32.21
ore while	0.84	0.48	4.24	....	....	....	0.30	1.46
ools.....	0.43	1.43	1.82	2.38	....	1.44	0.90	1.17
.....	9.74	16.67	6.67	16.67	3.33	8.92	11.04	11.28
.....	8.47	0.95	7.27	2.38	....	2.88	5.29	5.13
e, winze, ...	10.59	10.00	2.42	2.38	3.33	6.76	6.50	7.62
chute or	2.55	0.95	1.21	....	....	2.01	1.51	1.46
.....	2.55	2.38	2.42	2.38	....	2.16	2.87	2.34
.....	....	....	....	....	....	5.32	0.15	....
ral gases)	....	1.43	....	....	....	1.15	0.61	0.44
.....	....	....	....	....	....	1.73	0.76	....
.....	5.03	4.75	1.82	2.38	3.33	2.88	1.81	3.95
d'g'd.....	72.46	71.42	59.33	66.67	36.67	62.73	63.84	67.06
t.....	3.81	4.77	3.64	2.38	....	8.20	6.05	3.81
wn shafts.	2.96	2.38	1.21	2.38	....	2.59	1.82	2.20
.....	....	....	1.21	....	....	0.58	0.30	0.29
.....	0.43	....	0.61	2.38	....	....	....	0.44
.....	3.81	6.19	6.67	2.38	....	....	5.60	4.98
.....	2.55	0.95	1.21	....	....	3.89	1.36	1.46
shaft.....	13.56	14.29	14.55	9.52	....	15.25	15.13	13.18
notives...	0.43	0.95	2.42	4.76	....	0.58	0.91	1.32
comotives	0.84	0.48	....	2.38	....	1.15	0.76	0.58
.....	0.84	0.48	0.61	2.38	....	0.29	0.76	0.73
.....	1.27	2.38	0.61	4.76	....	1.15	1.66	1.61
.....	0.43	4.28	5.45	....	3.33	6.47	4.84	2.93
surface..	3.81	8.57	9.09	14.29	3.33	9.64	8.93	7.17
accidents ining	10.17	5.72	16.97	9.52	60.00	12.38	12.10	12.59
	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00

nts. Number of deaths varied from 8.47 and 7.27% in the large nes, using electric or mule haulage, to only 0.95% in gold and mis- where hand tramming is the rule. Best safeguards for mechanical drifts with small clearance, are: Keeping refuge holes well white- es on cars, and, on heavy grades, blocking of wheels of standing cars; veling ways; and strong, not dazzling headlights on locomotives.

e, winze, raise or stope caused 10.59% of the total accidents in generally have steep dips, but only 2.42% in iron mines. The small om this cause in open-pit and shallow iron mines more than offsets ich mines. Safeguards: grizzlies over chutes, fences and gates around hutes, and proper warning boards.

chute or pocket caused 1.46% of the deaths. These accidents are in steep veins, but well-designed chutes and gates are important.

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Electrical accidents underground caused 16 d
is about the same as in coal mining, in which electri
ing of trolley wires in the more constricted passage
Wires should be well guarded by side boards, especia
trolley-wires should be placed on the side of drift op
ing the ore is liable to strike the wire with his bar.
chutes on trolley roads. Power cables should be i
the voltage limited to 275 volts. Electrical rules pr
of America, and the U S Bureau of Mines should be

Mine fires. While their aggregate cost in metal
Though in 1911 there were several serious fires, caus
in 1912, and none in 1913 (Art 15).

Suffocation from natural gases is relatively r
infrequent in U S, except in coal mines. In 1913,
in Nev. In 1912, there were 4 and in 1911, 8 deat
the best preventive.

Inrush of water caused 12 deaths in 1911, 5 in
accidents are preventable (see Art 16).

Falling down shafts, causing 57 deaths in 1911
not occur. Safeguards are good fences and autom

Objects falling down shafts caused 18 deaths
These accidents are largely preventable by: smoot
loading cars or skips; dumping clear of shaft mouth
shaft and shaft house; tight or wire-mesh fences
and gates far enough back from shaft to prevent mer
of objects on men on cages or buckets can be preve
on crossheads over buckets or skips used for hoistin
or cages should have doors, as in some Mich mines.

Breakage of hoisting ropes causes few accide
great number of man hoists in use. In 1911, 4 liv
(all in Mich iron mines).

Overwinding caused 3 deaths in 1913 (none in 1

Skip and cage accidents cause more deaths t
1912, 34 in 1913. Some of these occurred in all
notable that, though hoisting by bucket without g
zinc district, only 1 death from this cause occurre
construction are entirely unnecessary. Men are pr
in absence of fences and gates at landings; or, whe
body may extend beyond the edge. Excessive cro
should be prohibited. Man cages should have inclo

Surface accidents directly connected with und
in 1911, 59 in 1912, 49 in 1913. Most of these are
by individual care and the use of warning bells, safet
figures do not include accidents in surface mining, wh
ing in Minnesota iron mines, and 14 in Florida open-
haulage. Next in importance in open-pit mining ar
caused 22 deaths in 1911, 5 in 1912 and 28 in 191
tirely is difficult. Proper methods of mining, care in
supervision are of prime importance.

14. Colliery Explo

The number of deaths annually from coal
steadily until 1907, when 1 148 men were kill
at Monongah, West Va, the worst that has occ
destructive of all explosions was in the Cour
resulting in 1 100 deaths. Since 1907, there
from this cause in the U S, both in the propo

ne rate per 1 000 employes (Table 23). This improvement is the recent general campaign for mine safety.

n of explosions into firedamp and coal-dust explosions and ttempted by different State Inspection departments. Hence, have been similarly compiled by the U S Bureau of Mines. agreed basis in this classification; some inspectors classify an originating cause, others by its means of propagation (usually ropean statistics also do not generally distinguish between ex- nd dust, and both agencies are often concerned. Only in the nthracite district are explosions definitely from either gas or e anthracite dust does not itself propagate an explosion. There- logical basis of classification appears to be the mode of pro- than the cause, no distinction is made in Tables 23 and 24.

lled in Gas and Dust Explosions in U S Coal Mines, Contrasted with Fatalities from all Causes

Total killed			Per 1 000 employed	
xplosions	All causes	% of total from explos	Explosions	All causes
1 148	3 197	35.9	1.75	4 88
469	2 449	19.1	0.70	3.64
449	2 668	16.8	0.67	4.00
518	2 840	18.3	0.71	3.92
379	2 719	13.9	0.52	3.73
301	2 360	12.7	0.40	3.27
514	2 785	18.4	0.68	3.73
348	2 451	14.2	0.47	3.30

xplosions. Blown-out or overcharged blasts of long-flame ex- discharge of these in the open, igniting either gas or coal dust flashes; flames of open lights, or defective safety lamps; mine ignite gas and in turn coal dust; and lastly, but very rarely, ossibly be ignited by sparks from flinty rocks falling from roof steep breasts or chutes, striking on each other, or on iron.

t explosions are undoubtedly caused by ignition of gas or dust through es, by open lights, or electric flashes from trolley wires and power cir- explosions, like those at Monongah, W Va, Hanna, Wyo, and Stark- een attributed to electric ignition of coal dust stirred up into a cloud l wreckage of trains of mine cars. Others have been attributed to of gas by mine fires, where the ventilation has been cut off by falls of of stoppings. (Art 15.)

of explosions. Firedamp was formerly believed to be the ne explosions, but bituminous and lignitic dust are now known ortant means of propagation. CH_4 rarely exists in explosive ughout large areas in mines, even in gaseous districts, because efficient ventilation of coal mines. Therefore, though a number nia anthracite mines are among the most gaseous in the world, ent explosions are localized, while in some practically non- ous mines explosions have swept through miles of entries. all important explosions are given in state inspection annual gn government reports, also in *Coal Age*, *Coll Eng*, *Coll Guar- oal Trades Rev*, etc.

d coal dust make a dangerous combination, each aiding the ; and propagating explosions; but, BITUMINOUS AND LIG-

Table 24. Explosion Disasters in Leading C which Over 100 Men

(Except in the first disasters reporte

United States		No killed
1839, Mch 18	Black Heath, Va.	40
1884, Mch 13	Laurel mine Pocohontas, Va.	112
1891, Jan 27	Mammoth, Mt Pleasant, Pa	109
1892, Jan 7	No 11, Krebs, Okla	100
1900, May 1	Winter Quarters No 1 & 4, Schofield, Utah	200
1902, May 19	Coal Creek, Fraterville, Tenn	184
1902, July 10	Rolling Mill mine, Johnstown, Pa	112
1903, June 30	Hanna No 1, Hanna, Wyo	169
1904, Jan 25	Harwick mine, Cheswick, Pa	179
1905, Feb 20	Virginia, Virginia City, Ala	108
1907, Dec 6	Monongah No 6 & 8, Monongah, W Va	361
1907, Dec 18	Darr mine, Jacobs Creek, Pa	239
1908, Nov 28	Marianna, Pa	154
1911, Apr 8	Banner, Littleton, Ala	128
1913, Oct 22	Dawson, N M	263
1914, Apr 28	Eccles, W Va	182
1915, Mch 2	Layland, W Va	115
Canada		
1891, Feb 21	Springhill, N S	125
1902, May 22	Fernie, B C	125
Great Britain		
1803, Sept 25	Wallsend	13
1835, June 18	Wallsend	102
1856, July 15	Cymmer, Wales	114
1857, Feb 19	Lund Hill	189
1860, Dec 1	Resea, Wales	142
1866, Dec 12	Oaks Colliery	361
1867, Nov 6	Ferndale, Wales	178
1875, Dec 6	Swaithe Main	143
1877, Oct 22	Blantyre	207
1878, June 7	Haywood Wood	189
1878, Sept 11	Aberdarne, Monmouth	268

Great	
188	
188	
188	
188	
189	
189	
189	
189	
190	
190	
191	
191	
191	
New	
Kil	
Franc	
181	
187	
188	
189	
190	
188	
Belgi	
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Germ	
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191	

DUST are highly dangerous in themselves, and in certain densities of dust cloud, will propag gas explosion; in fact, finely-divided coal du dusts, of sugar, soap, paper and cereal) when like explosive gases (1, 40, 42, 43, 47).

Ignition of firedamp. In still air, by sid at 5.5%, and Pittsburg natural gas at 5%. A in rapidly-moving currents the ignition limit lineal ft per min it is only 5%. But, such hi portions of CH_4 in strong ventilating curren

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s have occurred in some Belgian and B C mines. If firedamp ventilating current, men are ordinarily withdrawn from that ine. This should always be done (except in emergency work), ough CH_4 to show a cap in a safety lamp in a moving current, lysis there is over 2% CH_4, or, if open lights are used, when the 0.5% CH_4 (Art 6).

blowers or feeders in the face, from the floor, or from the roof, espe- have been falls. In still air it is usually given off faster than it diffuses. at the roof level, or in a roof cavity. Then, if ignited, it burns rather body of firedamp has considerable lateral extension, when it flashes elocity and violence. Men are often burned by igniting gas pockets, e explosions if enough CH_4 or dry coal dust be present. Pressure rated by the maximum explosive percentage of pure CH_4, 9.4%, is q in. Through loss of heat, not over 70 or 80 lb press is actually ob- e-compression.

f firedamp explosions. (*a*) Strong air current carried to face d line brattices; (*b*) prohibition of open lights and smoking of ligatory use of safety lamps or permissible electric lights; (*d*) ble explosives instead of black powder and dynamite (Sec 4, s for CH_4 before and after blasting; (*f*) if undercutting is done e motor should be explosion-proof; (*g*) prohibition of trolley bare power wires in gaseous sections of a mine; (*h*) daily analy- in each ventilation district, for control of ventilation; (*i*) rigid parts of mine, by fire boss, inspector or foremen; (*j*) sealing ial double walls, abandoned or unused parts of mine, difficult to ect, and if these areas produce much CH_4 issuing under high le should be drilled from surface (5, 41, 47).

losions. Investigations of the explosibility of coal dust were made in Faraday (1844), Galloway (1870–84), Hall (1876–93), Abel (1880), rrison (1878–80), Atkinson, W. N and J. B. (1886 and later), Garforth by several Royal Commissions, the last being in charge of the present station; in France, by Verpilleaux (1867), Mallard and Le Chatelier anel, beginning in 1907 at the Liévin testing station; in Germany, by p Com (1884), and later by the Westfalian and Silesian Coal Syndicate's t Dortmund and Beuthen; in Austria, by the Firedamp Com at the Rossitz various times, and recently under Czaplinski and Jicinsky; in Belgium, the government testing station at Frameries; in the U S, since 1909, by testing gallery at Pittsburg (Sec 4, Art 2), and in the Bruceton experi- under Rice and Jones (1, 43).

Definition of Terms used in Explosion Experiments

ans particles which will pass a 20-mesh sieve (Bureau of Mines empiric ced in mining and handling coal, or artificially by grinding. To obtain composition and size at different testing stations, coarse coal is ground nills, or other pulverizers.

ermed stone-dust in Great Britain), includes all inert dusts from clay, und in mines. At testing stations a uniform shale or limestone dust is ding. In mines having electric haulage much quartz sand is used to . While this dilutes the coal dust and gives some protection, since injurious to breathe, it should not be used as a substitute for shale or r general rock-dusting; moreover it is too dense.

s or coal dust (as the term is employed in Bureau of Mines experi- rting of inflammation by some initial source of heat. In case of CH_4, will pass rapidly into an explosion if there is enough CH_4 in proportion mine spaces. In case of coal dust, premixed with air which has been ion may not produce sufficient pressure or concussion to rais-

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dust, beyond the influence of the agency that brou
hence the flame will die away and it is not consider

Propagation is the continued inflammation of ga
as far as the gas or dust zone extends. It is syn
necessarily violent.

Explosion, while practically synonymous with p
erable dynamic force, advancing as a moving zone o

Inflammation in gas mixtures is transmitted
of heat by radiation and convection, and by such
expanding gases that each layer in succession of unb
batically and raised to ignition temp. Thus, when t
is almost instantaneous, and with certain gases the
velocity in tubes of uniform size and character.
Berthelot (1881), who termed it "l'onde explosive"

Detonation-wave is the term given by Dixon
He states "the rate of the explosion-wave is a defini
mixture; the wave travels with the velocity of sou
moving rapidly *en masse* in the same direction."

Retonation-wave (Dixon), "l'onde retrograde"
pression traveling backward through the spent gase

Reflection-wave (Dixon), "l'onde reflechie" (L
reflected from a closed end or other contraction o
irregular galleries these waves are numerous, and
the spent gases or air traversed.

Collision-waves (Dixon), "l'onde prolongie" (L
produced when 2 explosion-waves, generated sim
gaseous mixture, meet and extinguish each other,
in the direction of its originating explosion-wave.

Secondary waves, like the last three named, do n
tain them, and therefore gradually die away; they
merged as to be unrecognizable. Their velocities app
ing with the densities of the gases traversed and th
relative to the walls of the passageways.

Table 25. Velocities of Explosion-Waves

Kind of wave	C_2H_2 +
Explosion or detonation-wave	2 990
Retonation-wave	2 300
Reflection-wave	2 250
Collision-wave	2 050

Dixon states, if CH_4 and air are ignited at the clos
flame increases rapidly to 1 000 m (3 250 ft per sec)
not approach the rapid movement of a detonation-w

Coal-dust explosion-waves (term not used in
waves) resemble those from gas inflammations, a
are probably identical with those in gas mixtures; b
be obtained, laboratory experiments, like those
velocities, are not possible. In the large testing g
mine, dust clouds have not yet been made sufficient
eration of an explosion flame; velocities of 2 000 to

Shock-wave is a wave of compression which trav
ity varying with the density of the gas and characte

violent concussion, as from a blast, and hence is important in raising
cessive shock-waves may be sent off in advance of the slower-moving
Bruceton mine a shock-wave set up in still air by a blown-out shot
rts, at about 1 200 ft per sec, its velocity gradually decreasing to less
c in a distance of 1 300 ft.

ve, "pioneering-wave" (British), is a compression-wave or a series
ead of the flaming zone of an explosion. It makes possible the propa-
t explosion by bringing the dust into suspension.

wave of depression, is the flow of air at atmos press to fill the partial
when the expanded after-gases of an explosion cool. Depressions of
have been recorded in the Bruceton tests (43), and in a larger mine
would doubtless occur. Return-waves, being thus limited in press
nt than primary or even secondary waves, but move light objects
It is possible that in an explosion in a large mine an inward rush
entry before the heated gases cease discharging from another, pro-
es are disconnected, except in the interior of the mine.

5) **knowledge of coal-dust explosions,** as determined by
explosion disasters and at testing stations, such as the experi-
iceton, Pa:

ith a ratio of volatile combustible to fixed carbon of more than 10
in air if the percentage of ash is not too high.

nd fineness of dust. Other things being equal: the higher the per-
-combustible the more explosive is the dust; the finer the dust the
st, in presence of much fine dust, say less than 200-mesh size, dust
20-mesh may enter into the explosion; the higher the ash and mois-
ive the dust. Pittsburg coal dust, 70% passing through a 200-mesh
60% of shale dust will not ignite from a blast of 4 lb black powder;
irce of ignition the more readily will the dust ignite; thus, if a 50-foot
ust is ignited by a blast, it will in turn ignite an adjacent dust zone
tsburg coal and 60% shale, which will then propagate the explosion.
onditions, it requires 75% shale dust to prevent propagation. With
sts less shale or inert mixture will prevent ignition and propagation.

racite dust is explosive (British tests), but LOW-VOLATILE PENN-
TE DUST IS NOT EXPLOSIVE (Bureau of Mines tests), either alone or in
. This conclusion is supported by the immunity of the Penn anthra-
idespread explosions. It is not yet determined whether this is not
harder structure of the Penn anthracite. A certain variety of the
cture and volatile-combustible ratio of 10% or more, on testing would
e an explosion, but would do so in presence of 1% or more CH_4.

CH_4 in the air, in percentage below its own explosive limit, INCREASES
COAL DUST (except low-volatile anthracites). An increase of each 1%
3 to 5% or more increase in inert matter of the coal dust; hence the
anger of igniting originating dust explosions in gaseous workings.

ity of the atmosphere, even to saturation, or wet walls and floor,
IGNITION OR PROPAGATION of a dry dust explosion, for, while presence
ir is retardant, the quantity present even at saturation is insignificant
he heat energy of sufficient coal dust to propagate an explosion. More-
eat of water vapor is but little more than that of the inert nitrogen of
also be raised to ignition temp.

ure content of the dust IS FAR MORE IMPORTANT than HIGH HUMIDITY
ture must be converted into vapor or steam, and then raised to igni-
replaces some of the air and O present. Moisture content of dust
ntial than ash content. But, pure inflammable dust is ignitible by a
ignition, unless mixed with sufficient moisture to prevent the me-
a dust cloud. In tests, pure fine Pittsburg dust, with about 20%
ted an explosion. To be safe required 30% moisture, with which
be pressed into a compact lump in the hands. Normal moisture of
about 2%. Roughly it requires an amount of water equal to about

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⅓ of the weight of dust, to prevent mechanical raisi
dangerous unless it can be brought up into the air

(*g*) **The quantity of Pittsburg dust that will** c
pletely burned, is 0.123 oz per cu ft of air at atm
when the press reaches say 120 lb, if the additional
0.123 = 0.984 oz, to consume all the O, with corres
cally, since coarse grains are always present (whic
to use more than the theoretical weight to obtain m

(*h*) **Excess dust, if present in the air, retard**
practical importance, for, even if the explosive wave
the floor and the explosion flame, though moment
is equally destructive of life.

(*i*) **Minimum quantity of pure fine (200-mes**
plosion. A test reported by Taffanel, on the mos
showed 0.023 oz per cu ft of air, and in 2 tests by
cu ft. These are merely indications for particul
will theoretically consume all the O in 1 cu ft of air a
explosion, though not the most violent, because no
mixture with air never being perfect. In a heading
oz dust per cu ft, there would be 54 × 0.12 = 6.5
evenly on ribs, roof and floor, the thickness of the du
ness of writing paper). There is usually much mor

(*j*) **Velocity of a coal-dust explosion.** It start
from 0.5 to 1 sec to traverse the first 100 ft, or poss
ash dusts. With pure dusts the second 100 ft may
per sec, and the third 100 ft at over 1 000 ft per
exceed 2 000 ft per sec.

(*k*) **Pressures of coal-dust explosions** are to s
If the origin is a blown-out shot (of say 4 lb black po
mentary local press of 10 to 15 lb per sq in, falling t
then succeeds a slight depression, followed by a slowl
to only 5 lb at end of first sec. In a pure-coal dus
in press. At the Bruceton mine, 350 ft from the o
tered, at 550 ft 73 lb and at 750 ft 119 lb. At thi
distances from the origin, when the intensity of the
pressures have been reported in the Altofts and t
not rise steadily, but by violent pulsations. Wi
backward through the spent gases toward the orig
provides a continuous circuit into another part of
violent, throwing loaded cars and other objects towa
when the gases cool and produce a depression, caus
This is a slow wave of low press, probably rarely ove
light objects toward the origin or beyond it, followi

(*l*) **Coked or caked coal dust.** An explosion
usually leaves much coked material in its path.
thrown against the faces of timbers or projecting
gains headway coke may be found both facing and
high velocity but little is deposited, and then only
cesses. But, variations occur, due to retonation-w
of the inrush of dust-laden air. Globular coke is p
transit; usually found only in a *cul de sac*, or stub
when the flame has lingered, especially in gaseou
roof or ribs are also coked. Partial coking gives a

(*m*) **Tracing origin and course of an explosi**
dence is infallible. In extensive explosions, all data
gas ignition is the prime cause it is particularly diff
for if the gas is stratified along the roof, as it may
it may burn several hundred feet along the roof bef

from a dust explosion. Explosion of gas or dust is essentially the
of complex hydrocarbons, under complicated and rapidly varying
s, with deficiency or excess of carbonaceous matter. In violent explo-
ne pre-distillation of gas by radiant heat or by adiabatic compression.
re likely to occur in slow-moving explosions; in rapid ones the fine dust
s a whole, though coarser particles continue to burn after the fore-front
as passed.

pical Analysis of Gases Taken by Automatic Samplers at the nceton Mine, 350 Ft from Origin of Explosion (43)

Time interval, sec	CO_2	CO	O_2	CH_4	C_2H_4	H_2	N
0.00*	8.15	5.30	6.51	1.10	0.60	3.15	75.19
0.31	4.44	3.23	12.43	1.01	0.10	1.62	77.17
0.83	7.71	7.11	5.55	1.51	0.25	3.16	74.71
0.00*	13.56	5.36	1.03	1.15	1.31	2.38	75.21
0.55	7.93	5.14	7.05	0.97	0.12	2.45	76.34
3.00	9.98	7.38	2.60	1.51	0.25	3.52	74.76
0.00*	9.02	7.34	2.84	2.05	0.55	3.51	74.69
0.37	9.61	9.59	1.03	2.35	0.55	5.45	71.42
1.57	8.64	6.53	3.52	2.01	0.20	3.42	75.86

ng foil opposite sampler, the first sampler took 0.03 to 0.04 sec to close
foil. The second and third samplers operated by clockwork started by
nd thus breaking an electric circuit.

of coal-dust explosions. All the rules for prevention of fire-
s (Art 14) are equally applicable to coal-dust explosions, because
are caused by ignition of a pocket of CH_4, or more generally by
nall percentage of CH_4 in the air. The bad practice of shutting
t shot-firing time is prevalent in the middle west and southwest.
ne makes any firedamp, even if less than can be detected by
mp inspection, the danger of igniting dust is increased; whereas
ceton mine show no difference in igniting dust without gas, in still
ent. Most experiments have been made in practically still air.
other treatment, very little dry pure dust is required to propa-
on, and gangways can not be sufficiently cleaned of dust to insure
er, it is advisable to remove frequently from haulageways both
coal, to prevent the traffic making more dust.

There are two methods: the dry, to make the dust non-ex-
t, to lay it so that it can not be brought into suspension.

method is to spread 4 parts shale or limestone dust for each 1 part of
t, as subsequently checked by analysis. Haulageways and passages
oroughly and the rock dust thrown on roof, ribs, and floor, about 5 lb
ing required for first treatment in entries about 9 feet wide by 6.5 feet
es of this method: the rock dust is visible and does not change from
the degree of protection afforded is readily checked by rough analysis
unsuitable if coal is constantly spilled from mine cars, but if cars are
erloaded it is a good method. It is being extensively applied in many
and is approved by the Royal Commission on Explosions in Mines.

od consists in laying dust by watering, so that it will not be raised by a
t modes of watering are: (*a*) liberal hosing; (*b*) thorough wetting by
kling cars; (*c*) high-press fixed sprinklers, at close intervals on con-
, so that when turned on all surfaces will be wetted; (*d*) humidifying
rrent with exhaust steam from the fan or other engine, both to prevent

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drying out the mine, and by supersaturating to car on the dust on ribs and roof (1). Method (d) may sprinkling inside the mine. $CaCl_2$ is a deliquescer ways (preferably in granulated form) assists in pa ally dry mine, but is not sufficient in itself to allay t used, but is not effective, as pure NaCl is not deliq

Rock-dust and water barriers have been teste ton mine and in the Liévin gallery. They are used devices at certain critical points, and at entrances original TAFFANEL BARRIER consists of 10 to 15 sh near but not touching the roof, placed about 7 ft a They have proved to be effective in almost all test EASILY OVERTURNED WATER TROUGHS and tanks h because, if tripped by an advance air wave, the wa flame reaches the point. RICE ROCK-DUST BARRIER vanes set to operate at certain air velocities or pr rock dust in a dense cloud. They are made to o blow dust from open shelves. For mines where w they may be used without wetting the rock dust a pose. A number of types for use in different loc favorable results at the Bruceton mine, are ter barrier," "trough barrier," "rock-dust stopping," dust protected overcast" (40, 43, 47).

15. Mine Fi

Fires in coal and other mines containing next in importance to mine explosions, involv erty. Some mines have been completely ru not pay to attempt recovery. Though few m of fire, the risk is minimized by proper care, g construction at exposed or critical points (44,

Hundreds of fires have occurred, in coal and m lives have been lost, but in which the aggregate d lars. In the Lake Superior iron district alone, fro causing 22 deaths.

Causes of fires, common to all kinds of mi timbers, cloth brattices, wood stoppings, hay in st oils, and saturated cotton waste. Ignition of oily boards, by MATCHES, CIGARETTES, AND BURNING T FECTIVE ELECTRIC WIRING; incandescent lamps SMALL FIRES UNDERGROUND for heating lubricating flame of KEROSENE OR GASOLENE TORCH igniting HOT ASHES in open pits connected with undergroun naceous material or timbers. Fires from this ca region have cost vast sums to control. It is a serio coal beds, and where the caving system is used in contact with dry timber or carbonaceous material. or by weak detonators. FRICTION OF WIRE ROPES have stuck; ignition perhaps assisted by sparks or other iron. OVERHEATED BEARINGS of machine machinery like underground fans. Fans should causing fire in wood structures at top of shaft, in shaft. SPARKS FROM LOCOMOTIVES passing near s DUST EXPLOSION throwing hot coked dust on timbe ment of air current, fires start up. SPONTANEOUS occur in underground rooms containing oily waste by slow oxidation in gobs and working places; i sulphide ores, fires may originate by the sliding on the footwall, grinding together of the ore aid

iery Fires Causing Loss of 10 or More Lives, and Some Recent Metal Mine Fires

Name of mine	Place	Lives lost	Origin of fire
Avondale	Plymouth, Pa	179	Not reported
Hill Farm	Dunbar, Pa	51	"
Neilson	Shamokin, Pa	10	"
Diamondville	Diamondville, Wy	28	"
Milby & Dow	Dow, Okla	10	"
Hailey-Ola, No 1	Haileyville, Okla	29	Barrel black oil set on fire in shaft bottom
St Paul, No 2	Cherry, Ill	256	Hay set on fire at foot of intake shaft
Leyden	Leyden, Col	10	Underground elec hoist room
Price-Pancoast	Throop, Pa	73	Floor of und'g'd engine room
Cleophas	Upper Silesia	104	Wooden material in brick-lined shaft
Whitwich	Leicestershire, Eng	35	Goaf fire breaking thro' stopping
Hanastead	Staffordshire "	25	Box of candles in shaft bottom
Cadder	Scotland	22	Near downcast shaft; cause unknown
Belmont	Nev	17	Probably from candle end in pile of timber at bottom of winze
Hartford	Mich	7	Defective wiring, or candle end, in shaft
Giroux	Nev	7	Probably candle end in shaft station

ap of rapid oxidation. A prolific cause of fires in collieries is the ig- by long-flame explosives; or the coal itself may take fire from the d shots, or even in ordinary use of long-flame explosives. In metal n caused by lighted CANDLE ENDS dropping onto timbers or inflam- his is particularly dangerous at shaft landings or chutes. DROPPING ore chutes, for ascertaining the amount of ore, sometimes ignites the hute.

on. Most fires are preventable. Irresponsible persons are e. The foregoing list shows that the commonest causes are HANDLING LIGHTS, or in throwing away lighted candle ends ap wicks; these are preventable by using electric lights, or lanterns, at landings, stables, and sidings. At the working tric miners' lamps in coal mines and acetylene lamps in metal han oil lamps, torches, or candles, as there is no possibility of ing thrown away. Moreover, acetylene lamps go out when r any length of time.

n the surface, where feasible. If done underground, keep the oil in ith cement floor and an emergency fire door of steel plate. If oil it with hot-water coils. Keep at hand a supply of dry sand in buckets. rdles or other combustible material should not be stored on landings, or cans at a distance.

Underground pump rooms, engine rooms, a and metal mines, should be arched or lined, and f instead of timbering. There should be fire doo be quickly closed in case of serious fire.

Underground stables (Sec 11, Art 10) should tible material. Straw and hay should be dampe taken into the mine or handled by men using ope

Explosives. Only a day's supply should be st explosives should be kept in wood-lined magazine be of non-combustible material. No open light Thawing of explosives should be done at a distan (For methods of thawing and precautions, see Sec

Incandescent electric lighting, properly inst or bottoms and in stables, is a great measure of fixed, not hung loose from cords, and the bulbs sh

Fireproof construction. Fires often start in or shaft houses of metal mines, the burning em structures should be of all-steel or reinforced concr should be massive and the space between the shaft inclosed. At mouth of shaft steel trap doors sh from a safe distance.

Entrances to slopes, entries and tunnels shoul structures. If such exist, trap doors should be p be closed by steel ropes reaching beyond the entra concreted where possible. Permanent shafts less cost be lined throughout with reinforced concrete state regulations. It is even more important to When the escape shaft is near the hoisting, or ventilating doors in connecting passageways, one to the other two, and all close automatically by from short-circuiting, and burning out both shaf doors had been carelessly left open, thus permitti the flame to the main shaft after the air shaft wa

Power lines. Steam pipes inside the mine sh in contact with timber, and have asbestos or m should be insulated, and not in contact with coal

Fires in collieries caused by ignition of ga sives can with rare exceptions be prevented by u vent ignition of gas by open lights use safety lam

Fires in caved workings of metal mines are to broken timbers with dynamite, which has a fla ignite wood, especially if imperfectly detonated b sible explosives, as for coal mining, might be used

Spontaneous fires. Those in goaves, rooms, preventable by removing all coaly material and p filling of excavations with sand or culm, as exte anthracite district, and in many German mines. of heavy sulphide ores, spontaneous fires may oc where possible to apply, hydraulic filling would means of dealing with such fires is by use of fire adjacent live workings.

Fire-fighting equipment. All collieries ber and other combustible material should pipe lines laid in and around the mine open different shaft landings. In some cases the whole mine. Such systems have been inst are used for both fire protection and wetting

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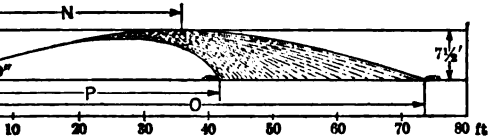
d be not over 100 ft apart; all hose taps should be uniform in size, n-pipe threaded hose connections. FIRE HOSE in 50-ft lengths, on with nozzles attached, should be kept in boxes along entries, timbered intervals of 600 to 1 000 ft. For underground use, experiments have efficient handling by one man in emergencies, 1.5-in hose is best size, cone nozzle. Best operating pressure for one man handling the hose



n Trajectory of Water Jet in an Entry 7.5 ft high. Press at Hydrant lb; Length of Hose, 50 ft; Diam of Hose, 1.5 in (45)

40 lb per sq in. Fig 15 and Table 28 give results of tests in the Bruce- ureau of Mines (45) and indicate the limitations of stream trajectories of ordinary height.

Table 28. Water thrown by Nozzle (*a*) (45)

o in above floor; curve of stream tangent to roof, which is 7.5 ft above floor; 1.5-in rubber hose, 50 ft long)

b)	*P*(*b*)	*O*(*b*)	Water thrown per min	Press per sq in at hydrant	Diam of nozzle	*N* (*b*)	*P*(*b*)	*O*(*b*)	Water thrown per min
	Ft	Ft	Gal	Lb	In	Ft	Ft	Ft	Gal
½	28	34	19¼	25	⅝	28½	35½	56½	55¼
½	31	37½	23¾		¾	28	35	56	73
	29½	35½	30½	30	7/16	31½	36½	62½	32¾
½	27	32½	34½		½	33½	41½	65½	40¼
	26½	32	48¾		9/16	32	38½	64	52½
½	32	43½	23¾		⅝	31	36	61½	60½
½	35½	47	28¾		¾	30½	36	61	79¼
	34	45	37	35	7/16	33½	37	66½	35
½	31½	42	43		½	35	42	70	43
	31	41½	58¾		9/16	34	39	68	56¼
	34½	51	27		⅝	33	36	65	65¼
½	38½	54	33		¾	32½	36	64½	85
½	36½	52½	42¾	40	7/16	35	37	70	37¼
	34	50	49¾		½	36	42	74	45½
½	33½	49½	66¼		9/16	35½	39	72	60
	36	57½	30		⅝	34½	36	69	69½
	40½	60½	37		¾	34	36	68	90½
½	38	58½	48						

ade under direction of G. S. Rice at the Bruceton mine during May, L. M. Jones and H. I. Smith.

istance from orifice of nozzle to point where curve of stream is tangent
oriz distance from orifice to point where lower part of stream strikes
iz distance from orifice to point where upper part of stream strikes
5).

re may be regulated in deep mines by automatic differen- es; or more safely, though more expensively, by a syste

neously, on account of gases and smoke on return side, it is better to topping on intake side, so the men can work in fresh air. This may most by completing with a door. Building the first stopping on the return dangerous, especially in a gassy colliery, as it backs up distilled and other mable gases over the live fire and may result in an explosion. Hence, ing should be on the intake side, and then, by carrying air to the site by ecting brattices, build the stopping on the return side, completing with an closing door.

ventilation promptly is often very important in saving life. At one dis- re, which started at foot of intake shaft, many lives would have been saved, have been quickly reversed. When it was finally reversed (after lapse of hour), the fire had reached the other shaft and a large number of lives lan should never be reversed without the knowledge and approval of the reman; hence the importance of an underground telephone system, con- surface.

notice of a mine fire the first duty of the management is to withdraw all nt needed for fire fighting. Underground alarm gongs have been tried, factory, as they are liable to derangement, and may cause a panic by ntally. An underground telephone system is safer and more useful.

e area. Stoppings should be as close to the fire as possible. If perfect, flame is soon extinguished in the inclosed area by), also in collieries by absorption of O in the coal (Art 3); then ion continues until the O is practically exhausted, with tight and a small inclosed area, this taking from say 1 week to several ding to size of fire and of area inclosed.

is impossible. Though the stoppings be airtight, air percolates through or along bedding planes, and most rocks are porous. A change in barom bove or below normal is equivalent to a difference of 13 in of water-gage ther of the inclosing pillars and stoppings. It is then a question whether d faster than it is renewed by leakage. This is tested by taking samples h pipes with valves, placed in each fire stopping. Where the nature of s of pillars permits rapid leakage, INJECTION OF INERT GAS (CO_2 or not because these gases have materially greater extinctive effect than to reduce the proportion of O below the 10 or 12% which permits active If the area is small LIVE STEAM may perhaps be used with advantage, o keep fresh air from entering, by maintaining higher press than atmos- fire area. If the area inclosed is large, as of an entire mine, injection of eam is of little value; steam is rapidly condensed, and the inert gases are the water. Moreover the volume of gas ordinarily available for injection ignificant as compared with the mine spaces to be filled. SULPHUR DIOXIDE burning sulphur, was tried at the Ziegler mine fire, Illinois, 1908, the whole closed and the gas blown in by fan. Results were inconclusive. In burn- he writer found it difficult to keep the O in the gaseous products down to CARBON DIOXIDE, produced by action of H_2SO_4 on limestone, was reported ed to extinguish a large fire in the Engleville mine, Colo. Ordinary flue lers have been proposed, but these contain too much O, unless the air feed which case CO is likely to be produced in amounts making the flue gases even explosive. If properly controlled, PRODUCTS FROM BURNING NAT- PRODUCER GAS, might be suitable for injection purposes. But, a fire s extinctive gases, and in ordinary conditions it does not pay to inject Carbon tetrachloride gas would be a good extinguisher on account s, but is too expensive for large-scale use.

ustion stops the HEAT OF A LARGE FIRE, surrounded by its own ashes, Y SLOWLY. The Monarch mine fire, Ill (1910), was sealed for a couple when the fire area was explored by men with oxygen breathing appa- ive heat was observed; but as soon as a ventilating current was started, s under heavy roof falls immediately revived and had to be dug out. In Frankfort (Ill) mine was sealed at the shaft collars on account of fire due . On re-establishing an air current, 3.5 months later, the fire revived an hour.

of water tanks in offsets from shaft or slope a
ated valves for automatic filling. When pos
be kept in elevated reservoirs or tanks, so as
of fire pumps in case of emergency. Source
must be carefully worked out for each min
insured by carrying the pipe-lines down the
airways, or by heating the water in a circuit

Automatic sprinklers are sometimes used for
bottoms, and underground stables. This system
underground workings, but has high efficiency if in
the Insurance Underwriters laboratories. Where f
filled FIRE BUCKETS, and WATER BARRELS painted r
ticularly at shaft landings and bottoms.

Emergency fire-doors at critical points
off fire areas; their maintenance cost is sma
fans and air currents is of great importance
should always be reversible. Many fires in
been controlled and loss of life prevented by
and fire doors at top of shafts.

Fire-fighting organization, ready for
Trained crews on each working shift should h
assumed points of fire. Copies of SPECIAL FI
lation, all water lines and taps, and position of
should be placed in each fire-hose box through

Where pipe lines do not run, FIRE EXTINGUISHE
that no more than 5 min will be required to reac
In some collieries large CHEMICAL FIRE EXTINGUIS
rapid transport. A good underground TELEPHONE
fires. SMOKE HELMETS, in which the air is renewed
a persistent fire, but the user can not go more than
are not well adapted for emergency work. OXYGE
is of great value in fighting fires, and is used for th
after explosions.

Fighting colliery fires. It is best to att
but if the fire still makes headway or, in case
because of roof falls, stoppings must be erec

In presence of gas, the closing of the stoppings is
able that the last stopping which shuts off intake a
pulleys for automatic closing. A counterweight is
filled with water which is allowed to trickle away
shuts the door. Or, the door can be held open by
charge of detonating caps or a few ounces of permi
rock dust to prevent ignition of possible gas. The
the ventilating current is from the backing up to
distilled gases from the coal. Under special cond
ated by fires in metal mines, particularly those w
the same precautions are to be observed. General
explosive. In collieries, if after the temporary sto
for some hours, and no explosion has resulted, pe
A pneumatic dam (Art 17) may be of assistance as

Direction and volume of ventilating curre
The current should not pass over the fire, but a
thick coal, overhead, should be brought up the ent
erected as to prevent air from feeding the fire. Th
breathing apparatus (Art 16, 19). The question
off the air by a stopping on the intake or the retu

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ely, on account of gases and smoke on return side, it is better to ing on intake side, so the men can work in fresh air. This may most completing with a door. Building the first stopping on the return ous, especially in a gassy colliery, as it backs up distilled and other e gases over the live fire and may result in an explosion. Hence, ould be on the intake side, and then, by carrying air to the site by g brattices, build the stopping on the return side, completing with an g door.

ation promptly is often very important in saving life. At one dis- ich started at foot of intake shaft, many lives would have been saved, e been quickly reversed. When it was finally reversed (after lapse of), the fire had reached the other shaft and a large number of lives ould never be reversed without the knowledge and approval of the n; hence the importance of an underground telephone system, con- ace.

ce of a mine fire the first duty of the management is to withdraw all eeded for fire fighting. Underground alarm gongs have been tried, ory, as they are liable to derangement, and may cause a panic by ly. An underground telephone system is safer and more useful.

ea. Stoppings should be as close to the fire as possible. If rfect, flame is soon extinguished in the inclosed area by so in collieries by absorption of O in the coal (Art 3); then continues until the O is practically exhausted, with tight a small inclosed area, this taking from say 1 week to several to size of fire and of area inclosed.

mpossible. Though the stoppings be airtight, air percolates through long bedding planes, and most rocks are porous. A change in barom e or below normal is equivalent to a difference of 13 in of water-gage of the inclosing pillars and stoppings. It is then a question whether ster than it is renewed by leakage. This is tested by taking samples ipes with valves, placed in each fire stopping. Where the nature of f pillars permits rapid leakage, INJECTION OF INERT GAS (CO_2 or because these gases have materially greater extinctive effect than uce the proportion of O below the 10 or 12% which permits active area is small LIVE STEAM may perhaps be used with advantage, fresh air from entering, by maintaining higher press than atmos- ea. If the area inclosed is large, as of an entire mine, injection of of little value; steam is rapidly condensed, and the inert gases are ter. Moreover the volume of gas ordinarily available for injection nt as compared with the mine spaces to be filled. SULPHUR DIOXIDE g sulphur, was tried at the Ziegler mine fire, Illinois, 1908, the whole and the gas blown in by fan. Results were inconclusive. In burn- cer found it difficult to keep the O in the gaseous products down to N DIOXIDE, produced by action of H_2SO_4 on limestone, was reported extinguish a large fire in the Engleville mine, Colo. Ordinary flue ve been proposed, but these contain too much O, unless the air feed h case CO is likely to be produced in amounts making the flue gases explosive. If properly controlled, PRODUCTS FROM BURNING NAT- CER GAS, might be suitable for injection purposes. But, a fire inctive gases, and in ordinary conditions it does not pay to inject rbon tetrachloride gas would be a good extinguisher on account is too expensive for large-scale use.

stops the HEAT OF A LARGE FIRE, surrounded by its own ashes, WLY. The Monarch mine fire, Ill (1910), was sealed for a couple the fire area was explored by men with oxygen breathing appa- at was observed; but as soon as a ventilating current was started, er heavy roof falls immediately revived and had to be dug out. In fort (Ill) mine was sealed at the shaft collars on account of fire due re-establishing an air current, 3.5 months later, the fire revived ur.

...ppliances. Some antiquated or crude and unsa...
...are those using air compressed to only a few atm...
...which CO_2 is not quickly ...
...good for distances to about 200 ...
...They are ... breathing equipment, and ...
...for rescue and explora...
...made light and s...

Characteristics of portable breathing a...

...oxygen compressed to ...
...or box; a rubber ...
...in breathing ...
...apparatus, made in ...
...countries, are of this t...
...breather. Wt, about ...
...made in England, is ...
...breather only and oxy...
...e, whereas all othe...
...have fixed feeds of 2 o...

...) Employing com...
...chemicals in loose or ...
...mesh must be shake...
...tion of CO_2; emergency ...
...Fleuss or Proto outfi...
...this type. It has a r...
...charged. This i...
...or extra supply of O, ...
...a, reducing v...
..., smoke goggles ...
...valve; i, pressure ...
...ing valve; m, pre...
...gen-supply tube ...
...breathing b...
...oxygen cont...

Employing com...
...It is re...
...unique in ...
...a device for i...
Breathin...
...ered knapsac...
...oxyg...
...absor...
...heavy ...
...across th...
...

fi
sh
ch
roc
the
distil
ated
the sa
explosiv
for some
A pneum...

Direction...
The current
thick coal, ove
erected as to pr
breathing appara
off the air by a sto...

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ns. In 1908 there were but 2 or 3 modern outfits. Only these
n found effective (to 1915) by the U S Bureau of Mines. Dr
reau, in 1915 designed a promising apparatus, with automatic
ve. First trials were favorable, but it has not yet been
d.

ents are disapproved, because of the uncertainty of getting a perfect seal
so the dead spaces of the helmet permit dangerous accumulation of CO_2.

e of apparatus (48). If not kept in perfect condition, it is
eath of the wearer. Every station should have at least 5 sets,
scue party should consist of at least 5 men, so if one becomes
e will be 4 to take him out. Men should be thoroughly trained
1 6 days, 2 hr per day, and be able to take apart and put to-
ratus. There should be a leader, paid to care for apparatus

ractice in apparatus 2 hr per week, and be examined periodically by a
eir fitness. No one should be permitted to use apparatus under dan-
without a certificate of training from the Bureau of Mines, state
r joint or private rescue station giving equivalent training. Apparatus
a room by itself, dust-free, dry and well ventilated, and not overheated,
ts will not be injured. It should be cleaned immediately after use,
oper place ready for emergency calls (48, 49, 50, 51).

cue apparatus at high altitudes made (1915) by Bureau of
s Peak, Colo, at from 7 000 to 14 000 ft above sea-level, show
ypes of apparatus are efficient, without reference to altitude
ier levels more work can be done by wearing apparatus than
g to the liberal oxygen supply.

n apparatus. Besides first-aid methods (Art 20), there is special
eviving rescued persons, by mechanical movement to assist res-
iration, and flush bad air out of the lungs by means of oxygen.

esuscitator, by the makers of the Westfalia apparatus, draws the air
e lungs by suction caused by an ejector, in conjunction with the com-
ternately forces in by light pressure, pure O, or a mixture of O and air.
uction to forcing movement is made by a hand lever. PULMOTOR, made
o, is designed to accomplish this movement automatically by pressure
In both of these, the movement may be made by hand, and in both
n an iron tank or bottle (like those used in rescue apparatus). When
s automatically it furnishes a mixture of O and air, the proportions of
ried to about 35% O. A committee on resuscitation from mine gases,
reau of Mines (53), reported adversely on automatic apparatus; on the
is improperly used, there is danger that the suction will cause collapse
e air passages in upper part of lungs; and that, in forcing O into the
y be caused by the press rising too high. The Committee favored
of artificial respiration, supplemented, in case of gas poisoning, by
nction with the "prone" manual method (Art 20).

ion apparatus in rescue operations. Portable telephones are
or communication between the temporary underground base
and between the base and the party using breathing apparatus.
employ ordinary telephones, though preferably of the iron-
To keep the advancing rescue crew in touch with the base is
When helmets are used with breathing apparatus, a satisfactory
nment to the helmet can be employed; but, since mouth-breath-
ferred, a special phone is necessary, requiring no lip movement.

lectric Co make a phone to be strapped to the head, so that it
of a man using the breathing apparatus. He "talks" while th

Flooding the mine with water is usu
since it is generally slow and costly, first to
damage. In collieries damage may be con
to squeezes where the bottom is soft shal
TOUR MAP of the mine is of the utmost im
flooding a fire area.

To limit the fire area dams may have to be c
on the low side on steep dips. Preparatory to
entrapped air or gases above the fire level by dri
not reach the high points of the fire. In the W
extinguish the fire by sealing, water was run inte
working level, but when pumped out, hot coals w
portion of the mine workings, not reached by th

16. Inundations and C

Mine inundations in foreign countri
worked under the sea, have led to serious
colliery, Japan, water from the sea broke
extending up 155 ft to the sea bottom, wh
clays. 237 men were drowned.

A similar disaster, though fewer lives were lost
sea at Whitehaven, England. Other colliery
place elsewhere, but inundations leading to large l
American mines. Except in the Lake Superior
bodies of water. In coal mining, of which the re
mining, there were, to 1915, 7 inundations caus
being 149 men. At Braidwood, Ill, 1883, 69 me
tending to the surface from shallow longwall w
River had overflowed its banks. At Nanticoke
inrush of quicksand and water. Four of the 7 in
from old workings, the last at Evans City, Pa, to
have been many lesser accidents from tapping o
from such catastrophes. In metal and miscell
light; 12 deaths in 1911, 5 in 1912, and none i
lakes there have been many serious accidents fro

Prevention of inundations. Surface i
tion of shafts and grading. To prevent fu
ings, plans and cross sections of excavations
made. In approaching old workings long
(preferably by diamond drill or equivalent)

Barrier pillars left around old workin
carry the load due to the overlying strata
factor in estimating the strength of barri
for, if the pillar begins to crush, water will
the hydraulic head cause its rapid failure i
barrier and other pillars, see Sec 10.)

Tunnels, even under stream or lake beds, if wi
locks if there is danger of inrushes; and if at g
be kept in advance of the heading. For metho
ground, see Sec 6 and 8.

Collapse or crush of mines occurs occ
by sufficient warning to withdraw the men.

The Atlantic Copper mine, Mich, collapsed i
vein. "Squeezes" in coal mines, causing the "
have usually been gradual enough to permit the

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e, Oklahoma, collapsed so completely and so suddenly that 13 men re killed. Many had been hoisted out of the slope just previously, lower, the mancar was blocked by the collapse.
e cavings have occurred in the Joplin, Mo, district, occasionally caus- gs and squeezes are both caused by insufficiency of roof support, due e too completely in the advance or first mining, and leaving too few . Timbering under these conditions, at any considerable depth below ate to support the superincumbent weight. In coal mining, squeezes o occur when the clay underlying the seam is softer than the pillar en forced downward, with side flow and heaving of the clay.

cavings and squeezes in coal mining: (*a*) in room and out in advance or first mining from 20% to not over 60%, de- n of the seam below surface; (*b*) pack the excavations with l them by sand flushing; (*c*) work the seam by the longwall ing (Sec 10). Similarly in metal mining, the size of pillars must ated with respect to the strength of rock or ore composing them, f hanging-wall to be supported. In deep work, so far as prac- on to leaving strong pillars, empty stopes should be filled with with rock blasted from the hanging-wall or mined in the foot- f sand from tailing piles is increasingly employed for filling or excavations back of the working face.

CUE AND RECOVERY WORK

Mine Rescue Apparatus and Rescue Stations

paratus (48, 49, 50) may be classified as: oxygen or air-breathing resuscitation apparatus, for reviving men overcome by gas or smoke; liances, as telephones, signals and guide ropes; lighting apparatus, hts, safety lamps; gas indicators, small animals for testing, safety apparatus; first-aid-to-the-injured appliances, as stretchers, medical nd hospitals; bratticing materials for restoring ventilation; fire- as smoke helmets, portable extinguishers, chemical engines, hose, pneumatic stoppings.

thing apparatus for rescue operations has long been used. ted in 1799 that at least one outfit be kept in readiness, and a in its use, in each mining district. Schwann (1853) designed tus, in which the contained air is resupplied with fresh oxygen, ted by a chemical substance. Other inventions followed in es.

olliery, England (1882), 5 men were rescued from bad air by means of us. The first notable rescue work in Germany was at the Könighere 14 unconscious men were saved by means of the Draeger oxygen U S (1908), after the Marianna mine explosion, Pa, the only living party equipped with Draeger outfits. In 1909, at the Cherry mine th breathing apparatus through a sealed shaft led to reopening the of 21 men 6 days later.

imes been lost due to defects in apparatus, or insufficient training. emann (Germany) lists 67 men rescued in 22 explosions or fires in nd 28 men lost in using apparatus in 23 separate accidents. Though show breathing apparatus to be far from perfect, there have been its successful use in preventing large losses of life.

2), if breathed undiluted, causes instant unconsciousness, quickly Before the introduction of breathing apparatus many men lost their rescue their fellows. Such fatalities are now comparatively ra is due more to failure to observe instructions, than to defects in

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of apparatus in irrespirable gases; a shower or bath an assembly room for lectures and first-aid training.

Joint rescue station is best where the min for any one to be reached in 30 min by auto t

Advantage of a central station, aside from the h permits employment of a foreman living at or near call, who shall be responsible for the upkeep of the lead the rescue crews. He should therefore be a hig apparatus, the station should contain copies of the mi ence to the possibility of explosions or fires, and sk rents, hose connections, etc. (Art 14, 15).

Auxiliary equipment for joint rescue stations m RESCUE TRUCK, equipped with 5 breathing apparatu auto motor, 2 folding stretchers, 2 large oxygen tank electric lights on a charging rack; also, oxygen rest ances, all placed on the truck in readiness for inst farthest mines are considerable, an auto bus for men For shaft mines, an emergency or collapsible cage, li cars, with reel of rope long enough for the deepest sh lene engine, may be very useful on occasions.

Refuge chambers in collieries have not been of the U S. The numerous instances of men explosions, or when escape has been prevented that refuge chambers of simple form should b off of a finished room, or unused entry in each pings in the break-throughs, and providing a do of the working. These need not be strong doo

If there are any prospect holes, these may be ca under them, thus securing ventilation and comm structed to go to such chambers in emergencies; drinking water, first-aid, and other supplies, may b might also serve as assistant foreman's office. Th signs, and their entrances whitewashed, to be read the rescue parties would search the refuge chambers would be mutually beneficial.

18. Rescue and Recovery Wo

Nearly all collieries run through a lifetime w dangerous conditions are sometimes present. expected; their possibility should be realized must adopt all means of prevention (Art 14). (Art 17) (whether individual or joint) should case of explosion in one of his mines.

First steps after an explosion. Action should not be reversed, if there is possibility o If fan is seriously damaged: (*a*) repair it if small self-contained fan, if it is available and (*c*) lower a steam jet into the upcast, or, if it wooden chimney say 18 ft high, close to entra in its base; (*d*) in extreme emergency, hang a or at mouth of slope or drift erect a wooden (Sec 14, Art 11). Unless a mine is unusually ga so diluted as not to be inflammable. It can be t with a safety lamp, who, if necessary, admits fr

k organization. State laws are not definite as to who shall be in of disaster. Generally the state mine inspector takes charge, as a iency, not legal right, though he has authority to stop underground fit. In other cases, operators' representative has assumed control.

oth sides have disclaimed responsibility, with unfortunate confusion. lines has no legal authority, and its rescue car and station men can be nteers only. In any event, SOME ONE PERSON MUST HAVE COMPLETE e work is to be done systematically. An efficient member of the mine be best man. If question is raised as to his legal authority, he should state inspector to settle the matter. Notices, signed by the state and , should then be posted, naming the person in authority, who may be MANAGER OF RESCUE WORK. This manager at once appoints GUARDS, nce into the mine of any except authorized persons, to keep back crowds gs and from the yard, and to guide and keep volunteers from interfering fore they have been assigned places. Guards are marked in some way, and pinned around the arm. Ropes are immediately strung about the nd working yard. The following CHIEF ASSISTANTS are also appointed, sible day and night for his part of the work.

rface guards.

gage volunteers and assign their duties.

scue work, with 7 sub-assistants in charge: (*a*) of breathing apparatus (*b*) of rescue men without apparatus; (*c*) of brattic ing crews. (Usu- work under general direction of state inspectors); (*d*) of keeping safety nd giving them out; (*e*) of checking men in and out of mine. Each ould have a numbered metal tag, to be given up when he comes out, . If metal tags are not available, use pasteboard or paper tag put into); (*f*) of stretcher gangs, for removing the injured and dead; (*g*) of of every dead body, with description of how it lay (generally best done ne surveyors).

charge of temporary hospital at mine mouth, transport of patients to d main hospital itself, in absence of regular organization. If there are s and surgeons, they are detached to underground bases.

general charge of morgue; bodies are taken in charge by coroner.

f supplies for rescue apparatus; brattice materials, timber and lumber.

y manager, to keep mine village supplied with food, and rescue crews food and coffee at all hours of day and night.

f temporary sleeping and living accommodations for the volunteers, such nd bedding near mine mouth and necessary sanitary arrangements, for bathing and drinking. The hard-worked and strained rescue and st have comfortable resting places, at least after the first day. Com- mines after large disasters often requires 1 to 2 weeks.

n man, charged with giving out non-sensational, but complete, satis- to the press, as well as to relatives and friends of entombed miners, circulation of sensational stories. He should be familiar with mining, rteous.

n great disasters, large quantities of food supplies and brattice erally be ordered by telegraph or telephone. An extensive and may throw down hundreds of stoppings, each requiring 4 yd of f a width a little more than aver height of seam up to 6 or 8 ft. requires at least 3 props and 2 boards.

bove) must immediately estimate what is on hand at mine, and then boring mines or supply houses what is needed. UNDERGROUND TELE- e also necessary for communicating with underground bases. Iron-clad est, but ordinary wall receivers will do temporarily (Art 17, Communi- . Say 100 to 200 suits of miner's overalls should be ordered; also boots, if mine is wet or flooded.

electric lamps. Even a safety-lamp mine rarely has enough issued to those entombed) to supply the rescue parties. Th[illegible]

Blasting down firewalls is sometimes good practic
must be done with great care; only quick-acting perm
tamped with fine rock dust or dry cement. This will
prevent ignition of firedamp or coal dust by permissible
ably not offer sufficient protection in case of dynamite.
ber suspended by wires or ropes from temporary fram
brick walls. In shafts provision is necessary for lower
through the airlock. If regular cages can not be used,
gency cage (Art 17) is required. After building an air
wearing apparatus begins investigation. (For organiz

In important cases a relief crew in apparatu
sionally water has collected on the shaft bottom
tion. In such case it may be best to fill the sh
point above the roof level of the entries, to pe
and ventilating to bottom. Then put down a p
with spiling, pump out the water. Finally, co
and explore the mine with aid of oxygen appara
extend as far as it is safe to travel, but not ex
trip) from base, so that if all goes well there wo
gen and soda supply available. This provides lit
are met, and 1 hr round trip from base is a safer
are clear and not hot, a distance of 1 mile from ba
adverse conditions, as extensive roof falls or acc
sult from an explosion, it is unsafe to advance
If seat of fire is beyond these limits, the party
the air, and study where new stoppings can be e

After conditions within the sealed-off area have bee
of the apparatus crews, the problem is: (*a*) whether
and ventilation restored; (*b*) if fire is too distant or ext
ing apparatus, to make new stoppings nearer to fire a
be recovered for operation, and to carry ventilation u
serve as a base for apparatus crews to make further in

Advance fire stoppings, put in by apparatus cr
and boards nailed to props. These may somet
generally sufficient to prevent serious escape of e
stantial stoppings are being erected in their rear.
are suitable for temporary use, when the passage

Restoring ventilation in a fire area where
fire or hot buried coals requires great care, and
liery. A series of mine-air samples should be
mine that the atmosphere on being diluted will n
to be explosive at one stage of dilution it is safer
mine as soon as the stoppings have been opene
tions should be opened automatically, or by bl
surface. Men should be kept out until, by ma
return air, the danger is judged to be over. Wh
the fire area is made by a party carrying a can
reviving, water lines are brought in quickly, an
rock, under which there may be coals hot enoug
air, should be drenched before removal.

20. First Aid to the Injured.

First aid to the injured, or simply "first aid," is t
to an injured person, to treat his injury with the me
becoming permanent, may save his life, tend to preve

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en pain. It is especially important in mining, since a severely injured point of a large mine may not be immediately found, and by the time be secured, hauled to the shaft and hoisted, an hour or more may s another hour before a surgeon can be brought to the mine receiving 's home. First aid was first taught to miners in the Penn anthracite ard extended to many mining districts by the Bureau of Mines and Red has been a marked success; surgeons have testified to the fine condition rought to hospitals; much suffering has been prevented, and many lives ght miners the importance of hygiene in their daily lives, disciplined sity of team work, and by inter-mine, state, and national first-aid con- centive and opportunity for young miners to advance themselves.

ections for injuries to which miners are subject are given in circular 8 (52, 53).

anization should be formed at every mine, to train men in and for social betterment and improved living conditions in Officers of the volunteer organization should include a medical ractice, membership should be divided into squads or teams of n, 1 patient, and 4 stretcher bearers.

eeded for practice. Each squad should have:

ndages	2 tourniquets
safety pins	2 rolls cotton, plain or absorbent
ze (plain or picric)	2 blankets (U S army preferable)
ts	1 stretcher (U S army regulation)
r yucca splints 3.5 in in long	6 wooden splints for legs and back fractures
es, assorted sizes	1 or 2 sets of first-aid charts

eams (52). So far as possible, men working in same part of me shift, should be grouped together. Each team should have cabinet at some convenient point in their part of the mine, and ould always carry at least one first-aid packet, containing a ge and sterile compress.

arm that someone has been hurt, a first-aid man should hasten to the but quickly and carefully examine the victim. If the injury is serious, the team should be collected, and the mine physician immediately , when the mine is equipped with phones) so he may know the nature r gathering the team, each member might have a policeman's whistle, being prearranged to signal the locality of an injured person. If there it should be stopped by pressure on the right spot. Place patient in ition as possible, keep him warm, and cheer him up.

ntiseptic conditions. Bacteria or germs are minute vegetable cells, er, the ground, clothing, and on and in the body. The mouth and with germs. There are many varieties of germs, some harmless, some ful, besides those which produce specific diseases. For propagation, rtain food, temperature, and environment. Having found a suitable hich may be in the human body, they multiply rapidly and give off ces, termed TOXINS. The latter may simply be irritants, or may de- ler some conditions harmful bacteria may produce septicemia or blood- ria gain entrance to the body through wounds, or the digestive system; the former source of infection.

om entrance of harmful bacteria into a wound, causing inflammation; ces are produced, destroying tissue, forming pus and preventing heal- psis, prevent germs from entering a wound by keeping unclean things it. Destroy or prevent growth of germs that may have entered at

e treatment of wounds by DISINFECTION. Disinfectant, or germicide, , and their spores or eggs. Destruction of disease germs in clothing ed STERILIZATION.

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Hemorrhage is loss of blood: loss of one-third of death. Compression and position are methods of applied by fingers, compresses, tourniquets, or cons belt strap, suspenders, or other improvised mean most dangerous, blood gushes in a bright red stream the wound and the heart. In CAPILLARY HEMORR slowly. It is easily controlled by applying a clea wound. In VENOUS HEMORRHAGE the blood is dark stream. The compression should be made on the Usually the bleeding can be controlled by applyi directly over the place. Elevating the bleeding pa After stopping bleeding patient should be treated fo

Shock is a sudden depression of vital powers, a tion, inducing exhaustion. Symptoms are sub-no pulse; cold, pale, profusely-perspiring skin; irregu scious, but stupid and indifferent, lying with partly concealed hemorrhage, needing immediate attent head, wrap him in blankets, or whatever clothing is nary stimulant like black coffee, to be sipped hot; spirits of ammonia may be given every 20 or 30 useful; artificial respiration is necessary in some and spine should be made if practicable.

Contusion, or bruise. Skin is unbroken, or me ness, swelling, and numbness, followed by aching pa surface contusions. TREATMENT: elevate injured p ing and control swelling. When available, apply ic

Wounds are classified as: INCISED wounds, made by a blunt object, like a fall of rock; LACERATED, by ery. Symptoms: Pain, bleeding, and gaping or reti clothing from wound at once; arrest bleeding, as dressing. Never attempt to wash wounds; leave th there is great danger of infecting the wound by was

Fractures are the most important class of injur aid, because the victim is temporarily crippled, an there may be permanent crippling. First-aid man fracture: INCOMPLETE FRACTURE, bone is not ent the bone is broken; SIMPLE FRACTURE, broken bone which is uninjured; in COMPOUND FRACTURE, eithe or the force which caused the fracture has cut or t compound fracture is nearly always accompanied Symptoms of fracture: pain, swelling, discoloratio and grating together of bone ends. TREATMENT: limb as little as possible; draw it into natural posit injury is in doubt, await arrival of doctor; a suffere moved until the part is properly supported by spli above is about to fall, when victim must be moved

Dislocation is displacement of surfaces of a joi ends are separated without injury to surrounding t ligaments around joint are torn. In complicated nerves are injured. Symptoms: Pain, swelling, dis of limb is changed and length altered. TREATMEN and hold it in place. Properly to reduce the dislo First-aid men should never try to reduce dislocation

Sprain is twisting or wrenching of a joint, produc times of surrounding soft parts. It is followed by and discoloration. Sprains are important injuries immediately. TREATMENT: Let injured person re in place by splints, or wrapping joint tightly with Give hot or cold applications by placing injured pa wrung out of hot water or ice water.

g or tearing of a muscle or tendon, and is usually caused by violent unexpected movement. It causes a sharp, excruciating pain. ured person rest; bandage injured part tightly, or apply adhesive etimes necessary.

. FIRST DEGREE burn is simply a scorching or reddening of outer ND DEGREE burn destroys entire thickness of skin; THIRD DEGREE d tissue beneath. Shock is caused by severe burns. TREATMENT: thing from burned surface, from which exclude air at once with covering in first-aid is picric-acid gauze (sterile gauze, saturated picric acid, or 0.5 teaspoonful to 1 pt water). Moisten the gauze apply it to the burned surface. Place a layer of absorbent cotton pply a bandage to hold it in place. Vaseline, sweet oil, olive oil, good dressings. If nothing better is at hand, dissolve bicarbonate ater. Gauze dipped in this and spread over the burn gives relief. e of equal parts of lime water and linseed oil) has been used, but

Electrical contact with a live wire causes dangerous shocks, burns, . Symptoms: sudden loss of consciousness; absence of, or very ak pulse, and burns at points of contact. If victim remains in not catch hold of him without first short-circuiting the current. witch, as time is too important, but lean a drill or bar of iron from line, preferably at a point between victim and power station. If nd on a dry board or piece of paper, and pull him off with a gloved love, use cap, coat, or other article of clothing, so as not to receive e victim from the wire with a rope, belt, or bandana, looped over r: As soon as patient has been freed, begin artificial respiration

hyxiation may be caused by an object blocking the windpipe, by lation of gas and air mixtures, which fill the lungs without sup- en, or by inhalation of poisonous gas, like CO (Art 5), causing possibly suspension of breathing by poisoning rather than absence NT: Quickly remove patient to good air; apply artificial respira- n in poisonous gas give him oxygen treatment.

tion. There are 2 principal methods, the Schaefer, or prone, n which the patient is placed on his back. The Schaefer is by physicians.

ions for its use, by Drs Cannon, Crile, Erlanger, Henderson, and ly feel with finger in patient's mouth and throat, to remove any false teeth, etc); then be- spiration. Do not stop to ng; every moment's delay ubject on his belly, with forward, with face to one mouth are free for breath- ssistant draw forward sub- o laying subject down that e pressed upon. Do not crowd about and shut off ddling the subject's thighs, rest the palms of your s (on muscles of small of early touching each other d over the lowest ribs (Fig ight, swing forward slowly f your body is gradually e subject (Fig 17*b*). This or 3 sec, and must not be

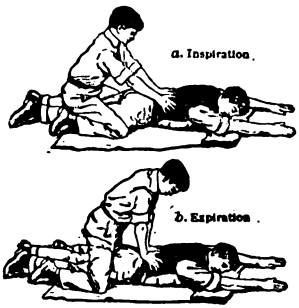

Fig 17. Schaefer Method of Artificial Respiration

rgans be injured. Lower part of chest, and also the abdomen, are air is forced out of the lungs. Then swing backward, to remove

pressure, but leave your hands in place, returning to
the chest walls expand and the lungs are supplied w
ward again, and repeat the 2 movements deliberately
plete respiration in 4 or 5 sec. Having no watch or
deep breathing, swinging forward with each expiratio

While this is being done, an assistant should loo
tient's neck, chest, or waist. Continue treatment,
interruption, until natural breathing is restored,
natural breathing begins see that it continues; if i
During the operation keep subject warm by proper c
bottles or rubber bags of warm (not hot) water. Ke
an assistant. Do not give any liquids whatever by

It is considered unnecessary to give oxygen in
patient had been "gassed." Mechanically-forced
of possible injury to lung cells, and when oxygen is a
lation, keeping time with normal respiration. A
unison with the giving of manual treatment, as by
fastened over the nose and mouth, and the apparat

Bandages are to keep dressings in place, to re
bleeding by pressure, and as slings. Kinds of ba
bandage of the U S army. TRIANGULAR BANDAGE
easy to make and to apply. It should be of unbleach
is too soft and difficult to fold properly; bed sheetin
should be 34 to 40 in sq, and by cutting diagonally
and knots for fastening. ROLLER BANDAGE is of n
wound on itself to form a tight roll. It requires co

Tourniquet, used to stop profuse bleeding from
around the limb, and a pad under the strap, to press o
nets contain U S army regulation tourniquet, consist
and catch. An improvised tourniquet may be mad
bandage, pair of suspenders, or belt strap. Tie arou
vein, pass a stick through the loop, and twist until b

Transportation of the injured. In a mi
an empty car near the point where the victim
be carried long distances. If done in a clum
of first-aid may be undone.

Method varies with the situation, facilities, and
should be carried on a stretcher, sometimes impro
through the sleeves of 3 or 4 coats, the coats being b
aid methods of carrying a patient by 1, 2, 3, or 4 m

Ambulance car should be maintained by
anthracite mines. It saves much suffering if
sion supports for a stretcher.

Underground hospital. In large mines,
man may be delayed many minutes at botto
this account, and for maintenance of medical
have installed underground hospitals where i
ceive supplemental treatment. They should
and kept clean; have an operating table for fi
ment, drinking and washing water, etc.

Surface hospital, or receiving room, ne
as most mines are distant from a town hospi
wait in heat, or cold, or a dusty building, for

A small well-lighted room is sufficient, painted w
with first-aid supplies, cot, table, water supply, an
ground hospitals may be inexpensive, but should b

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MINE SAFETY

. Safety Organization for Mining Companies

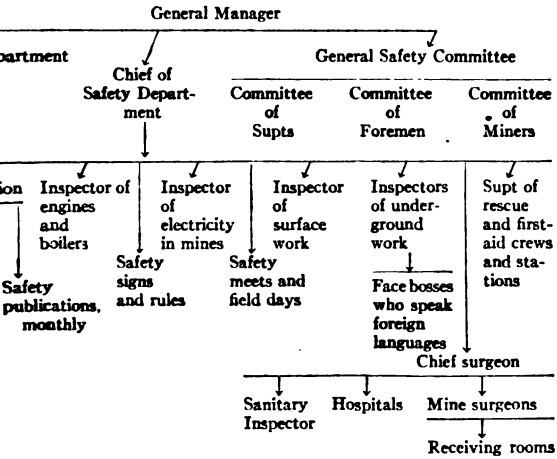
nization is essential to sound business policy, as well as on grounds.

f compensation acts by many states has greatly increased the cost of d injury claims; in some states this cost, in percentage of payroll, is 2 to ch prevailed up to 1910 (Art 24). But the compensation now goes more ed persons or needy heirs, and is not as formerly so largely consumed in l charges. Mining companies should care for widows, orphans, and in- from accidents, instead of depending, as in the past, on public charity. o is benefitted by safety improvements, since the cost of accidents must ded to selling price of product.

accidents in mines of the U S form from 60 to 75% of the total, nparison of our statistics with those of other countries, or with ds in this country. These can be lessened by inspection, educa- , and safety appliances, combined with proper organization.

f safety organization is one that is parallel to, but independent ng department.

fety Organization for Large Companies, Suggested by Author



manager, in the effort to economize in mining, transporting, and loading duct, may overlook questions of safety, and for economy or speed may unsafe machinery that should be scrapped. On the other hand, an over- anager, constantly fearing accident, becomes inefficient. But, if the ludes a competent inspector, responsible to a higher authority than the ger, and he reports a certain practice unsafe or recommends a change, the ved that he is not responsible for the cost incurred.

22. Mine Supervision and Inspection

. Mine accidents are often due to the isolation of the miners, a working place, rendering supervision difficult. While there is man with shift, timber, and other special bosses, there may be

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a large number of men scattered in remote w
man or shift-boss to supervise them. This i
tract system (as in collieries). As the forem
working every day, proper timbering and otl
be neglected. In general, there should be en
bosses to inspect every working at least twice
tice in the Yorkshire longwall mines. The be
20 to 25 miners.

Face-boss. Since over half the underground
the U S can not speak English, it is important to h
certain collieries, who should be a good miner and
visit the working places to direct foreigners as to t
been given certain foreigners by the foreman, to
often happen because the foreman on giving instruc
understood when they said "yes, yes," while know

Inspection by general company officers e
supts 2 or 3 times a month, and by mine sup
important. In addition, there must be a de
termed inspectors. In collieries, especially
FIRE-BOSSES should and must, under some s
abandoned places which are not walled off,
day shift, to test for gas or firedamp, fence
dangerous places in the roof for the setting of
entrance, and before the men are allowed to
concerning dangerous or gaseous places in th
and enter notes regarding the inspection in a b
mines may require several fire-bosses.

Private mine inspectors, so distinguished from
been employed by large companies to report on ol
of the company, in some cases to the General M
This system has in general resulted in great impr
tions; also, through information secured from the m
uals, and produces better relations between compa

Checking men in and out of mines. A
there should be a complete record of every on

The simplest mode of doing this is to number
place of working either in the pay-roll ledger or in a
entrance. Before entering the mine, each man mu
or mine entry being fenced in), and receive a metal
arranged with a safety-pin, for fastening to his clo
passed through a button hole. Before distribution
hang on nails on a frame, with a number over ea
changes, unused numbers are covered by hanging
corresponding checks being kept in a drawer or bo
and passes by the checking office, he leaves his ch
amination of the board at any time shows how ma
of the book in which each man's working place is
an important matter in case of a fire or explosion.

Inspection of men and equipment ent
be regularly made by a responsible man at t

He should examine lamps, and test by blowing
of an occasional man and all new employes or s
matches, smoking tobacco, or pipes. It is possib
inserted through safety lamp gauze, or by produc
lation of an electric lamp or battery; hence possess
dangerous practices.

23. Safety Meetings and Publications

tings, as an aid to increased care in prevention of mine acci-
ɔved to be of value, operating to educate miners and foremen
ɔring about more harmonious relations between employer and
e and national contests similarly serve to attract attention of
ɔovement and its importance to themselves.

ications, issued by mining companies from time to time (some-
by the safety department), do great good, because every em-
read English learns how a certain accident occurred, and how it
n prevented.

s and direction signs are useful in the prevention of accidents,
g uniform symbols, especially the red ball for danger, and the
the direction in which to go, the non-English speaking miners
what to avoid and where to go safely.

24. Mine-accident Liability Insurance

compensation acts. Until some mining states began the
hese Acts, the cost of death and injury cases from mine acci-
rally small as compared with payroll cost; widows, orphans, and
nerously treated by some companies, not by others (Sec 22, Art 8).

, coal-mine accident insurance ranged in different states from $1.50 to
d insurance was given by one cooperative company in a central state
s at 75¢ per $100 of payroll; in the same state rates now range from
100, and are quoted at $7 on coal mines in a western state that has had

ath and injury compensation costs have been similarly increased by the
ts of the respective states, but they have been effective too recently
at the final rates will have to be to meet the imposed conditions.

for meeting liability arising from mine accidents has been of
lue to the operators in the past, when flat rates were given in
standard insurance companies, regardless of whether safety pre-
observed or not (Sec 22, Art 8, 9).

e of the more important accident insurance companies have proposed
lits or discounts for rating organization and for different accident-pre-
and devices, and to employ mine inspectors who will not only in-
dvice. If the plan is found commercially successful, it should greatly
rators, possibly excepting those large enough to carry their own insur-
pendent corps of inspectors. It should lead to improvements for safety,
mple, in the whole mining industry.

State Authority, Regulations, and Inspection

Each state has police power over the mines within its borders
own regulations, which unfortunately differ widely, instead of
for the same class of mines and mining conditions.

egulations have been on the statute books in the leading coal-
for many years; one of the best codes is the recently revised
nnsylvania bituminous district, including electrical regulations.

ne-fire prevention regulations, as an outcome of the Cherry mine-fire
ois regulations are the most explicit and stringent, though some fea-
ent, are impracticable if taken literally.

ng regulations in many states are neither complete nor satis-
ly the most complete is the code now adopted in California. A

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committee of metal-mining men have also f
which has been published as a model by the

State colliery inspection. Practically e
in number, has a mine inspection departmen
Pennsylvania and West Virginia, the head i
of Mines; in Ohio, the Safety Commissioner
in general charge; in most states, the head
spector of Mines; in others, simply Mine Ins

In the important coal-producing states as Penn,
inspectors, each having a specified district, roughl
produced. There is no definite unit, but, in stat
inspector for every 2 million to 8 million tons, pr
mines and many wagon coal banks. Good servic
of every mine in a district 4 times a year; which m
that not over 30 large mines can be thoroughly i
unit in point of production should not be more tha
widely scattered small mines, not over half this. C
widely. Most important coal-mining states requir
an examination, the inspectors being chosen by th
the anthracite district, inspectors are elected fro
Oklahoma, the inspectors are elected, no examinati
men, and laborers, who know nothing about mini
outvote the mining men.

State metal-mine inspectors in the past have
Latterly, increased attention has been paid to this
several have passed more or less stringent regulati
regulations for metal mining are very lax as compa

County mine inspectors. In some metal-min
counties have appointed inspectors who may or ma
Generally, county inspectors have little authority
political jobs. Occasionally, as in certain Michig
ergetic county inspector has been able to introduce

26. Federal Safety Investigati

Origin and scope. In 1908, Congress ap
mine accidents, especially mine explosions,
Technologic Branch of the U S Geol Surv, whic
tural materials. In 1910, Congress established
to it the mine accidents and fuel investigation
passed, widening the scope, but defining the p

It established "in the Department of the Interio
Mineral Technology . . . to conduct inquiries . .
ration, treatment, and utilization of mineral substa
conditions and increased safety, efficiency, and eco

Restrictions. Bureau representatives hav
except as permission is voluntarily given by

The Bureau's reports on individual mine accide
conflict with state inspector's reports, and lead to
to the mine operator, with recommendations, so th
be avoided in future.

Rescue and first-aid equipment. To e
aid and the use of oxygen-breathing appa
safety or rescue cars with crews consisting o

te through different mining districts, giving instruction, and in case arest car or cars are rushed to the scene. Besides the R R cars, 5 d for giving training, at Pittsburg, Birmingham, Urbana, McAlister, eman and field mining engineers going immediately to any disaster ently to investigate the cause.

es and conditions for safe use in mines. The Bureau conducts, urg Experiment Station, experimental testing of explosives, electric mps, breathing apparatus, mechanical safety appliances, and at the ntal Mine (12 miles from Pittsburg) gas and coal-dust explosion

on of mines is conducted only in Alaska, on the segregated Indian where the Department of the Interior acts as agent for the lessees d in a few mines on government lands and in the National Parks in

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1. Introduction

Character of mining laws. These laws define the for mineral deposits, establish his methods of procedu sion while searching for mines, and give him assurance conditions have been fulfilled and valuable minerals for the operation of mines, and for the safety and health jects of Federal and State legislation, but are rather in sanitation rules for the daily guidance of mine work frequent change, and are not considered here. (See

Importance of an understanding of mining la considerations:

1. There is always a prescribed course of procedure ing and perpetuating such title as is granted by the la is in the exploratory stages and after ores have been strict compliance with these statutory provisions the mines after they have been proved to be of value.

2. The peculiar apex provisions of the U S minin of the Supreme Court are of vital importance to the n where the laws apply. The titles to many mines hav or entirely forfeited by inattention to the obvious and the statutes.

3. A mining engineer, called upon to place a valuati where the apex laws are in force, must be familiar wit of apex litigation hangs over many mining investme safely recommend the purchase of a quartz (vein) mi consideration.

2. Theories Relating to Acquisition of

Principles of construction of mining laws of th in two classes: 1. That in which the State or a private has the right to grant concessions or leases to indiv discretion, or under certain general restrictions. T system. 2. That in which any individual, under speci as to nationality or color, has the right to locate on disc limited areas of ground, and under certain conditions of the same. This is the "Claim" system (Alford, p

Concession system had its origin in the rights of to the mineral products of the ground, and it prevails t all the ancient civilizations. Its advantage is that th mining rights may select its concessionaires, and thu the hands of those who are able to operate them and t taxes, or royalties. There is incidental economy in t collection of dues, and the greater stability of mining is criticised for placing unduly large powers of property few men, and for taking away from the poor prospec wealth; it is also said to destroy competition in the ordinately large profits in the hands of a few; and that of mining ground unworked, thus reducing mineral p opportunity for labor. These objections may perhaps facts that the terms of concession may and usually do of development; that they reserve to the Governm financial transactions and capitalization of the conc

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f large corporations are generally conducted with greater efficiency
n both prospecting and mining stages, than those of smaller con-
viduals.

n originated in the early days of mining in the U S. Prospectors
ewly discovered gold districts of California, later to those of
to preserve public order, some arrangement became necessary
on the spot the area of ground that each man was allowed to
onditions under which he could hold and deal with it. Hence
f the discoverer to a "claim," a term now in general use in legal
re consolidation of claims is prohibited, the result is waste and
where claims are permitted to be held without systematic develop-
results to the country at large, nor to the claim holder; and it
eing "a system which, although it may appear at first sight to
idual enterprise, in reality leads to an enormous waste of both
rganized energy."

ental principles form the basis of all mining laws: 1. The right
rs to a secure and indefeasible title to their property so long as
in specified conditions, compliance with which is absolutely
n power; 2. The right of the state or other landlord to certain
or taxes on the profits of the mines, and to reasonably constant
effective work on the mines, so that capital, represented by prop-
shall not be unduly idle. These principles are essentially com-
s of all countries, though in individual cases modified to suit
gical, and physical conditions, as well as the history and theories
f the particular mining region (Alford, p 7).

ITED STATES MINING LAWS

3. Sources of the Laws

ral domain. Since nearly all of the domain in the states and
ct to the federal mining laws was originally acquired by treaty
m France and Mexico, wherein the civil law was the basis of
nd at the time of session both of these nations had well established
es of mining law, it is natural to expect to find the influence of
nish laws in the growth and development of our own system
Traces of the principles of the common law, under which, in
s were the landlord's property and went with the surface, are
e old ante-revolution British grants. Some customs and rules
West are traceable to Cornwall, and from there to Germany.
f common law prevail in the U S, and establish in the owner of
rima facie title to minerals beneath it, until rebutted by evidence
of the mines and minerals had, from some special cause, become
nat of the soil and surface. Such separate and distinct title
y arise through legislative acts of Congress or of any state, or
ry act of transfer by the individual who once owned both surface

nts. In almost all crown grants to the colonies, there was
sovereign a certain fixed proportion of the "royal metals" (gold
vered. Thus the charter of N C (1584), granted to Sir Walter
abeth, contained the following: "Reserving always to us, our
sors, for all services, duties and demands, the fifth part of all
and silver that may from time to time and at all times after

such discovery, subduing, and possessing, be ther
form of reservation occurs, with few exceptions
the three charters of Va (1606, 1609, 1611), the
fifteenth of all copper; Mass Bay (1629); grant
council of New England to Capt John Mason (
Lord Baltimore (1632); grant of the province o
(1639); R I and Providence plantations (1643
of Carolina, by Charles II to Earl of Clarendon,
(1663, 1665), reserved a royalty of one-fourth o
of the province of Me by Charles II to Duke of
of 40 beaver skins when demanded, and in retu
minerals. William Penn was required to pay 2
of all gold and silver ores.

The U S, in first dealing with its public lands
made similar reservations, suggested by preced
of public policy. The general policy of the U
executive acts, and proclamations prior to 184
from sale absolutely. These lands, so far as ther
copper and zinc lands, in the part of the U S terr
or Indian Territory, and comprised the area no
Ia, Mo, and Minn.

Federal leasing system. Within the territ
acquired from France in 1803, lead mining be
to Law's Mississippi colony, while that region
March 3, 1807, Congress passed a law providing
in the Indiana territory . . . shall be reserved
United States; and any grant which may her
land containing a lead mine which had been disc
of such tract from the United States shall be
and the President of the United States shall
lease any lead mine which has been or may be her
territory for a term not exceeding five years."

This legislation inaugurated the policy of th
These leases were given under supervision of
tracts at first 3 miles, afterward 1 mile square,
the mines with due diligence and return to the U
p 64). No leases were issued under the law unti
of lead was raised previous to 1826, from which
rapidly. For a few years the rents were paid
1834, because of the immense number of illega
Wis land office, smelters and miners refused to
government was unable to collect them. Afte
was, in 1847, finally decided that the only way
do away with all reserves of lead or any other m
source of embarrassment to the department. M
tion of the same act (afterward declared invalid),
to speculators in the Lake Superior copper regio
the scene of wild and baseless excitement. The
issue of permits and leases was suspended as illeg
ing sale of the mineral lands and a geological sur
dation of a more substantial prosperity (Hewitt

Mexican grants. The treaty of Guadalupe
Mexico, was signed Feb 2, exchanged May 30

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lded to the national domain an area of more than 500 000 sq miles, ates of Cal, Nev, Utah, Ariz (except Gadsden purchase of 1853), of the Rio Grande and north of Gadsden purchase, State of Colo Rocky Mts, and southwestern part of Wyo. Gadsden purchase public domain 45 532 sq miles, forming part of the present States Mex. By the treaty, all property theretofore belonging to Mexico its defined by the compact between the two nations passed to the hich government was imposed the obligation to protect titles ac- Mexican rule. A valid right before the cession was equally valid

n of Mexican grants. As to those in Cal, Congress provided ntment of a board of land commissioners, to whom all persons by virtue of any right or title derived from the Spanish or Mexican ere required to present their claims. Appeals could be taken from is board to federal courts, even the highest. By acts of July 22, 1861, and Feb 24, 1863, the surveyor-generals of N Mex, Col, and ely, were instructed to report to the Interior Department upon rivate land claims in their jurisdictions. Some grants so reported esented to Congress and confirmed; but the greater proportion ssage of some general law providing a uniform method of adjust- law was passed Mar 3, 1891. It created a court of private land ing of a chief justice and four associate justices, to which all per- lands within the limits of the territory acquired by the U S from alled upon to submit their claims. The object for which this court as been accomplished. It expired by operation of law June 30, ent, it is presumed that all rights and claims of every nature to out of Mexican grants have been finally adjudicated (Lindley,).

4. Source of the Public Domain

l government acquired no property rights within the present he 13 original states, nor in the states of Vt, Ky, Me and W Va, ved out of territory once forming part of the original states. The n of national domain which became subject to disposal of Con- essions of territory claimed by 7 of the original states. These ning with that by N Y (March 1, 1781), and ending with that of 802), brought within the jurisdiction of the federal government n of the present area of the U S now comprising Tenn, Ill, Ind, is, those portions of Ala and Miss lying north of the 31st parallel, rtion of Minn lying east of the Mississippi River. In 1803, the red by purchase from France, commonly called the "Louisiana ded over 1 000 000 sq miles to the national domain, embracing l Miss, the states of La, Ark, Mo, Iowa, Kas (except a portion in corner), Neb, all of Col east of the Rocky Mts and north of the , both Dakotas, the greater part of Mont, part of Wyo, and the The acquisition of land from Mexico has already been mentioned. chased from Russia in 1867. By treaty of cession between the , adopted by Congress July 7, 1898, all lands within the Hawaiian l by the treaty, which belonged to the republic as distinguished nds, passed to the U S, and became subject to the disposal of ame as other public lands. As yet, the public-land laws of the een extended to the new territory of Hawaii, and until further ess, the land laws of the republic existing at the time of the treaty ontinued in force. These laws do not provide for sale or other

disposition of lands classified as mineral. By
was ceded to the U S by Spain, and all crown l
of Congress, July 1, 1902, all such lands were c
of Porto Rico for the use and benefit of its peo
disposal under territorial laws of that island.
ownership of the Philippine Islands from Spain
was approved an act containing a code of la
lands valuable for minerals. This law was ame
692; Comp Stats, Supp 1911, 527; 10 Fed Stats
that federal statutes regarding the public doma
controlled by the U S. As to one portion, the
tion for lack of title, and as to still other portio
tion by act of Congress.

5. Theory of United States

Character of mining property. Mines in t
property, to be worked by the federal govern
a "public utility," but a private industry, to
all other economic industries (Lindley, p 120).
crown's ownership of the royal metals, where
theory that these metals were a prerogative of
England, France, Spain, and Mexico, was never
grant or conveyance by the U S carries all mi
or by implication in the law or instrument purp
loc cit). The government of the U S does not
nor the mining industry after it parts with the
patentee absolutely. No royalties are reserved,
vision attempted. Upon issuance of the deed
becomes private property, subject to the same
erty (Lindley, par 22). Although quite within
government has never engaged in mining operati
radium ores, proposed to give the government
whose owners do not work them diligently. The
in Alaska for naval use may likewise lead to gov

Minerals in private lands. No provision is
tion of mining claims upon or within privately
countries in Europe and South America, and the
under the theory that mining is a public utility.
lands only may be located and entered under
the courts are clear on this point. If at any time
is all located or reserved, and the discovery of m
tion declines, it may become a matter of nat
prospecting for minerals beneath the surface of
the only method would seem to be a congressi
after all a public utility, and the adoption of a co
izing passage of legislation permitting prospec
mining laws are said to consist of "an aggregat
quilt,' in the sense that they lack harmonious
The efforts of mining men are now directed to a
and codification which shall simplify them, and
their interpretation less subject to uncertainty.

Reservation of mining rights. Authority
given to the Department of the Interior under

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6 it was the settled policy of the U S government to reserve
ands sold or granted to individuals, states, or railways, and
The first sale of mineral lands by the government was that
and zinc mines in Mo at $2.50 per acre under act of Mar 3,
uly 11, 1846, sales of similar lands were authorized in Ill,
1. Under act of Mar 1, 1847, lands in the Lake Superior
ead, copper, or other valuable ores, were offered at public
and by act of Mar 3, 1847, the sale of lands in the Chippewa
uthorized on similar terms. Excepting these special sales of
s the uniform policy of the government to reserve mineral
rant in all cases. Thus, in grants for educational purposes
provements within the states, there was often a reservation
e mineral at the time of grant. If, however, minerals were
npletion of the transaction, the state's title thereto would
ne same is true in general of railroad grants. Non-mineral
e granted; but discovery of mineral character after title has
not invalidate title, because issuance of patent to the land
een examined by the land department and found to be non-
, and establishes vested rights which cannot be attacked.
es of the law are presented briefly, without detailed reference
ity, in the following pages. For fuller discussion see text-
ibliography.

6. First General Mining Act

ims to land which afterwards became part of the public
in Nev and Cal. The first miners made their own laws un-
n by territory, state, or nation. Their regulations were
l conditions, and embodied ideas and knowledge of the sub-
ners from Mexico, from Cornwall and other parts of Great
e continent of Europe. In accordance with these rules "the
mines is made to depend upon *discovery* and *development;*
made the source of title, and *development*, or *working*, the
tinuance of that title. These two principles constitute the
l laws and regulations respecting mining rights" (Halleck,
Fooz on Law of Mines, p vii).

1866, was the first general statute by which title could be
ic mineral lands within what are known as the mining states
s act has been largely superseded or repealed by subsequent
d to have established at least 3 important and beneficent

nineral lands of the public domain should be free and open
ccupation.
ch had been acquired in these lands under a system of local
rent acquiescence and sanction of the government, should
nfirmed.
t least certain classes of mineral deposits or lands contain-
ltimately obtained" (Lindley on Mines, par 54).
augurated and declared a policy of the greatest liberality
neral lands. Abandoning the idea of collecting royalties
e license to all who chose to become citizens, or were such
for minerals anywhere within the limits of the public
d theretofore been technically a trespass became thence-
ivilege, untrammeled by governmental surveillance or

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exaction of burdensome conditions." No pro
patenting of placer claims, although up to dat
chief producers of gold. This omission was co

7. Mining Act of

On May 10, 1872, Congress passed a law
development of the mining resources of the U
additions and amendments, is the law under
today, and in accordance with its provisions t
in the territory to which it applies have been l

Text of the statute. Title XXXII, Cha

§ 2 318. **Reservation of mineral lands from sa**
lands valuable for minerals shall be reserved fro
directed by law.

§ 2 319. **Valuable mineral deposits open to**
valuable mineral deposits in lands belonging to the
surveyed, are hereby declared to be free and open
lands in which they are found to occupation and
States and those who have declared their intentic
prescribed by law, and according to the local cust
mining districts, so far as the same are applicable
the United States.

§ 2 320. **Length of lode claim. Discovery ess**
End lines must be parallel. Mining claims upo
rock in place, bearing gold, silver, cinnabar, lead, t
heretofore located, shall be governed as to length
toms, regulations, and laws in force at the date
located after May 10, 1872, whether located by one
not exceed, 1 500 ft in length along the vein or loc
shall be made until the discovery of the vein or lode
No claim shall extend more than 300 ft on each sid
face, nor shall any claim be limited by any mining
side of the middle of the vein at the surface, except
10, 1872, render such limitation necessary. The en
to each other.

§ 2 321. **Proof of citizenship,** under this chapt
dividual, of his own affidavit thereof; in the case o
rated, of the affidavit of their authorized agent,
information and belief; and in the case of a corpor
United States, or of any State or Territory thereof,
charter or certificate of incorporation.

§ 2 322. **Extralateral and intralimital rights.**
heretofore made or which shall hereafter be made
situated on the public domain, their heirs and assi
May 10, 1872, so long as they comply with the law
or Territorial regulations not in conflict with the law
possessory title, shall have the exclusive right of po
face included within the lines of their locations, and
out their entire depth, the top or apex of which lie
downward vertically, although such veins, lodes or
pendicular in their course downward as to extend
surface locations. But their right of possession to
ledges shall be confined to such portions thereof
downward as above described, through the end lin
their own direction that such planes will intersect
ledges. And nothing in this section shall authoriz
lode which extends in its downward course beyond
upon the surface of a claim owned or possessed by

ghts. Length of tunnels. Where a tunnel is run for the develop-, or for the discovery of mines, the owners of such tunnel shall have of all veins or lodes within 3 000 ft from the face of such tunnel on previously known to exist, discovered in such tunnel, to the same from the surface; and locations on the line of such tunnel of veins g on the surface, made by other parties after the commencement of the same is being prosecuted with reasonable diligence, shall be prosecute the work on the tunnel for six months shall be considered the right to all undiscovered veins on the line of such tunnel.

trict rules. Marking boundaries. Records. Annual labor. ption of work. Forfeiture to co-owners. The miners of each nake regulations not in conflict with the laws of the United States, e State or Territory in which the district is situated, governing the ecording, amount of work necessary to hold possession of a mining following requirements: The location must be distinctly marked on s boundaries can be readily traced. All records of mining claims contain the name or names of the locators, the date of the location, n of the claim or claims located by reference to some natural object ent as will identify the claim. On each claim located after May 10, ent has been issued therefor, not less than $100 worth of labor shall rovements made during each year. On all claims located prior to orth of labor shall be performed or improvements made by Jan 1, hereafter, for each 100 ft in length along the vein, until a patent has but where such claims are held in common, such expenditure may be aim; and upon a failure to comply with these conditions the claim or failure occurred shall be open to relocation in the same manner as if e had ever been made, provided that the original locators, their heirs, sentatives, have not resumed work upon the claim after failure and

Upon the failure of any one of several co-owners to contribute his enditures required hereby, the co-owners who have performed the provements may, at the expiration of the year, give such delinquent tice in writing or notice by publication in the newspaper published at least once a week for 90 days, and if at the expiration of 90 days riting or by publication such delinquent shall fail or refuse to con- of the expenditure required by this section his interest in the claim erty of his co-owners who have made the required expenditures.

oceedings. A patent for any land claimed and located for valuable ined in the following manner: Any person, association, or corpora- ate a claim under this chapter, having claimed and located a piece ses, who has, or have, complied with the terms of this chapter, may l office an application for a patent, under oath, showing such com- plat and field notes of the claim or claims in common, made by or the United States Surveyor-General, showing accurately the bound- claims, which shall be distinctly marked by monuments on the t a copy of such plat, together with a notice of such application for uous place on the land embraced in such plat previous to the filing a patent, and shall file an affidavit of at least two persons that such posted, and shall file a copy of the notice in such land office, and shall to a patent to the land, in the manner following: The register of he filing of application, plat, field notes, notices and affidavits, shall such application has been made, for the period of 60 days, in a news- esignated as published nearest to such claim; and he shall also post e for the same period. The claimant at the time of filing this appli- e thereafter, within the 60 days of publication, shall file with the f the United States Surveyor-General that $500 worth of labor has provements made upon the claim by himself or grantors; that the uch further description by such reference to natural objects or per- s shall identify the claim, and furnish an accurate description to be tent. At the expiration of the 60 days of publication the claimant showing that the plat and notice have been posted in a conspicuous ing such period of publication. If no adverse claim shall have been

filed with the register and the receiver of the prope
60 days of publication, it shall be assumed that the a
the payment to the proper officer of $5 per acre, ar
thereafter no objection shall be heard, except it be s
comply with the terms of this chapter.

§ 2 326. **Adverse claims, filing of, and suit up**
during the period of publication, it shall be upon the
the same, and shall show the nature, boundaries, an
all proceedings, except the publication of notice an
thereof, shall be stayed until the controversy shall ha
of competent jurisdiction, or the adverse claim w
adverse claimant, within 30 days after filing his claim
of competent jurisdiction, to determine the question
cute the same with reasonable diligence to final judg
a waiver of his adverse claim. After such judgment
entitled to the possession of the claim, or any portion
notice, file a certified copy of the judgment-roll with t
with the certificate of the Surveyor-General that th
expended or improvements made thereon, and the
and shall pay to the receiver $5 per acre for his c
whereupon the whole proceedings and the judgment
to the Commissioner of the General Land Office, an
claim, or such portion thereof as the applicant sh
court, to rightly possess. If it appears from the dec
are entitled to separate and different portions of th
portion of the claim with the proper fees, and file t
Surveyor-General, whereupon the register shall certi
to the Commissioner of the General Land Office, a
shall issue to the several parties according to thei
contained shall be construed to prevent the alienatic
a mining claim to any person whatever.

§ 2 327. **Description of claim on surveyed an**
control locations. The description of vein or lod
designate the location of the claims with reference t
need not conform therewith; but where patents ha
upon unsurveyed lands, the surveyors-general, in e
just the same to the boundaries of said patented cla
or change the true location of such claims as the
ground. Where patents have issued for mineral la
gated and shall be deemed to be patented which are b
defined, and established upon the ground by the m
which the patent grant is based, and surveyors-ger
surveys, whether upon surveyed or unsurveyed la
The said monuments shall at all times constitute th
is patented, and in case of any conflict between th
claims and the descriptions of said claims in the pa
on the ground shall govern, and erroneous or inco
patent descriptions shall give way thereto.

§ 2 328. **Rights under patents issued before A**
for mining claims under former laws now pending i
in the General Land Office; but in such cases where a
patents may issue in pursuance of the provisions of t
ing claims upon veins or lodes heretofore issued shal
conferred by this chapter where no adverse rights ex

§ 2 329. **Placers and other forms of deposit**
patented. Claims usually called "placers," inclu
veins of quartz, or other rock in place, shall be sub
circumstances and conditions, and upon similar pro
lode claims; but where the lands have been previc
the entry in its exterior limits shall conform to the l

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es of claims. Group entries. Maximum extent of placers. 40 acres may be subdivided into 10-acre tracts; and two or more s of persons, having contiguous claims of any size, although such n 10 acres each, may make joint entry thereof; but no location of after July 9, 1870, shall exceed 160 acres for any person or associa- location shall conform to the United States surveys; and nothing on shall defeat or impair any bona fide preëmption or homestead l lands, or authorize the sale of the improvements of any bona fide er.

ations must conform to public surveys. Homesteads. Where surveyed lands, and conform to legal subdivisions, no further sur- quired, and all placer mining claims located after May 10, 1872, shall racticable with the United States system of public-land surveys, and visions of such surveys, and no such location shall include more individual claimant; but where placer claims cannot be conformed survey and plat shall be made as on unsurveyed lands; and where mineral lands in any legal subdivision a quantity of agricultural es remains, such fractional portion of agricultural land may be qualified by law, for homestead or preëmption purposes.

tained on adverse possession. Where such person or association, s, have held and worked their claims for a period equal to the time ute of limitations for mining claims of the State or Territory where ated, evidence of such possession and working of the claims for such nt to establish a right to a patent thereto under this chapter, in the e claim; but nothing in this chaper shall be deemed to impair any attached in any way whatever to any mining claim or property to the issuance of a patent.

or lodes within placers. Where the same person, association or ession of a placer claim, and also a vein or lode included within the pplication shall be made for a patent to the placer claim, with ncludes such vein or lode, and in such case a patent shall issue for the to the provisions of this chapter, including such vein or lode, upon r acre for such vein or lode claim and 25 ft of surface on each side der of the placer claim or any placer claim not embracing any vein paid for at the rate of $2.50 per acre, together with all costs of pro- vein or lode, such as is described in § 2 320, is known to exist within lacer claim, an application for a patent for such placer claim which pplication for the vein or lode claim shall be construed as a conclusive laimant of the placer claim has no right of possession of the vein or e the existence of a vein or lode in a placer claim is not known, a claim shall convey all valuable mineral and other deposits within of.

ineral surveyors. Expenses of survey and patent. Publica- signation of newspaper. Fees of officers. The Surveyors-Gen- ates may appoint in each land district containing mineral lands as eyors as shall apply for appointment to survey mining claims. The y of vein or lode claims, and the survey and subdivision of placer antities than 160 acres, together with the cost of the publication of by the applicants, and they shall be at liberty to obtain the same at ates, and they shall also be at liberty to employ any United States make the survey. The Commissioner of the General Land Office to establish the maximum charges for surveys and publication of apter; and, in case of excessive charges for publication, he may aper published in a land district where mines are situated for the notices in such district, and fix the rates to be charged by such d that the Commissioner may be fully informed on the subject, each h the register a sworn statement of all charges and fees paid by such ion and surveys, together with all fees and money paid the register d office, which statement shall be transmitted, with the other papers mmissioner of the General Land Office.

§ 2 335. **Affidavits, before what officers to be m**
made under this chapter may be verified before any offi
within the land district where the claims may be situ
may be taken before any such officer, and, when duly
same, shall have the same force and effect as if taken
the land office. In cases of contest as to the mineral o
testimony and proofs may be taken as herein provide
days to the opposing party; or if such party can not
least once a week for 30 days in a newspaper, to be de
office as published nearest to the location of such land;
that such notice has been given.

§ 2 336. **Cross-lodes and uniting veins.** Where t
each other, priority of title shall govern, and such pr
ore or mineral contained within the space of intersec
shall have the right of way through the space of inter
venient working of the mine. And where two or mo
location shall take the vein below the point of union,
tion.

§ 2 337. **Millsites, classes, patents for.** Where
the vein or lode is used or occupied by the proprietor
milling purposes, such non-adjacent surface ground m
application for a patent for such vein or lode, and the
subject to the same preliminary requirements as to sur
veins or lodes; but no location hereafter made of su
5 acres, and payment for the same must be made at th
for the superficies of the lode. The owner of a quartz
a mine in connection therewith, may also receive a pa
this section.

§ 2 338. **State legislatures may pass supplement**
in the absence of necessary legislation by Congress, t
Territory may provide rules for working mines, involv
necessary means to their complete development; a
expressed in the patent.

§ 2 339. **Prior rights to water protected. Right**
by priority of possession, rights to the use of water fo
ing, or other purposes, have vested and accrued, and th
edged by the local customs, laws, and the decisions o
of such vested rights shall be maintained and protecte
for the construction of ditches and canals for the purpo
and confirmed; but whenever any person, in the con
jures or damages the possession of any settler on the p
such injury or damage shall be liable to the party inju

§ 2 340. **Patents granted, subject to easements.**
tion or homesteads allowed, shall be subject to any v
rights to ditches and reservoirs used in connection wi
been acquired under or recognized by the preceding s

§ 2 341. **Homesteads upon mineral lands.** Wh
designated as mineral lands, which have been exclude
been homesteads made by citizens of the United Sta
their intention to become citizens, which homesteads
for agricultural purposes, and upon which there hav
silver, cinnabar, or copper discovered, and which a
settlers or owners of such homesteads shall have a rig
be entitled to purchase the same at the price of $1.25 pe
160 acres; or they may avail themselves of the provisi
ing to "Homesteads."

§ 2 342. **Secretary of Interior may set apart agr**
of the lands described in the preceding section, the Sec
and set apart such portions of the same as are clearly

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: to preëmption and sale as other public lands, and be subject to all ions applicable to the same.

al land districts, establishment of. The President is authorized al land districts, and to appoint the necessary officers under existing ay deem the same necessary for the public convenience in executing chapter.

tion of act, generally. Nothing contained in this chapter shall ir, in any way, rights or interests in mining property acquired under affect the provisions of the act entitled "An act granting to A. Sutro other privileges to aid in the construction of a draining and exploring ck lode, in the State of Nevada," approved July 25, 1866.

aws not applicable to certain states. The provisions of the pre- s chapter shall not apply to the mineral lands situated in the States inn, which are declared free and open to exploration and purchase, odivisions, in like manner as before May 10, 1872. And any bona nds within the States named since May 10, 1872, may be patented any of the foregoing provisions of this chapter. Such lands shall be in the same manner, at the same minimum price, and under the same as other public lands.

lands not granted to states or corporations. No act passed at 38th congress, granting lands to States or corporations to aid in the or for other purposes, or to extend the time of grants made prior to e so construed as to embrace mineral lands, which in all cases are o the United States, unless otherwise specially provided in the act ant.

y act of Congress approved May 5, 1876, deposits of coal, mineral within Mo and Kas were excluded from the operation o, 1872; and by a similar act approved Mar 3, 1883, Alabama ed from its operation.

8. Mining Laws for Alaska

ress, approved May 17, 1884, provided for a civil govern- nd extended the mining laws to that district (23 Stats at her legislation on this subject was approved June 6, 1900, as Large, pp 321, 326, 330):

ctive recorders shall, upon the payment of the fees for the same, orney-General, record separately, in large and well-bound separate *First.* Deeds, grants, transfers, contracts to sell or convey real of real estate, releases of mortgages, powers of attorney, leases which ed or proved, mortgages upon personal property; . . . . *Ninth.* ork done on mining claims; *Tenth.* Notices of mining location and s; *Eleventh.* Such other writings as are required or permitted by including the liens of mechanics, laborers and others; *Provided,* mining claims shall be filed for record within 90 days of the date of described in the notice, and all instruments shall be recorded in in which the property or subject matter affected by the instrument e the property or subject matter is not situated in any established instrument affecting the same shall be recorded in the office of the the court having supervision over the recording division in which ct matter is situated. . . . . *Provided,* Miners in any organ- ay make rules and regulations governing the recording of notices claims, water rights, flumes and ditches, millsites and affidavits of ith this act or the general laws of the United States; and nothing nstrued to prevent the miners in any regularly organized mining recording district established by the court, from electing their own as such until a recorder therefor is appointed by the court; *Pro-* s heretofore regularly made by the United States commissioner

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Dyea, Skagway, and the recorder at Douglas City, n
larly made with the United States commissioner at
all records heretofore made in good faith in any reg
hereby made public records, and the same shall be del
ing district including such mining district within six

Sec 26. The laws of the United States relating to r
rights incident thereto are hereby extended to the
subject only to such general limitations as may be n
artificial obstructions, all land and shoal water betw
shores, bays and inlets of Bering Sea, within the jur
be subject to exploration and mining for gold and oth
United States, or persons who have legally declared th
such reasonable rules and regulations as the miners
have heretofore made or may hereafter make governi
for exploration and mining purposes until otherwise
That the rules and regulations established by the n
the mining laws of the United States. And no exclu
Secretary of War authorizing any person or persons,
or mine under any of said waters below tide, and if su
it is hereby revoked and declared null and void; but
sons who have legally declared their intention to b
dredge and mine for gold or other precious metals in
to such general rules and regulations as the Secretary
vation of order and the protection of the interests of c
shall not, however, deprive miners on the beach of the
into or pump from the sea opposite their claims, except
obstruct navigation; and the reservation of a roadwa
of the act of May 14, 1898, entitled "An act extendir
for right of way for railroads in the district of Alaska
apply to mineral lands or town sites (see also Act of

An act to modify and amend the mining
territory of Alaska was approved Aug 1, 1912

By act of Congress, Aug 24, 1912, Alas
and provision was made for a legislative assemb
the privileges of a territory, and may pass such
the federal laws, in the same manner as states
may legislate on mining subjects. An elaborat
session of its legislature, Apr 30, 1913 (Sess, La

By an act of April 20, 1915, an abstract of
30, 1913, was repealed.

Placer claims. 1. The locator of a claim shall
ment a notice containing: (*a*) Name or number of cl
of discovery and of posting notice; (*d*) Length and

2. Boundaries must be clearly defined by placing
least 3 ft above ground by 3 inches diam, hewed on
at least 3 ft high by 3 ft diam. Each post or monun
number of claim and number of corner or angle, on
monument. If location is on ground covered by b
blazed along claim lines.

3. Within 90 days after posting notice the locat
the district a certificate, containing data (*a*), (*b*), (*c*)
of the boundaries referring to some natural object, p
mining claim. Failure to record such certificate sh
claim, which is then open to relocation (with the usua
owner).

4. No individual claim shall exceed 1 320 ft in
claim shall exceed 2 640 ft in greatest length. Any
in area or length (see 5), may be relocated as to suc
that end of the claim farthest from the initial monu

on placer claim shall hereafter be located in Alaska in excess of 40
y placer claim hereafter located, and until a patent has been issued
than $100 worth of labor shall be performed or improvements made
including the year of location, for each and every 20 acres or fraction
nt work for contiguous claims under one ownership may be done as
e par 6, under Lode claims).
all locate any placer claim as attorney for another, unless he is author-
power of attorney, duly acknowledged and recorded in the district
nor may such attorney locate claims for more than one individual in
district during the same calendar month. No person shall locate, or
for himself, in any one recording district, more than 2 placer claims
onth, one or both of which may be included in association claims.
im after the annual assessment work has been done, an affidavit must
istrict recorder, not later than 90 days after the end of the calendar
work was done, setting forth: (*a*) Name and number of claims, and
) Number of days' work and kind and value of improvements made;
nance of assessment work; (*d*) Place where work was done, with ref-
undaries; (*e*) At whose instance work was done; (*f*) Actual amount
improvements and by whom paid, when not done by owner.
claim by a U S mineral surveyor may be credited to assessment work,
lit does not exceed the assessment for one year. To obtain such
eld notes must be filed in the recorder's office, with sworn voucher of

. Location notice posted upon the claim shall contain: (*a*) Name of
Name of locator; (*c*) Number of linear feet in length along center
point of discovery, with width on each side of center line, the general
being marked as near as can be determined.
upon which the location is based must be defined by the removal of
aterial to expose the discovery claimed, and in such manner as to pre-
far as practicable, from obliteration; also its locus must be witnessed
post, bearing the location notice.
ll exceed 1 500 ft along center line, nor more than 300 ft on either side
thin 30 days after posting location notice, boundaries shall be marked
stake, or the blazing of a tree, at least 3 inches diam, at each corner
the side lines. Each mark shall bear the initial of the lode or claim,
point upon the boundaries which said monument represents. Center
nes shall be marked by blazing trees or cutting brush, or erecting line
ers or angles at inaccessible points may be marked by a witness monu-

ys of the posting of location notice, a certificate (similar to that for
ante) shall be recorded with the district recorder.
ms is to be made as prescribed in the California law (Art 16). Corner
should be connected by course and distance with the nearest corner
y, or with a U S monument, if the claim lies within 2 miles of such

ment work shall be $100 worth of labor or improvements, the period
ms to begin on Jan 1 following the date of location. If the claim in-
ne location, contiguous and under one ownership, $100 worth of work
ay be done in common, provided such work benefits the entire group.
be credited as for placer claims (see *ante*). Affidavit of assessment
placer claims; with added provisos that the wages charged shall be
g in the district, and that not more than 8 hours' work per man per
d. A declaration must be made that the time and expense of travel-
claims has not been included in computing cost of assessment work.
eceive the customary fee for filing a notice of location in his district,
ch claim included above a single claim.
tion, and location of tunnel rights and millsites, may be made as pre-
fornia law (Art 16). Assessment work for tunnel sites shall be $500
vey not to be included. Assessment work for millsites shall be $25 per
clude cost of survey; or $25 cash may be paid to the Treasurer of

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Location of coal lands in Alaska was n
tion by Congress in acts of June 6, 1900 (31 Sta
(33 Stats at Large, p 525); May 28, 1908 (35
laws were rendered inoperative by reason of w
dent Roosevelt and continued by President
applications were made for coal claims in acco
they have been almost without exception d
cancellation. The recently approved law go
in Alaska is given in full in Art 38.

9. Mining Laws Extended

All unoccupied public lands of the United
deposits of salt in any form, and chiefly valu
subject to location and purchase under the prov
mining claims; *Provided*, That the same pers
than one claim thereunder (31 Stats at Large,

10. Withdrawals of Pu

The President may, at any time in his discr
settlement, location, sale or entry, any of the
the District of Alaska, and reserve the same
classification of lands, or other public purpose
withdrawals, and such withdrawals or reserva
revoked by him or by an act of Congress.

Par 2. All lands withdrawn under the provisions
to exploration, discovery, occupation and purchase,
States so far as the same apply to minerals other
Provided, that the rights of any person who, at the
tofore or hereafter made, is a *bona fide* occupant or
and who, at such date, is in diligent prosecution o
gas, shall not be effected or impaired by such order,
shall continue in diligent prosecution of said work
shall not be construed as a recognition, abridgment
or claims initiated on any oil or gas-bearing lands af
prior to the passage of this act: and *provided further*
force and effect of any withdrawal made under the
are, on the date of such withdrawal, embraced in
entry theretofore made, or upon which any valid set
date being maintained and perfected pursuant to
shall not continue to apply to any particular tract
shall continue to comply with the law under which th
provided further, that hereafter no forest reserve sha
be made to one heretofore created within the limit
Mont, Col, or Wyo, except by act of Congress.

Par 3. The Secretary of the Interior shall report
the beginning of its next regular session after the da
1910; 36 Stats at Large, p 847).

By act of Congress approved Aug 24, 191
amended so as to provide that all lands withd
shall at all times be open to exploration, dis
under the mining laws "so far as the same
(37 Stats at Large, p 497).

11. Regulations of the Land Department

ature. The establishment of the Department of the Interior
y act of Congress, Mar 3, 1849, and supervision of mineral lands
o the general land office in that department (9 Stats at Large,
partment, consisting of the Secretary of the Interior, the Com-
General Land Office, and their subordinates, is a special tribunal,
n judicial powers to hear and determine claims to public lands,
y to execute its judgments by conveyances to proper claimants.
department as to questions of fact within its jurisdiction are
appeal to any court, nor may the courts anticipate its action
hemselves administration of public lands (Lindley, p 1644).
rtment, when not in conflict with the law, have the force and
d courts take judicial notice of them. It does not have the
grant land, but to facilitate and consummate grants authorized
ands are withdrawn by order of Congress or lawful executive
tion is suspended; and when lands have passed into private
al title has been transferred from the government, its author-
here the land officers have clearly mistaken the law of the case
he facts, courts of equity may give relief; but "where there is
of law and fact and the court cannnot so separate it as to see
mistake of law is, the decision of the tribunal to which the law
natter is conclusive" (Lindley, p 1 666).

of mining claims. There are two classes: Lode claims and

12. Lode Claims

Under the law of 1872, lode claims may have a maximum
, and a maximum width of 600 ft, in no case exceeding 300 ft
he vein at the surface. The owner of a valid location is also
ins of which the tops or apices lie within the exterior bounda-
provided they were not adversely claimed on May 10, 1872.

title. No lode claim shall be located until after discovery of
thin the limits of the claim. Sufficient development work is
d by the department to disclose existence of a vein, and if
al course and indications that it contains valuable minerals.

ce should give the course and distance as nearly as practicable,
haft on the claim to some permanent, well-known points or
one monuments, blazed trees, confluence of streams, prominent
in the immediate vicinity; also names of adjacent claims. A
t of stones should be erected at the discovery point or shaft,
ner of the ground claimed. Notice posted at the discovery
of the lode claim, names of the locators, number of feet claimed,
on from the discovery, and whether the claim thus staked is
on one side of the discovery point. Location notice must be
required by the state or territorial laws and local miners' rules,

of possessory title to a lode claim located since May 10, 1872,
the performance of at least $100 worth of work upon it each
This work is not required after entry for patent. Failure to
ired amount of work subjects the claim to re-location, unless
his heirs, assigns, or legal representatives, have resumed work
. Non-contributing partners may be "advertised out" by

publication of notice and summons in the ne
90 days, or by personal notice in writing. If,
ing, or 180 days after first newspaper publicat
ner shall have failed to contribute his proport
work, his interest in the claim passes by law
for the work.

Method of procuring patent. The owne
made of the claim, under the authority of th
territory containing the claim. One copy of t
the field notes shall be retained in the office
the plat given to the claimant for posting upo
field notes given to the claimant to be filed wi
transmitted by that official, with other paper
and one plat to be sent by the surveyor-gen
land district to be filed for future reference.
spicuous place on the claim, together with notic
date of posting, name of claimant, name of cla
district and county, and names of adjoining a
the survey plat. After posting notice, the cl
register and receiver a copy of plat and field
at least two credible witnesses that such plat a
upon the claim, giving date and place of such
posted to be attached to and form a part of s
also make affidavit that he has possessory rig
virtue of a compliance upon his part and tha
purchase) with the mining rules, regulations a
mining laws of Congress. The affidavit must
compliance, the origin of his possession, and
The vein or lode must be fully described, the
as to kind and character of mineral, extent
tracted and of what amount and value, and su
applicant's allegation that the claim contains
affidavit must be supported by a copy of eac
legal custodian of the record thereof, and als
claim, certified by the legal custodian of record
ized abstractor of titles. The certificates
affecting or purporting to affect the title to th
other than those set forth. The register will
publish a notice of the application for 60 days;
ing an affidavit of publication and another
certificate of the surveyor-general testifying to
worth of work upon the claim, the claimant
per acre, taking a receipt, and receiving in du
found to be regular.

12. Placer Clai

Nature. As in case of lode claims, a disco
is necessary before location of a placer claim
for placer claims, including all forms of min
quartz or other rock in place, are similar to tho
for vein or lode claims; but where a placer cl
and conforms to legal subdivisions, no furth
The price fixed by law for placer claims is $2.
an acre.

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tions should contain, in addition to the recitals necessary in or lode and placer applications, such detailed data as will sup- at the land applied for is placer ground containing valuable not in vein or lode formation, and that title is sought not to rses or to obtain valuable timber, but in good faith because of in. This statement, of course, must depend upon character of ral features of the ground, but the following details should be s possible: If the claim be for a deposit of placer gold, there he yield per pan, or per cu yd, as shown by prospecting and k, distance to bedrock, formation and extent of the deposit, s upon which the applicant bases his allegation that the claim deposits of placer gold. If a building-stone deposit, or deposit e claimed under the placer laws, applicant must describe fully and extent of deposit, stating the reasons why same is by him uable mineral claim. He will also be required to describe fully res of the claim; course of streams, amount of water carried, ll within the claim; also kind and amount of timber and other n, and adaptability to mining and other uses.

all placer ground, that fact must be stated in the application by accompanying proofs; if of mixed placer and lodes, it should th description of all known lodes situated within its boundaries. tion, as required by section 2 333, Revised Statutes (Art 7), as to each lode intended to be claimed. All other known lodes e of the applicant, excluded by law from all claim by him, of e, possessory or otherwise.

ions by the Department provide for methods of proof and pro- millsites, proof of citizenship, possessory right, adverse claims, surveyors, charges for publication of notices, conduct of hearings applications for entry, and various other matters of detail not quoted here (Land Dept Regulations, approved Mar 29, 1909, nphlet form, Nov 6, 1912).

LAWS OF DIFFERENT STATES

14. Authorization and Nature

e Revised Statutes (Art 7) authorizes supplemental State and tion, for regulating the working of mines, granting of easements, ainage, and other necessary means for their complete develop- slation may also provide limitations and conditions modifying g with the federal statutes on the subject. "The state may not n, nor impair the rights conveyed by a federal patent, but may, quently does, exercise its police power and regulate the manner ines, in the same manner that it might regulate any other in- , p 53).

15. Arizona

relating to Location, Development and Forfeiture of Mining mended in 1909 (Ariz Laws, 1909, ch 36, p 112), is summarized ext not given here):

of a lode may locate for himself, or for others.

"discovery" must be placed a monument of stones at least 3 ft high, or curely fixed, at least 4 ft above the ground, and attached thereto shall n notice signed by the locators, setting forth: 1. Name of claim located; s of locators; 3. Date of location; 4. Length and width of claim in

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feet, and distance in feet from point of discovery to e
of the claim; 6. Reference to some natural object or
§ 3. No rights attach until performance of each o
§ 4. Locator has 90 days within which to do the f
recorder's office a copy of location notice; 2. To sink
least 8 ft deep from lowest part of rim of shaft at the
make disclosure of mineral in place; 3. To monume
can be readily traced.
§ 5. Failure to do all things enumerated shall be
right and claim of the discoverer shall be forfeited.
§ 6. Six posts, at least 4 ft above ground, or stone m
one at each corner of the claim, and at the center of e
§ 7. Any open cut, adit, or tunnel, equal in amoun
wide and 6 ft long, disclosing a lode or mineral 8 ft b
of the required discovery shaft.
§ 8. Location notices may be amended and monu
spond, so long as there is no interference with the rig
§ 9. The amount of annual work required shall be
§ 10. The form of the affidavit of annual represent
§ 11. Such affidavit shall be *prima facie* evidence o
claims may be an extension or continuation of locati
§ 12. Placer claims shall be located as follows: E
containing name of claim, name of locator or locator
claimed, and a description of the claim with referen
nent monument that will identify the claim; and by
a post or monument of stones at each angle of the
it must be at least 4 in diam by 4 ft 6 in in length, se
by a mound of stone or earth.
§ 13. Where it is practically impossible, on acco
ground, to sink such posts, they may be placed in a pi
diam at the base. Witness posts or monuments may
§ 14. Copy of location notice containing all requ
of the county recorder within 60 days.
§ 15. Delinquent co-owners whose shares are sou
requiring affidavits of publication and service of not
notice and filed in the office of the county recorder
and within 180 days after first publication.
§ 16. The original of such notice and affidavits of
of delinquency, provided the writing or affidavit here
§ 17. If, within the 90 days provided, the delinqu
expenditures, a receipt therefor shall be given, and
with the county recorder.
§ 18. Co-owners who fail to sign such a receipt an
ment shall be liable to a penalty of $100; and in th
interested persons may make affidavit of payment
shall be *prima facie* evidence of such contribution.
§ 19. In all actions, judgments, grants or conveya
tion of a mining claim if it can be intelligently learn
the district, county and territory where same is situa
thereof is recorded.
§ 20. County recorders are authorized to procure
§ 21. Mines and claims properly located prior to
§ 22. No relocation of an abandoned mining clai
be held invalid because of failure to state in notice
part or in whole an abandoned mining claim.

16. California. Act of

Lode claims, how located. 1426. Any perso
who has declared his intention to become such, wh
or other rock in place, bearing gold, silver, cinnaba
deposit, may locate a claim upon such vein or lode, b

inafter described, and by posting a notice of such location, at the point notice must contain: *First*, Name of the lode or claim; *Second*, Name cators; *Third*, Number of linear feet claimed in length along course of n point of discovery, with width on each side of center of claim, and ein or lode, as near as may be; *Fourth*, Date of location; *Fifth*, Such by reference to some natural object, or permanent monument, as will ocated.

l extent of lode claim. 1 426a. Locator must define boundaries they may be readily traced, and in no case shall the claim extend more the course of vein or lode, nor more than 300 ft on either side thereof, er line of vein at surface.

ion of lode claim. 1 426b. Within 30 days after the posting of n upon a lode-mining claim, the locator shall record a true copy thereof ounty recorder of the county in which such claim is situated, for which ecorder shall receive a fee of $1.

cation of. 1 426c. Location of a placer claim shall be made in the By posting thereon, upon a tree, rock in place, stone, post or monucation containing the name of claim, name of locator or locators, date of feet or acreage claimed, such description of claim by reference to t or permanent monument as will identify the claim located, and by laries so that they may be readily traced; *provided*, that where the y has been extended over the land embraced in the location, the claim gal subdivisions, and no other reference than those of said survey shall e boundaries of a claim so located and described need not be staked The description by legal subdivisions shall be deemed the equivalent

ion of placer claim. 1 426d. Within 30 days after posting of notice er claim, the locator shall record a true copy thereof in the office of the he county in which such claim is situated, for which service the recorder { $1.

cation of. 1 426e. Locator of a tunnel right or location shall locate ice at the face or point of commencement of the tunnel, which must me of locator or locators; *Second*, Date of location; *Third*, Proposed f tunnel; *Fourth*, Description of tunnel, with reference to some natural t monument as shall identify the claim or tunnel right.

tunnel location. 1 426f. Boundary lines of the tunnel shall be es or monuments placed along the lines at an interval of not more than e or point of commencement of the tunnel to the terminus of 3 000 ft

el location. 1 426g. Within 30 days after posting notice of location location, the locator shall record a true copy thereof in the office of r of the county in which such claim is situated, for which service the ve a fee of $1.

e of location. 1 426h. If at any time the locator of any mining hereafter located, or his assigns, shall apprehend that his original locafective, erroneous, or that the requirements of the law had not been re filing; or in case the original notice was made prior to the passage shall be desirous of securing the benefit of this act, such locator, or his additional notice, subject to the provisions of this act; *provided*, that tion notice does not interfere with the existing rights of others at the filing such amended location notice, and no such notice, or the record ude the claimant, or claimants, from proving any such title as he or d under previous locations.

y of mining claim is prima facie evidence. 1 426i. Where a locahas the boundaries and corners of his claim established by a United ral survey, or a licensed surveyor of this state, and his claim connected the public or minor surveys of an established initial point, and incorord of the claim the field notes of such survey, and attaches to and files

with such location notice, a certificate of the su
survey was actually made by him, giving the date
surveyed and the location thereof; *third*, that the
atory statement is sufficient to identify; such st
the record, and such record is *prima facie* evidenc

Millsite, location of. 1 426*j*. Proprietor of a
a quartz-mill or reduction works, or any person qu
may locate not more than 5 acres of non-mineral
be made in the same manner as hereinbefore requ

Record of location of millsite. 1 426*k*. Loc
within 30 days from the date of his location, recor
the county recorder of the county in which such l
recorder shall receive a fee of $1.

Annual labor required. 1 426*l*. The amoun
during each year to hold possession of a mining cl
of the United States, to wit: $100 annually.

Record of proof of annual labor. 1 426*m*.
corporation shall have performed the labor and m
upon any mining claim, the person in whose behal
ments made, or someone in his behalf, shall withi
forming such labor or making such improvements
recorder, in books kept for that purpose, in the
situated, an affidavit setting forth the value of lab
the claim, and the name of the owner or claimant
was made or performed. Such affidavit, or a cop
recorder, shall be *prima facie* evidence of the perf
such improvements, or both.

Fee for recording proof of labor. 1 426*n*.
quired, the county recorder shall receive a fee of

Forfeiture to co-owners. Contribution by d
ever a co-owner or co-owners of a mining claim
co-owners the notice in writing or notice by pu
Revised Statutes of the United States, an affid
stating the time, place and manner of service, and
was made, shall be attached to a true copy of su
must be recorded in the office of the county reco
the county in which the claim is situated, within
for the recording of which, said recorder shall rece
law for recording deeds; or if such notice is give
shall be attached to a printed copy of such notice a
or principal clerk of such paper, stating the date
such notice therein, and where the newspaper w
name of such newspaper. Such affidavit and noti
180 days after the first publication thereof. Th
or a duly certified copy of the record thereof, sh
linquent mentioned in § 2 324 has failed or refus
expenditure required by that section, and of the
provided, the writing or affidavit hereinafter pro
linquent shall, within the 90 days required by
owner or co-owners, his proportion of such exper
the notice required by this section, whether incur
such co-owner or co-owners shall sign and deliver t
stating that the delinquent or delinquents by name
aforesaid, contributed his share for the year
therein the district, county and state wherein the
where the location notice is recorded, if said mi
this act; such writing shall be recorded in the offi
for which he shall receive the same fees as are n
If such co-owner or co-owners shall fail to sign an

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in 20 days after such contribution, the co-owner or co-owners so failing e liable to the penalty of $100, to be recovered by any person for the ent or delinquents in any court of competent jurisdiction. If such co- s fail to deliver such writing within the said 20 days, the delinquent, ted persons having personal knowledge of such contribution, may make rth in what manner, the amount of, to whom, and upon what mine, as made. Such affidavit or a record thereof, in the office of the county unty in which such mine is situated, shall be *prima facie* evidence of

received in evidence. 1 426*p*. The record of any location of a ite or tunnel right, in the office of the county recorder, as herein pro- ived in evidence, and have the same force and effect in the courts of iginal notice.

rd as evidence. 1 426*q*. Copies of the records of all instruments rded by the provisions of this act, duly certified by the recorder, in whose ls are, may be read in evidence, under the same circumstances and rules be hereafter provided by law, for using copies of instruments relating executed or acknowledged or proved and recorded.

n mining districts. 1 426*r*. The provisions of this act shall not in nstrued as affecting or abolishing any mining district or the rules and within the state of California.

orm annual labor — Relocation void. 1 426*s*. The failure or neg- of a mining claim to perform development work of the character, in thin the time required by the laws of the United States, shall disqualify relocating the ground embraced in the original location or mining claim f under the mining laws, within 3 years after the date of his original ttempted relocation thereof by any of the original locators shall render

l parts of acts in conflict with this act are hereby repealed. ll take effect and be in force on and after July 1, 1909.

mining law has been quoted in full as being one of the more s of state legislation on the subject of the location of mining public domain, within the particular states. Special provisions nd sale or lease of state mineral lands have also been passed by , Nev, N Y, Tex, Wis, and some other states.

consideration of the mining statutes of the Western States, made only of those particulars in which they differ from the

17. Colorado

claims. § 2. The width of lode claims hereafter located in Gilpin, der, and Summit counties, shall be 150 ft on each side of the center of ; in all other counties the width of the same shall be 300 ft on each side vein or crevice; and the owner or owners of any lode claim or claims and having a less width, desirous of securing the benefit of this act, ional certificate claiming such additional width as herein provided; after any county may, at any general election, determine upon a greater g 300 ft on each side of the center of the vein or lode, by a majority of t at said election, and any county by such vote at such election may less width than above specified (Mills' Annot Stats, § 3 149; Amd , 413).

t. § 6. Before filing such location certificate the discoverer shall : *First*, sinking a discovery shaft upon the lode, to the depth of at least est part of the rim of such shaft at the surface, or deeper, if necessary ned crevice; etc (Mills' Annot Stats, § 3 152).

t, time within which it must be sunk. § 9. The discoverer shall the time of uncovering or disclosing a lode to sink a discovery shaft ot Stats, § 3 155).

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square by 4 ft 6 in long, set 1 ft in the ground, depth, in which case the post should be set upon a mound of earth or stone of at least 4 ft diam requisite size, surrounded by such mound, shall mound; 3, a stone at least 6 in square by 18 in ground, with a mound of earth or stone alongside boulder at least 3 ft above the natural surface of other monuments, or monuments of lesser dime used, it shall be a question for the jury, or for the a jury, as to whether the location has been marked can be readily traced. Whatever monument is of the claim, and the designation of the corner, ei

Third, within 60 days after posting such notic lode or deposit at or near the point of discovery Such shaft shall be sunk to the depth of at least of the rim of such shaft at the surface, or deeper if located, and the cubical contents of such shaft sh that any cut or tunnel which discloses the vein, lo of at least 10 ft below the natural surface of the 150 cu ft of excavation, shall be deemed the equi that, where the vein, lode or deposit located is d deficiency in the depth of the discovery shaft, cut any horizontal extension of such working, or by a claim, equalling, in cubical contents, the cubical case at least 75 cu ft of excavation shall be mad 1907, § 2 283):

Millsites. § 3. Millsite claims may be locate other claims, except that no discovery or discove claim is appurtenant to a mining claim, the certi shall describe, by appropriate reference, the minin Code, 1907, § 2 285).

Validating locations heretofore made. § 4. under the laws of this state, heretofore in force, th to the requirements of such laws, shall neverthele parties accruing prior to the passage of this act, such locations conform to the requirements of th

Rights of relocator. § 7. The rights of a r mining claim, hereafter relocated, shall date fro thereon, and, while he is duly performing the acts his rights shall not be affected by any re-entry or r or claimant (*Loc cit*, § 2 287).

Relocation by owner. § 9. A locator or cl own claim for any purpose, except to avoid the pe by such relocation, may change the boundaries of both; but such relocation must comply, in all res as to an original location (*Loc cit*, § 2 289).

Rights of third persons not affected. § 1 mining claim by the locator or claimant thereof sh person existing at the time of such amendment or

Defective locations good against persons w prescribed by this law for the performance of a where the act is performed before the rights of defect in the posted notice or recorded certificat against one who has located the same ground, or without notice. Notice to an agent, who makes deemed notice to his principal, and notice to one notice to all (Rev Code, 1907, § 2 293).

(For reference to miscellaneous legislation

20. Nevada

t and equivalent. Marking boundaries. § 2. • • • • • locator of all within 20 days from date of posting notice of location define the claim by placing at each corner and at the center of each side line one described monuments, and shall within 90 days of date of posting said form the location work hereinbefore prescribed (Rev Laws, 1912, § 2 423).

record. Records made prior to passage of act. § 3. • • • • • the date of posting location notice upon the claim, locator shall record mining district recorder and the county recorder of the mining district h such claim is situated by location certificate which must contain: e or ve n; *Second*, Name of locator or locators; *Third*, Date of location on of the location of said claim, with reference to some natural object ument as will identify the claim; *Fourth*, Number of linear feet claimed course of the vein each way from the point of discovery, with the width center of the vein, and the general course of the lode or vein as near Dimensions and locations of the discovery shaft or its equivalent, sunk *xth*, Location and description of each corner, with the markings thereon. he location of a lode mining claim which shall not contain all the re- in this section shall be void. All records of lode or placer mining tunnel rights heretofore made by any recorder of any mining district or r are hereby declared to be valid • • • • • (Rev Laws, 1912, § 2 424).

Amount required. Value of a day's labor. § 9. • • • • • In rth of labor required to be performed upon any mining claim, to hold e laws of the United States, the value of a day's labor is hereby fixed dollars; • • • 8 hours of labor actually performed upon the mining ute a day's labor (Rev Laws, 1912, § 2 430).

tion claiming more than one location is void. § 12. (Same as , § 15.)

tion certificate. Record. § 14. Within 90 days after the posting cation of a placer claim, the locator shall perform not less than $20 on the claim for the development thereof, and shall have recorded by recorder and the county recorder of the district and county in which d a certificate which shall state: *First*, Name of claim, designating it *Second*, Name of locator or locators; *Third*, Date of location; *Fourth*, cres claimed; *Fifth*, description of claim, with regard to some natural nt monument; *Sixth*, kind and amount of work done; *Seventh*, place rk was done. But if such certificate do not state all the facts herein d it shall be void (Rev Laws, 1912, § 2 435).

r location, how located. Posting. § 19. The locator of a tunnel all locate his tunnel right or location by posting a notice of location at f commencement of the tunnel, which must contain: *First*, Name of ; *Second*, Date of location; *Third*, Proposed course or direction of ight and width thereof; *Fifth*, Position and character of the boundary Description of the tunnel by such reference to a natural object or nt as shall identify the claim or tunnel right (Rev Laws, 1912, § 2 440).

y lines, how established. § 20. Boundary lines of tunnel shall be es or monuments placed along such lines at an interval of not more ace or point of commencement of tunnel to the terminus of 3 000 ft nkes or monuments shall be of the same size and character as those placer claims in this act (Rev Laws, 1912, § 2 441).

covered in tunnel, how located. § 22. All blind lodes, or veins ously known to exist, discovered in a tunnel run for the development for the discovery of mines, and within 3 000 ft from the face of such ted upon the surface and held in like manner as other lode claims under is act (Rev Laws, 1912, § 2 444).

all title to lands containing mineral. How such lands are to be a public use. The several grants made by the United States to

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State of Nevada reserved the mineral lands. Sal
made subject to such reservations. Any citizen
declared his intention to become such, may ente
notwithstanding the State's selection, and explor
or other valuable mineral, and upon the discove
and mine the same in pursuance of the local rul
laws of the United States; provided, that after
the State has made valuable improvements thereo
or injured without full compensation. But such
the uses and purposes of mining in like manner a
and taken for public use. Mining for gold, sil
valuable mineral, is the paramount interest of th
public use (Rev Laws, § 2 456).

(For reference to miscellaneous legislatio

21. New Me

Proof of annual labor. § 6. (Must be file
required by law to be performed. Must give th
was performed. Otherwise, like the California la

Local rules, to be made by certain land o
or owners of lands within this Territory, the tit
patent from the United States government, may
clerk of the county in which such lands are situa
consistent with the laws of the United States, an
governing the location and acquisition of mining
tions when so filed shall be binding upon all parti
the county recorder shall be received and admitte
relating to such mining claims; such rules and r
mented from time to time by other rules and re
that such change shall not affect rights acquired (p

Deposits subject to placer location. § 1.
Mexico, chiefly valuable for the deposits of gypsu
alluvial deposits of gold, and all other material rec
as placer material, may be located as placer minin
1909, p 190).

Recording notice. Time within which disc
cate of such location notice shall be filed and rec
of the county wherein the land located is situate
made and such notice placed on the claim as afo
the locator or locators must have made a *bona fi*
claimed in said notice, or said location will be vo
person or persons; *Provided*, however, that in c
petroleum oil or natural gas, the locator or locators
location to the end of the calendar year succeedin
make a discovery of petroleum oil or natural gas;
been located for petroleum oil or natural gas, or b
have the right to the exclusive possession and oc
location for the purpose of prospecting for petrole
of time provided in this section for making disco
both (Laws, 1909, p 190). (As to validity of such

(For reference to miscellaneous legislatio

22. North D

Lode claims, width. § 2. The width of lod
center of the vein or crevice; provided, that any
termine upon a greater width, not exceeding 300
or lode, by a majority of the legal votes cast at suc

y determine upon a less width than specified; provided, that less than f the vein or lode shall be prohibited (Rev Code, 1905, § 1 801).

. **Posting notice. Manner of locating claim.** § 5. Before filing icate the discoverer shall: *First*, locate his claim by first sinking a eon sufficient to show a well defined vein or lode; *Second*, post at the n the surface a plain sign or notice, etc (Rev Code, 1905, § 1 804).

aries. § 6. Such surface boundaries shall be marked by 8 substantial ed on the side facing the claim, and plainly marked with the name of ner, end, or side of the claim that they respectively represent, and sunk ows: one at the corner and one at the center of each side line and one ode. • • • (Rev Code, 1905, § 1 805).

hich discovery shaft must be completed. § 8. Discoverer shall he time of uncovering or disclosing a lode in which to sink a discovery Code, 1905, § 1 807).

andoned lode claims. § 12. Relocation of abandoned lode claims king a new discovery shaft and fixing new boundaries in same manner ation of a new claim, or relocator may sink the original shaft, cut or epth to comply with sections 1 804 and 1 808, and erect new or adopt renewing the posts, if removed or destroyed. In either case a new be erected. In any case, whether the whole or part of an abandoned location certificate must state that the whole or any part of the new s an abandoned property (Rev Code, 1905, § 1 813).

ate must contain description of but one location. § 14. (Same on, § 15.)

and regulations, how far binding. § 15. In actions respecting f must be admitted of the customs, usages, or regulations established bar or diggings embracing such claim; and such customs, usages, or ot in conflict with the laws of this State or the United States, must of the action (Rev Code, 1905, § 7 536).

to miscellaneous legislation on mining, see Lindley, p 2 523.)

23. Oregon

• • **marking boundaries.** § 1. • • • • • Such boundaries shall o days after posting of such notice by 6 substantial posts, projecting ove the surface of the ground, and not less than 4 in square or in diam- ial mounds of stone or earth and stone, at least 2 ft high, to wit, one of rock at each corner, and at the center ends of such claims (Laws

e of location, etc. § 2. Such locator shall, within 60 days from and the location notices by him upon the lode or claim, file for record with eyances, if there be one, etc. • • No location notice shall be entitled led, until the work required by section 3 977 has been done and the ereof is attached to the notice to be recorded (Laws, 1901, p 140).

ork, affidavit of performance of. § 3. Before the expiration of ate of the posting of the notice of discovery upon his claim as afore- ording the notice of location as required by section 3 976, the locator ery shaft upon the claim located to a depth of at least 10 ft from the im of such shaft at the surface, or deeper if necessary to show by such of mineral deposit in place. A cut or cross-cut or tunnel which cuts of 10 ft, or an open cut at least 6 ft deep, 4 ft wide, and 10 ft in length the point where the same may be in any manner discovered, is equiva- ery shaft. Such work shall not be deemed a part of the assessment he Revised Statutes of the United States. The locator, or some one ork upon and has knowledge of the facts relating to the sinking of the ll make and attach to the copy of the notice of location to be recorded g the compliance by the locator with the provisions of this section, be recorded with such copy of the location notice (Laws, 1901, p 141

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Number of claims that may be located by hold one claim by location, as hereinafter provide by purchase as the local laws of the miners in th may allow; and the discoverer of any new lead or be allowed one additional claim for the discovery be so construed as to allow any person, not th claim upon any lead or vein (Hill's Annot Laws c

Location on stream or contiguous to placer to discharge debris. § 8. Any location of an stream, or contiguous or near to any placer mine, mine, shall be subject to the prior right of all mi such location, to discharge debris, gravel, earth, a or may be discharged, at the time of making such claims (Laws, 1901, p 122).

Grubstake contracts to be in writing. § 9 ship, commonly known as "grubstaking," shall the recorder of conveyances of the county where Such contracts must contain: *first*, names of the of; otherwise, such contracts shall be null and v

Mining claims may be located on state la mineral claim upon state land shall be in accord ing the location of mineral claims on governme citizens who may have found minerals on unsold this act, and posted notices in accordance with t United States, shall have preference right to le after the passage of this act, in which to make such lease (Laws, 1907, p 214).

(For reference to miscellaneous legislatio

24. South D

Width of lode. § 2. Width of lode claims vein or crevice; provided, that any county may less width than above specified, provided, that le lode shall be prohibited (Rev Pol Code, 1903, § 2

Location certificate, effect of failure to co Any location certificate of a lode claim which sha name of the locator, the date of location, the nu of the discovery shaft, the number of feet in widt and such description as shall identify the claim (Rev Pol Code, 1903, § 2 535).

Marking surface boundaries. § 6. (Same

Time within which discovery shaft must Dakota law, § 8.)

Relocation of abandoned lode claims. § 1

Location certificate must contain descriptio as Colorado provision, § 15.)

(For references to miscellaneous mining l

25. Utah

Notice of assessment work being done. P owning a group of claims and doing the develop at one point, shall post a notice upon each claim a such work is being done, and also post a notice said work is done, stating the names of the cla Laws, 1907, § 1 499).

ecording affidavit of performance. § 6. The owner of any mining claim • • • must within 30 days after the completion of ments file in the office of the county recorder of that county in which min ng district in which such claim is located is situated his affidavit, person or persons who performed or directed such labor or made or ements, and shall file a duplicate thereof with the district mining t in which said claim is situated, showing: *First*, Name of claim and d, Number of days' work done and the character and value of the thereon; *Third*, Date or dates of performing said labor and making d number of cubic feet of earth or rock removed; *Fourth*, At whose d work was done or improvements made; *Fifth*, Actual amount paid rovements, and by whom paid, when the same was not done by the id claim.

uly certified copies thereof shall be *prima facie* evidence of the facts Laws, 1907, § 1 500).

mining districts. § 7. Mining districts may be organized, and ay be reorganized and the rules and regulations of the said districts district according to the laws of the United States, in cases where a s desired; provided, that the nearest boundary line of any mining ithin 10 miles from the county recorder's office of any county (Comp

o miscellaneous mining legislation, see Lindley, p 2 546.)

26. Washington

claims governed by law in force at date of location. § 1. on any veins or lodes of quartz or other rock in place, bearing gold, le mineral deposits heretofore located, shall be governed as to length e by the customs, regulations, and laws in force at the date of such allinger's Codes of 1909, § 7 351).

of lode claims. Discovery. End Lines. § 2. • • • nor shall by any mining regulation to less than 50 ft of surface on each side vein or lode, at the surface, excepting where adverse rights, existing roval of this act, shall make such limitation necessary. • • • (Rem of 1909, § 7 352).

Contents. Record. (Act of 1899.) § 1. Discoverer of a lode from the date of discovery, record in the office of the auditor of the lode is found, a notice, etc (Laws of 1899, p 69).

rfeited claims, relocation of. § 8. (Provisions practically same , § 12.)

n. Location notice, posting, contents, recording. Marking opment work, affidavit of. § 10. • • • *Third*, within 60 days overy the discoverer shall perform labor upon such location or claim e to an amount which shall be equivalent in the aggregate to at least or for each 20 acres or fractional part thereof contained in such loca- • • (Rem & Ballinger's Codes of 1909, § 7 367).

Power to apply road-building on assessment work. § 14. Any have the power to make road-building to mining claims within such assessment work or improvement upon such claims; etc., • • • • Codes of 1909, § 7 371).

o miscellaneous mining legislation, see Lindley, p 2 556.)

27. Wyoming

aim. § 2. The width of any lode claim located within Wyoming on each side of the discovery shaft, the discovery shaft being always the side lines of the claim. Nor can any mining district limit the less than 150 ft on either side of the discovery shaft (Comp Stats

Discovery shaft. Posting notice. Marki of a location certificate in the office of the county discoverer of any lode, vein, or fissure, shall de *First*, by sinking a shaft upon the discovery lode lowest part of the rim of such shaft at the surface; on the surface, a plain sign or notice, containing t the discoverer, and locator, and date of such di boundaries of the claim, which shall be marked posts hewed or marked on the side or sides whic the ground, one at each corner, and one at the ce that no right to such lode or claim, or its posses person or persons unless such person or persons s ing rock in place (Comp Stats, 1910, § 3 469).

Time within which discovery shaft must be lode or vein in this state shall have 60 days from in which to sink a discovery shaft thereon (Comp

Location certificate, contents and record. lode, ledge, or vein shall within 60 days from da recorded in the office of the county clerk • • • (

Location certificate must contain one locati provision. § 15.)

Relocation of abandoned claim. § 11. (I § 12.)

Placers. Annual labor. Character of. § work as hereinafter provided shall be done durin first day of January following the date of locati in manual labor, permanent improvements mad ditches made for the benefit of working such clai work done accrues to the improvement of the clai the part of the owner or owners, and their intentio Stats, 1910, § 3 475).

Amount of annual labor. Time of perform tofore or hereafter located in this state not le shall be performed during each calendar year, fro of location (Comp Stats, 1910, § 3 476).

(For references to miscellaneous mining le

NOTE. — Attention is again called to the California, only those state laws are quoted from the California laws, and are importa mining claims.

SUMMARY OF UNITE

28. How Mining Rights

Discovery. Prerequisite to initiation of or placer claim, is the discovery of valuable to the U S. From the earliest times, the entitled to reward, and his recognition has stowal of title of nobility. Gamboa remark ing to the common benefit of society are justl of mines, the precious metals produced fro and the animating spirit of commerce, are such encouragement." He also speaks of " eral prosperity of the state mainly depends chap viii.)

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is the second condition of the right of possession. Thus Discovery is made the source of title, and development, or dition of the continuance of that act." Just as there can be iation of a mining claim without an antecedent discovery, so ning states, except California and Utah, a certain amount of rk is essential to the completion of a location. In addition to work, a certain amount of labor and improvement is required as erpetuation of possessory title. This amount is fixed by the $100 worth annually, and is by different states made to apply ments than actual excavation, or translated into an equivalent

otice is not specifically required by the federal law. Such ever, made one of the steps preliminary to completion of loca- of practically all the mining states; and the place and manner of prescribed. It is, moreover, in some instances specifically pro- ation shall be void if the requirement be not complied with. It is le in each case to read carefully the local or state regulations.

ocation on the ground, so that its boundaries can be readily the requirements of the federal statute. This law has been most of the mining states; and the number and character of nonuments particularly specified.

tice of location is not demanded by the national law, but is the laws of all the mining states. The time allowed in which from 30 days after date of discovery to 90 days after location. is needed.

29. How Mining Rights may be Lost

a mining claim may be caused by failure of the claimant to per- acts required by the laws for maintenance of possession. On S Supreme Court has said that right of possession "may be for- ure of the locator to do the necessary amount of work; or if, several locators, he neglects to pay his share for the work one by his co-owners, his right and interest in the claim may be co-owners under the provisions of the statute." Except in re will only ensue upon the lapse of the statutory period, upon nt the claim, and upon entry and location by another." (Lind-

t likewise results in loss of mining rights. In the same case as Elkhorn Mining Co), the Supreme Court held that "his interest also be forfeited by his abandonment, with an intention to t of possession." In this connection Judge Lindley remarks: perates instanter. Where a miner gives up his claim and goes hout any intention of returning, and regardless of what may who may appropriate it, an abandonment takes place, and the to its original status as part of the unoccupied public domain." 7.) "Abandonment is always a question of intention. In ment of intent is not involved." Resumption of work in good tion of possessory rights by others operates to prevent forfeiture.

30. Patent Proceedings

leed or grant of the U S, conveying the title authorized to be ng or other land laws. It establishes the character of the l-

which it conveys, and is conclusive eviden
steps leading to its issuance and delivery.

Land Department is that quasi-judicial
diction over the disposition of lands of the
Congress and the supervision of the Depart
ment consists of the Secretary of the Interio
Land Office, and subordinate officers, such
offices, and surveyors-general. These official
former for the term of four years; the latter
general may appoint deputies who are subje
missioner.

First step in proceeding to obtain a pa
is to make application for a survey. The ap
general of the district, must be signed by on
in behalf of all the petitioners, and should

1. Name or names of the claimants in full.
2. Name of each location embraced in the ap
3. Names of the mining district, state, and cou
4. Name of the U S deputy surveyor to whom

Applicant must also furnish a copy of the
be embraced in the survey, each duly certifi
and similar copies of each amended location,
estimated cost of fees in office of the sur
employ any deputy mineral surveyor in th
additional to the amount deposited with th
between deputy and claimant by mutual a

In making the survey the following po

1. Exterior boundaries of claim, number of fee
possible, direction of the vein, and number of fee
point of discovery.
2. Points of intersection of survey lines with th
3. Conflicts with unsurveyed claims, where a
area in conflict.
4. Total area of claim, and also area in conflic
5. If claim is situated within two miles of any co
notes must show a line connecting the claim with
within two miles, it must be connected with a U
6. End lines of each claim embraced within th

Where PLACER CLAIMS are upon surveye
divisions, no further survey or plat is requi
the placer, it must be described in the surv
situated elsewhere.

31. Adverse

Any person asserting an adverse minera
land for which patent is sought may file an
proceedings are called for only where one mi
another mineral claimant." Controversies
whether mineral or agricultural, etc, are not
but may be heard by the land department
adverse claim shall show the nature, bounda
proceedings, except the publication of notic
davit thereof, shall be stayed until the con

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rt of competent jurisdiction, or the adverse claim waived. Ex-dverse claims must be filed prior to the expiration of the 60 days or publication of the application notice. In Alaska, they may the time of publication or within 8 months thereafter. The led with the register and receiver of the land office in whose is situated. Within 30 days the adverse claimant must then a court of competent jurisdiction, to determine the question of on, and prosecute the same to final judgment. The jurisdiction s involving title to real estate is in the state courts; and such st invariably heard by such courts. Where no action is com-before the expiration of the 30 days, the land department may dverse claim had ever been filed.

ure of Title Conveyed by Mineral Lode Patents

e claim conveys: 1. Exclusive right of possession and enjoyment of uded within the limits of the location, as described in the patent, sub-isting easements.

les, and ledges throughout their entire depth, the tops, or apices, of ne boundaries, the right to pursue the vein in depth outside of such imited, however; • • • •

such a patent confers the right to everything found within vertical ugh the surface boundaries; but these boundaries may be invaded by ator holding the apex of a vein under a regular valid location, in the on its downward course underneath the patented surface (Lindley,

onveys to the patentee everything within vertical planes drawn down-surface boundaries, EXCEPT: 1, such lodes or veins whose tops or he placer limits, whose existence was known prior to the filing of the cer patent, and were not included in the placer application; 2, such aving their tops or apices elsewhere, as may underlie the placer sur-ay lawfully be taken by the apex lode locator under a regular valid uing his vein on its downward course. In the last class of cases the of location is wholly unimportant (Lindley, p 1 915).

33. Extralateral Right

gin. The right given by the mining laws of the U S to follow the surface of land owned by another was aptly termed the ht" in an early discussion by Dr. R. W. Raymond (*Trans* p 387), and has passed into current use in legal phraseology. the law has given rise to interminable litigation; and in its olumes of opinions have been furnished by the courts and oks. The manner of its introduction into our system of laws be clearly understood. It is at variance with the ordinary ican codes, and is not to be found in mining laws of any country was tried for a long period by the Germans in the sixteenth and uries, where a sort of inclined location was allowed, the miner not only to follow the vein beneath the surface, but having also 5 or 30 ft) on each side of the vein, parallel to its walls, within R. W. Raymond, "Mineral Resources," 1869, p 195); but it oned because it gave rise to so much litigation. It was later ime in British Columbia, but promptly given up, for the same lark, *Trans* A I M E, Vol 43, p 617.) It may have arisen from laws resulted from the local rules of the miners, and that these luct of the evolution and development of the western mines, ace rights only, and at first solely with reference to placers, an

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being later extended to lodes. Each man l
regardless of its dip or inequalities and w
neighbors. In many districts, moreover, t
early days, that the veins were all parallel
such complexities of structure and uncertai
could not have been foreseen.

Extralateral law is contained in Sec 2
full, Art 7). Section 2 320 provides that en
to each other. It is not believed that this p
extralateral rights, where end lines conver
lines must not be broken (Lindley, p 863
seems to depend somewhat on the definitio
apex of the vein to such portions of the
summary of the court decisions relative to
rights was published by Judge Clayberg (
E & M Jour, Sept 20, 1913, pp 537–543), a

Ideal claim defined. An ideal quartz
land, 1 500 ft or less in length, along the vei
illust
show
and
Fig
claim
of, a
by its
locat
strik

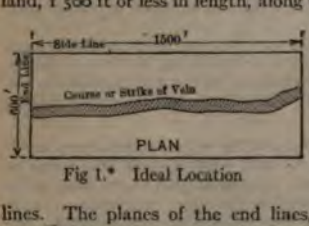

Fig 1.* Ideal Location

lines. The planes of the end lines, drop
extended in their own direction until they
vein, bound the extralateral rights on such
ideal location is the exception rather tha
rule. Such conditions result from the ina
of the locator to determine the actual cou
strike of the vein at the time of completi
location. As a matter of fact, the locato
mining claim usually judges such course or
from a small portion of the vein disclosed
place of discovery. It is almost impossible
termine correctly the true course or strik
vein from such a small exposure. Veins are
ular in their course or strike, and may be
running through the earth in almost every
ceivable direction.

Principles of extralateral rights. The
cation of the law of extralateral rights, a
vided for by the Act, to the varied cond
which have presented themselves, dependi
the course or strike of the vein within th
face boundaries of the location, and the fo
shape of the location itself, has been the
of prolific litigation in the mining regions o
certain well-established principles must al

* Fig 1 to 10 reproduced by permission fr

of extralateral rights is the EXISTENCE OF THE APEX of vein within es of the location, dropped downward perpendicularly and there- along the vein should always be limited by the extent of the apex . This is recognized by the Supreme Court of the U S in Del Co *vs* Last Chance Mining Co (U S, 55) in which the court says:

e in mind in considering the questions presented, that we are dealing ory rights. There is no showing of any local customs or rules affecting in and prescribed by the statute, and beyond the terms of the statute, . They have no power of legislation. They can not assume the exis- al equity, and rule that by reason of such equity a party may follow a ory of his neighbor and appropriate it to his own use. If cases arise has made no provision, the courts can not supply the defect. Congress he conditions upon which extralateral rights may be acquired, a party within those conditions, or else be content with simply the mineral e of his territory. It is undoubtedly true that the primary thought of sposal of the mines and minerals, and in the interpretation of the statute se must be recognized and given effect. Hence, whenever a party has the ground within whose surface area is the apex of a vein, with a few its course or strike, a right to follow that vein on its dip for the same awarded to him if it can be done, and only if it can be done under any nstruction of the language of the statute. If the surface of the ground el, and veins constantly pursued a straight line, there would be little ion to provide for all contingencies; but mineral is apt to be found in as where great irregularity of surface exists, and the course or strike egular as the surface, so that many cases may arise in which statutory o secure to a discoverer of a vein such an amount thereof as equitably ught to receive.

at extralateral rights exist in any location, its END LINES MUST c 2 320, U S Rev Stats, Iron Silver Mg Co, *vs* Elgin Mg Co, It is not held that the end lines of a mining location must be that such location be valid, but only that they must be par- give extralateral rights to any vein, the apex of which may lie ce boundaries A prospector ight to make uch shape as will be held herwise com- . It has also at the paral- lines need not lly accurate; l parallelism essary.

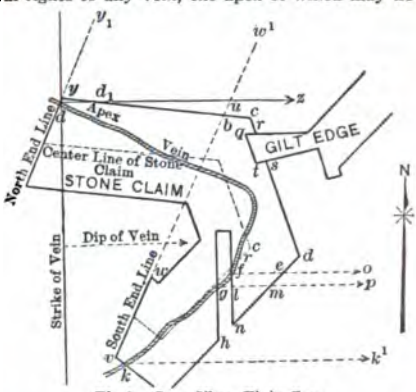

Fig 3. Iron Silver-Elgin Case

on Silver Min- lining Co, the was one of pe- Fig 3). The nes on the loca- were parallel, of the location vein passed on efore not end lines governing extralateral rights. The peculiar condi- the vein was probably occasioned by erosion. The vein lying very orces of nature had disintegrated certain portions thereof, by wearing of necessity, caused a portion of the apex of the vein to recede from ad formerly occupied.

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The only method whereby the subject of e
discussed is by considering the various conditio
and applying the law to each condition.

1. **Ideal location.** When apex of a vein c
end lines of a location, which are parallel to e
to such vein are clearly defined by the statute,
planes of the end lines of the claim, extended
intersect the exterior portions of the vein. T
trated in Fig 1 and 2, and needs no further co

2. **Outcrop cutting both side lines.** W
passes through both side lines of a location, wh

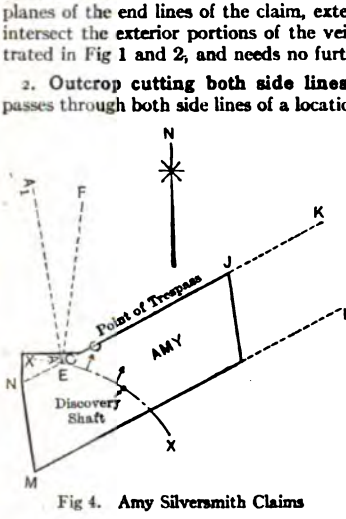

Fig 4. Amy Silversmith Claims

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Fig

3.
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can exist, because the end lines are not parall
much of the vein on its dip as he has length o
lines. If the apex of a vein within the boun
number of feet in length, equitably the
owner of such location should be given
the same number of feet on the strike of
the vein extralaterally, at any depth, and
no more (33 Mont, 46). See Fig 5.

4. **Outcrop cutting two converging boundaries.** Where the apex of a vein crosses two boundaries of a location which converge toward each other in the direction of the dip of the vein, it has been held, in the case of Carson City Gold Mining Co vs North Star Mining Co (86 Fed, 658), that the extralateral rights would exist, bounded by the vertical planes of the end lines, projected in their own direction this point Judge Lindley states:

Fig

Fig

There can be little doubt that extralateral rights or patents issued under either Act where the end din of the vein, giving the locator or patentee less in

however, that the decisions of the highest courts are not yet question, and some lower courts have held the contrary of the herein. This case is shown in Fig 6.

tting end and side line. Where apex of a vein on its course e and one end line of a location, the doctrine has been established urt that extralateral rights are bounded by a plane dropped perward through hrough which d a plane parropped downarly through e vein departs daries, across ese planes exwn direction extralateral n under such Monte Mining Mg Co, 171 s Fitzgerald, Fig 7.

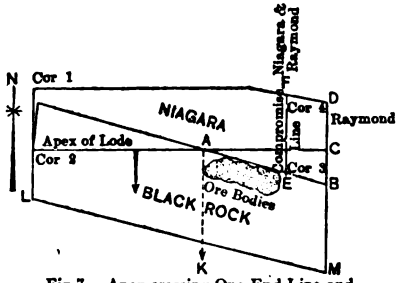
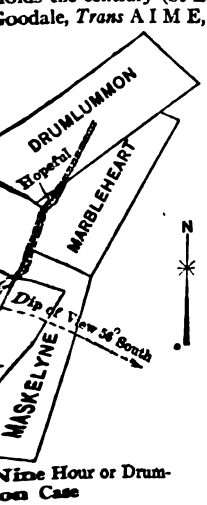


Fig 7. Apex crossing One End Line and One Side Line

ering and leaving through same side line. Where apex ourse enters and departs from the claim through the same Supreme Court of Colo has held that no extralateral rights Olds, 23 Colo, 433). The U S Court of Appeals for the Ninth olds the contrary (St Louis Mg Co *vs* Montana Mg Co, 104 Goodale, *Trans* A I M E, Feb, 1914).



Nine Hour or Drumon Case

The extent of extralateral rights on a vein in a case such as that last referred to presented an interesting question. The vein in controversy was a wide one, entering and departing from the St Louis claim, at an acute angle, through the same side line. At the point of departure from the St Louis claim, the apex of the vein was conceded to be 25 ft wide, measured along the boundary which it crossed. The St Louis claim was conceded to be the prior location, and the owner thereof claimed that it was, therefore, entitled to all extralateral rights on the vein, so long as any portion of its apex remained within the boundaries of the claim. The Court of Appeals upheld this contention, saying:

> The only deduction which can be made from the foregoing rules is that, inasmuch authority permits a division of the crossing portion of the vein, hority favors the senior locator, the entire vein must be considered or location until it has wholly passed beyond its side line.

his condition, showing properties in conflict in St Louis case

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This decision of the appellate court has
and thus appears to have received the san
of the land. In a later opinion the Co
Fed, 61):

We are therefore of the opinion that the rig
rights in the Drumlummon to the extent that
claim, has been previously determined by this
affirmed by the Supreme Court of the U S (20
law of the case.

The vein in question was an incident
covery vein; but the rule is the same, beca
of a lode claim is entitled to follow extral
surface boundaries. The doctrine here a
the same extent along the course of the v
claims as that which he has upon the sur

7. **Wide outcrops.** Where the apex
by a boundary line of a location, or whe
itself, the U S Su
location is entitle
vein, and the jur
of the senior an
Terrible Mg Co,
207 U S, 1; also
Co, 114 Fed, 41

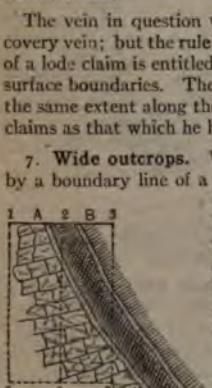

Fig 9. Vein Split by Boundary Line (Vert Sec)

8. **Outcrop c**
apex of an origin
through one lin
reaches any othe

9. **Outcrop c**
apex of the vein
cross any boundary. These two condi
Supreme Court, in discussing the Del M
Chance Mg Co, 171 U S 55), as follows:

The law places a limit on the length of the
does not say that he shall not go outside the v
reaches the end lines. Nowhere is it said th
below the surface extends from end line to e
dip outside the vertical side lines. Naming
not equivalent to saying that nothing is gran
The locator is given a right to pursue any v
on its dip outside the vertical side lines, but m
end lines. And this is all the statute provides
terminates halfway across the length of the
dip beyond the vertical side lines is as plainly
it had extended to the farther end line. It
of such surface lines extended downward vert

The Court of Appeals for the eighth
Min & Mill Co vs Doctor-Jack Pot, 194

It does not follow that to secure extralater
to end line, or, for that matter, intersect eithe

NOTE. — The rule laid down by Judg
ground in cases 8 and 9. Veins are some
differential movements which give them

with continuity beneath the surface. In such case, the extra-
ght be bounded by vertical planes parallel to the end lines of
l through the ends of the apex on the surface rather than by
line planes themselves.

departing and returning through same side line. Where
ough both end lines of a location, but in its course departs from
h a side line and reënters the claim through the same side line
o be still a situation without specific ruling by the courts, al-
in the case of Waterloo Mining Co *vs* Doe (82 Fed, 45) as fol-

odes having their apex in the ground patented. The fact that a part
e in the ground patented would not give any right to any part of the
hich was not therein, although the apex might be cut by both end lines
nises.

therefore be concluded, in general, to have extralateral rights
allel to the end lines, for each segment of apex included within
aries.

lateral Rights on Incidental or Secondary Veins

xtralateral rights to *all* veins whose tops or apices lie within
of the location. The vein on which the location is based is
very" or "original" vein; others are called "secondary" or
s. While all veins have equal and similar rights under the
on or discovery vein, in its relation to the surface boundaries,
letermines the direction of the planes limiting the extralateral
veins apexing within the claim.

the courts in the Providence-Champion case (63 Fed, 552;
Sup Court Rep, 909, 43 L Ed 170), the Drumlummon case
, Ltd, *vs* St Louis M & M Co, 102 Fed, 430; 104 Fed, 664),
d (Jefferson Min Co *vs* Anchoria-Leland M & M Co, 32 Colo,
o), and Ajax *vs* Hilkey, in the Supreme Court of Colorado
o *vs* Hilkey, 31 Colo, 131; 72 Pac, 447; 22 Morr Min Rep,
se to much discussion. The weight of authoritative opinion
he views of the Colorado court, contained in the following:

titute a barrier, beyond which a locator can not follow a vein on its
a discovery or a secondary vein; and they also limit the bounding
his extralateral rights are to be exercised in following such vein on
; such extralateral rights the locator can not in any case pursue the
d the bounding planes drawn through the end lines. • • • The
epends upon the length of the apex, and the extralateral rights are
rly by the end lines, and only so when the vein passes across both
nding planes drawn parallel to the end lines passing through the
ere it enters into and departs from the same. It would seem, there-
low that the extralateral right depends, *inter alia*, upon the extent
e surface lines, and, while the end lines of the claim as fixed by the
ines of all veins apexing within its exterior boundaries, the planes
ts of different veins may be as different as the extent of their respec-
l such planes must be drawn vertically downward parallel with the
o difference in what portion of the patented claim the apex is. Its
er this rule can be easily ascertained.

summing up the opinion of Justice Brewer in the Del Monte
ws:

at for all veins, both discovery and secondary, of a patented cl
teral rights at least for so much thereof as apex within the su

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lines: that such rights as to secondary veins are
the same segment of the claim in which the apex
the end lines of the location as fixed and descrit
veins apexing within the surface boundaries, and
such extralateral rights, and in no case can the l
the surface lines beyond such planes continued i
such veins, yet these bounding planes, which in a
lines, need not be coincident.

Judge Lindley (p 1 378) remarks that "t
apply to all secondary veins at the points
end li
the o

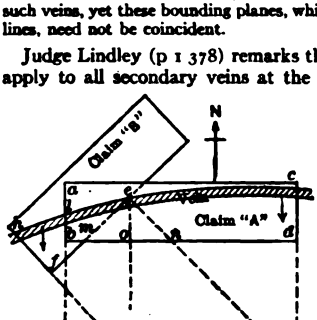

Fig 10. Conflicting Extralateral Rights

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its en

If *A* be junior, the vein from *h* to *E* belongs
end line and the parallel plane *ek* (with possi

35. Veins Dipping into and ber

Whether veins properly located under th
beneath the surface of patented agricultur
cided by the U S Supreme Court. The N
(Amador Min Co *vs* S S H Min Co, 36 Fe
followed by their owners beyond the points
the surface of such prior agricultural gran
takes issue strongly (p 1 462), as follows:

> We simply affirm that lodes or veins having
> grant are, to the extent that such lodes or vein
> reserved by law out of such grant. Nor does
> established rule of law that the patent is conclusi
> The land covered by an agricultural patent is
> but this does not necessarily imply that a lode
> cannot be reserved without impeaching the pat
> the land. The two classes of grants may exist w

By mutual agreement, the boundaries
abolition, may be legally fixed, thus doing
of the federal law. Indeed, such agreemen
few years in many of the copper mining dis

36. Some Unsettle

The opinion is frequently expressed that
of extralateral rights under the statute of
marvellous perfection as a piece of trouble-
legislation is abundantly illustrated by the

his statement is still far from true. Judge Lindley remarks ack of harmony between courts of different jurisdictions in the he mining laws is a matter of frequent occurrence. For this often remains in a state of uncertainty until the questions are y the Supreme Court of the U S. Even then, courts at times proper construction to be given to the decisions of the court of

g, in the article quoted above at length, admits that "the locations may be so variant, and the position of veins within daries thereof so strange and peculiar, that many complications, of, may present themselves for adjudication." On this point xpressed himself as follows:

at the U S Supreme Court has handed down decisions which definitely principles of apex law, it is very far from true to state that no other questions remain undetermined. There are points upon which the eal in different circuits are in direct conflict. There are questions in- now pending which have never been passed upon by the Supreme hese questions contain possibilities which one shudders to contemplate. aws Should be Revised. H. V. Winchell, *Trans* A I M E, v 48, p 361.)

7. Coal Land Laws of the United States

with authority given in sec 2 351 (below) the land department regulations and blank forms for the purpose of making the manner provided in the law. These may be had upon applica- ment.

nds, who may enter. Size of location. Price. Every person years, who is a citizen of the United States, or who has declared his such, or any association of persons severally qualified as above, shall, the register of the proper land office, have the right to enter, by legal antity of vacant coal lands of the United States not otherwise appro- by competent authority, not exceeding 160 acres to such individual to such association, upon payment to the receiver of not less than $10 nds, where the same shall be situated more than 15 miles from any and not less than $20 per acre for such lands as shall be within 15

tial rights of purchase. Any person or association of persons sever- ve provided, who have opened and improved, or shall hereafter open al mine or mines upon the public lands, and shall be in actual posses- all be entitled to a preference right of entry, under the preceding s so opened and improved; provided, that when any association of ersons, severally qualified as above provided, shall have expended in working and improving any such mine or mines, such association ding 640 acres, including such mining improvements.

ory statement, when filed. All claims under the preceding sec- ed to the register of the proper land district within 60 days after the sion and the commencement of improvements on the land, by the statement therefor; but when the township plat is not on file at rovement, filing must be made within 60 days from the receipt of ict office; and where the improvements shall have been made prior months from March 3, 1873, 60 days from the expiration of such wed for the filing of a declaratory statement, and no sale under the ion shall be allowed until the expiration of six months from March 3,

on shall make but one entry. The three preceding sections shall nly one entry by the same person or association of persons, and n any member of which shall have taken the benefit of such sections

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either as an individual or as a member of any o
other lands under the provisions thereof; and n
have taken the benefit of such sections shall en
provisions; and all persons claiming under § 2 3
spective rights and pay for the lands filed upon
for filing their respective claims; and upon fail
for the land within the required period, the sam
qualified applicant.

§ 2 351. **Conflicting claims.** In case of conf
improvements shall be commenced after March 3,
ment, followed by proper filing and continued g
right to purchase. And also where improvement
3, 1873, division of the land claimed may be m
near as may be, the valuable improvements of th
of the General Land Office is authorized to issue
ing into effect the provisions of this and the four

§ 2 352. **Application of preceding sections.**
shall be construed to destroy or impair any ri
March 3, 1873, or to authorize the sale of lands va

38. Alaskan Coal-leasing Act

The Secretary of the Interior is hereby author
the United States in the Territory of Alaska kn
coal, preference to be given first in favor of surve
known as the Bering River, Matanuska, and N
areas or coal fields as lie tributary to established
or water transportation lines: *provided*, that such
with existing laws and rules and regulations gove
is hereby appropriated, out of any money in the
sum of $100,000 for the purpose of making the
available until expended: *provided*, that any surv
or by the approval of the Department of the In
purposes of this Act.

§ 2. The President of the United States shall
sale, lease, or disposition not exceeding 5 120 acres
field and not exceeding 7 680 acres of coal-bearin
to exceed one-half of the other coal lands in Ala
such reserved areas may be mined under the
opinion, the mining of such coal in such reserved a
becomes necessary, by reason of an insufficient su
requirements of Government works, construction
for the Navy, for national protection, or for relief

§ 3. The unreserved coal lands and coal depos
the Interior into leasing blocks or tracts of 40 acr
form as in the opinion of the Secretary will permi
in such blocks, but in no case exceeding 2 560 acr
thereafter, the Secretary shall offer such blocks o
ated minerals therein for leasing, and may awar
competitive bidding, or such other methods as
any person above the age of 21 years who is a
association of such persons, or to any corporati
laws of the United States or of any State or Ter
of the stock of such corporation shall at all tim
United States: *and provided further*, that no railro
to take or acquire through lease or permit under
of such area or quantity as may be required an
limitation of use shall be expressed in all leases o
carriers hereunder: *and provided further*, that
qualified to become a lessee under this Act and ow
land laws to any coal lands in Alaska, may, withi

gement with the Secretary of the Interior by which such claim shall be
o the United States; and if in the judgment of the Secretary of the
stances connected with such claim justify so doing, the moneys paid
claimants to the United States on account of such claim shall, by direc-
y of the Interior, be returned and paid over to such person, association
consideration for such relinquishment.
ing rights to any of such lands in which final proof has been submitted
pending before the Commissioner of the General Land Office or the
terior for decision shall be adjudicated within one year from the pas-

ssociation or corporation holding a lease of coal lands under this act
oval of the Secretary of the Interior and through the same procedure
terms and conditions as in the case of an original lease under this Act,
new lease covering additional lands contiguous to those embraced in
ut in no event shall the total area embraced in such original and new
aggregate 2 560 acres.
showing by any lessee to the Secretary of the Interior that all of the
f coal within a tract covered by his or its lease will be exhausted,
oved within three years thereafter, the Secretary of the Interior may,
n, lease to such lessee an additional tract of land or coal deposits,
e coal area remaining in the original lease, shall not exceed 2 560
ame procedure and under the same competitive conditions as in case

he approval of the Secretary of the Interior, lessees holding under
r areas may consolidate their said leases or holdings so as to include
ot to exceed 2 560 acres of contiguous lands.
all be for such leasing block or tract of land as may be offered or ap-
ing in area 2 560 acres of land, to be described by the subdivisions of
person, association, or corporation, except as hereinafter provided,
take or hold any interest as a stockholder or otherwise in more than
this Act, and any interest held in violation of this proviso shall be
ted States by appropriate proceedings instituted by the Attorney
pose in any court of competent jurisdiction, except that any such
t hereby forbidden which may be acquired by descent, will, judgment,
d for two years, and not longer, after its acquisition.
ho shall purchase, acquire or hold any interest in two or more such
in provided, or who shall knowingly purchase, acquire, or hold any
n having an interest in two or more such leases, or who shall know-
to one disqualified to purchase, or except as in this Act specifically
to acquire, any such interest, shall be deemed guilty of a felony,
shall be punished by imprisonment for not more than three years
xceeding $1 000: *Provided*, that any such ownership and interest
ch may be acquired by descent, will, judgment, or decree may be
ts acquisition and not longer, and in case of minority or other dis-
he court may decree.
trustee, officer, or agent of any corporation holding any interest in
, on behalf of such corporation, act in the purchase of any interest
who shall knowingly act on behalf of such corporation in the sale
h interest in any lease held by such corporation to any corporation
any interest in any such lease, except as herein provided, shall be
shall be subject to imprisonment for a term of not exceeding three
exceeding $1 000.
ands or deposits leased under the provisions of this Act shall be sub-
sed, or controlled by any device permanently, temporarily, directly,
n any manner whatsoever, so that they form part of or are in any-
y combination in the form of an unlawful trust, with consent of
ect of any contract or conspiracy in restraint of trade in the mining
ed into by the lessee, or of any holding of such lands by any individ-
ation, corporation, or control in excess of 2 560 acres in the Terri-
e thereof shall be forfeited by appropriate court proceedings.
e of mining and extracting and disposing of the coal in the land

covered by his lease the lessee shall pay to the Un
specified in the lease, which shall not be less than 2
of each month succeeding that of the shipment of t
rental, payable at the beginning of each year, on th
rate of 25¢ per acre for the first year thereafter, 50¢
and fifth years, and $1 per acre for each and every y
of the lease, except that such rental for any year s
as they accrue for that year. Leases may be for pe
subject to renewal on such terms and conditions as
of such renewal. All net profits from operation of
and rentals under leases as herein provided, shall
United States, in a separate and distinct fund to be
Government of the United States on account of any
tion of railroads in Alaska, and the excess shall be
Alaska Fund, established by Act of Congress of Jan
in said last-mentioned Act.

§ 10. In order to provide for the supply of strict
the Secretary of the Interior may under such rules
in advance, issue to any applicant qualified under
permit granting the right to prospect for, mine and d
States on specified tracts not to exceed 10 acres to
sons in any one coal field for a period of not exceed
inconsistent with this Act as in his opinion will sa
payment of royalty for the coal mined or for the lan
sition or holding of a lease under the preceding secti
acquisition, holding or operating under the limite
And the holding of such a license shall be no bar to
lease or interest therein.

§ 11. Any lease, entry, location, occupation, or
reserve to the Government of the United States the
in, over, through, or upon the land leased, entered,
necessary or appropriate to the working of the sa
authority of the Government and for other purpos
his discretion, in making any lease under this Act,
right to lease, sell, or otherwise dispose of the surfa
lease under existing law or laws hereafter enacted, in
for use by the lessee in extracting and removing t
reservation is made, it shall be so determined before

The said Secretary during the life of the lease is
easements herein provided to be reserved, and to per
in the Territory of Alaska as may be necessary for
coal washeries or other works incident to the mining
be occupied and used jointly or severally by lessees
by said Secretary.

§ 12. No lease issued under authority of this Ac
with the consent of the Secretary of the Interior. E
the purpose of insuring the exercise of reasonable dili
of said property, and for the safety and welfare of
undue waste, including a restriction of the workday
for underground workers except in cases of emerge
complete freedom of purchase, requiring the payme
lawful money of the United States, and providing p
fair and just weighing or measurement of the coal
provisions as are needed for the protection of the i
prevention of monopoly, and for the safeguarding of

§ 13. The possession of any lessee of the land or c
all purposes involving adverse claims to the leased pr
of the United States, and for such purposes the les
the property leased as if operated directly by the Un

§ 14. Any such lease may be forfeited and canc
court of competent jurisdiction whenever the lessee

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l general regulations promulgated under this Act; and the lease may forcement of other appropriate remedies for breach of specified conditions

lter the approval of this Act no lands in Alaska containing deposits of om entry or sale shall be disposed of or acquired in any manner except s act: *Provided*, that the passage of this Act shall not affect any proceed- n the Department of the Interior, and any such proceeding may be carried ination in said department notwithstanding the passage hereof: *Pro-* no lease shall be made, under the provisions hereof, of any land, a claim ing in the Department of the Interior at the date of the passage of this less such claim is finally disposed of by the department adversely to the

ments, representations, or reports required, unless otherwise specified, of the Interior under this Act shall be upon oath and in such form and as the Secretary of the Interior may require, and any person making ntation, or report shall be subject to punishment as for perjury.

etary of the Interior is authorized to prescribe the necessary and proper ons and to do any and all things necessary to carry out and accomplish his Act.

and parts of Acts in conflict herewith are hereby repealed.

nmary of Act Authorizing Exploration for Potassium

(Approved, Oct 2, 1917)

es prescribed by the Secretary of the Interior, any citizen or associa- the U S, or corporation organized under State or Territorial laws, may ng permit, giving exclusive right for not more than 2 years, to prospect nerals on public lands, except lands in and adjacent to Searles Lake . 26, 27, south of ranges 42, 43, 44 east, Mount Diablo meridian), Cal; in such permit not to exceed 2 560 acres, in reasonably compact form. ving to the Secretary of the Interior that valuable potassium deposits rea covered by the permit, a patent may be secured for an area not rth of the prospecting area, taken in compact form, or, if unsurveyed t exceeding 2 miles long; survey to be made at cost of patentee. All prospecting permit, not covered by leases, may be leased by the Sec- t exceeding 2 560 acres, on payment by lessee of a rental in advance, alty as may be fixed by the Secretary. The royalty shall not be less ross value of the output at the shipping point, paid in the product of e rental shall not be less than 25 cents per acre for first year, 50 cents nd 5th years, and $1 per acre for each year thereafter, except that the ar shall be credited against royalties accruing in that year. Leases rminate periods; at the end of each 20-year period, a readjustment of made by the Secretary of the Interior.

in lands adjacent to Searles Lake, as exempted in § 1, may be operated sed by the Secretary of the Interior under the provisions of this Act. y also issue leases for potash deposits in public lands in Sweetwater ng coal, on condition that the coal be reserved to the U S.

or lessees under this Act may secure the right to use, during the life ase, a tract of unoccupied nonmineral land, not exceeding 40 acres, for sary to the proper development of the deposits covered by the permit

n up under this Act must be worked with due diligence, on pain of rmit or lease.

having interest in a lease under this Act may, directly or indirectly, ersonal or corporate interests hereunder as would make his aggregate a 50 miles square exceed the maximum number of acres allowed to one Also, no person or association, holding a lease hereunder, may hold interest direct or indirect in any other agency or association engaged e of the products from such lease.

l provide for the protection of the interests of the U S, and the pre- ly.

§ 9. This act applies also to all deposits of potassium s
been or may be disposed of under laws reserving such dep
to prospect and mine the same.

§ 13. Leases under this Act shall contain a provisio
regulate the price of all mineral mined and sold from the
the prices fixed shall be such as to yield a reasonable retu
ment, and secure a reasonable price to the consumer. F
have authority so to regulate the disposal of the potas
this Act as to secure their distribution and use wholly wi

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SECTION 25

EXAMINATIONS, VALUATIONS AND REPORTS

BY

WILLIAM YOUNG WESTERVELT

CONSULTING MINING ENGINEER

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...s in parentheses refer to Bibliography at end of this section.

1. General Considerations

examination is to form the engineer's judgment; that of a ...rth this judgment in such form and detail as will be most in... imately useful to the client.

...erament and circumstance of engineers and clients, coupled with that ...cation, size, value, and state of development of properties, occasions ... method and scope of both examinations and reports. Examina... ...nines that have been worked for considerable periods tend to become ...portion to growth of standard conceptions of operating principles ... an examination even of such a property, while giving due weight ...le, must be treated as an original problem if the client's needs are to

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Standardization of method is sought
gineers whenever feasible, but no method
likely to be developed.

Since the value of a mine is its resources for pro
more susceptible of accurate determination than
ideal case of an absolutely developed mine. Fluct
(except gold) may change a profit into a loss or
determinable whether or not further discoveries,
an apparently exhausted deposit, or whether in
facilities, or labor conditions, will render profita
engineer must bear in mind these uncertainties, us
probable combined effect and, though it is clear t
as definite a "yes" or "no" as is possible, the unc
sions should be so expressed in the report as will i
portance they occupied in the engineer's judgment

In this section an attempt is made to set fort
practice, to describe methods found useful under
tions which may help practitioners to meet the pr

Theory of procedure in an examination
assign to its proper place in the engineer's ju
bearing on the success of the enterprise req
able, or which will tend to prove that profit
attainable conditions. In examination work
far as to eliminate investigation which the
gain to his client proportionate to the outlay

Conservatism, though as essential to sou
other undertaking, should not be made an ex

To attain the closest approximation to fact (not
able data) is the engineer's obligation and the clie
An exact forecast being impossible, and error on t
less disastrous to the client than a mistaken opti
kept on the conservative side. Proper margins
underestimate of value is as great an error as an ov
the client, and it is but little less blameworthy, th
ous censure. As close an approach as possible t
made, and the engineer must do this fearlessly,
unfavorable aspects.

Relation and responsibility of engineer
lawyer. No honorable means of serving the c
and every effort should be made so to present
shall be that which the engineer's judgment i

Though the engineer is justified in charging such
worth to his client, or in contracting to do the work
own interest, it is improper and ultimately unprofita
ness of the examination (or the care in preparing
the fee. Within the scope of the examination (whic
preferably embodied in a letter or other writing be
has a right to expect that the engineer will give th

Client is the individual, individuals, or org
employed and to whom he is responsible, mal

In dealing with organizations, the engineer has b
that the legally authorized officers who employ hin
and will use his services solely for its advantage.
gineer, and to the ultimate advantage of his reputati
himself with the character and conduct of these
are not being used to the detriment of his true cli

the individual or individuals responsible for the examination
SSISTANTS are here included to the extent of their responsibility
ion.

on, Geology, Maps, Equipment, Accounts, and Reports

Distance from sources of SUPPLIES and from the MARKET for
o cost per unit, safety and reliability of existing means of TRANS-
oe ascertained. If transport improvements are essential or de-
oncerning means, costs, and advantages should be secured.

ng much high-grade ore have been successful in unfavorable locations and
eans of transport, as, for example, the backs of men; but hard-rock,
ties (the output of which has a gross value of say $5 per ton of crude ore)
ly when on rail or water transport lines. Difficulty of access to the
he FINANCIAL CENTER furnishing development capital may sometimes
ent of small or low-grade deposits. Location with respect to supply of
nd FUEL, or of HYDRO-ELECTRIC POWER, is of prime importance, and if
e are lacking, the cost of supplying them should be at least roughly esti-
E and its effect on operation of the property should be noted. Nature
ENT of the country in which the property is located is important, if there
its stability or ability and inclination to give proper protection to in-
cy to levy excessive taxation, to pass restrictive laws, or to obstruct
property by permitting illegitimate interference should be considered.

gy is studied for the light it may throw upon the probability of
sence of economic quantities and values of the minerals sought,
ion of exposed values to quantity of assured mineral (Art 11),
able persistence laterally and in depth of values and character-
evealed, and upon the best methods of exploitation.

itches, pinches, stratifications, and other rock characteristics, faults, horses,
condary enrichments of possible influence (Sec 2), should be considered in
e amount of ore proved by existing or likely to be proved by proposed
hese data, particularly when scanty, should be supplemented by similar in-
ing neighboring properties or districts; especially those having the same
f similar character with different values. Government or private geologi-
orts of the property and district, or of neighboring properties or districts,
igated for such information. U S Geol Surv folios, which include topo-
ological maps, geological sections, and descriptive matter, are valuable.
by writing the Survey office, Washington, D C, ordering by number as in
losing price in currency or P O order, payable to Director of the Survey.
o 1 to 184 inclusive, have been damaged by fire but are entirely usable.

ologizing should be undertaken so far as is likely to throw light
f the property, with expenditure of available time and money.
ude a thorough examination of surface and underground ex-
udy of existing drilling records or cores (Sec 9). Character and
l rocks, and of the deposits themselves, should be considered as
ce in determining the best method of exploitation.

Method and cost of working flat-lying seams, as many coals, clays, and
may be influenced by the strength and permeability of roof and floor.
, and relation to topography of the overburden of flat-lying secondary
the copper porphyries, determine the utility of the caving system of
removal of overburden and open-cut work (Sec 10). The workability
sit by dredging, hydraulicking, ground sluicing, or drift mining, is de-
as much by the character, condition, and slope of bedrock, occurrence,
ize of boulders, and depth and character of overburden as by the engi-
of quantity and head of water and debris disposal (Sec 10).

ine, property, and general topography, both private and
ould all be investigated and their reliability determined (a- f--

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Silver City (N Mex).....	....	25¢

as concerns data of interest) by comparison
They are studied for information as to di
lines, existing or possible transportation,
derground exposures, and drilling developm

Whenever practicable, copies should be made
are on linen, silver negatives are recommended
can be cheaply made when desired. Otherwise
be made, which, though more expensive, are rea
ing, maps should be supplemented by original
to the extent justified by the circumstances (S
GRAPHICAL MAPS are valuable for study of gener
the geological folios at 10¢ each, or from loca
sporting goods houses, who often carry, at an ad

Mine equipment should be examined
its alteration, enlargement or replacemen
tions. Except in cases of obvious inutility,
If much of the value of the property is in
acquired on purchase or lease, or if the
subsequent operations, a fairly complete
state the condition as well as the items of a
tions, grading, costs of erection, tools, supp

Items of plant having value if moved should
valueless except in place (as grading, foundatic
being designated as MOVABLE and FIXED PLANT.
cash and balance of good accounts receivable and
acter and cost of improvements and additions n
should be noted with more or less care and deta
bility in the matter, respecting machinery equip
transportation system, and general equipment, i
management, staff, foreman and general labor a
and general sanitation (Sec 22, 23).

Operating methods in use should be s
tages could be gained by modification or
in such detail as is justified by the circumst
change and of the probable saving to be eff

Accounts and reports; operative a
reports of enterprises that have been base
ined for the light they may throw upon to
equipment and operating costs, profits and
ized, expenditures and resources, the skill
tered in operating with the equipment used

These reports may range from daily to annu
production, assays, financial transactions and
Those of sufficient importance to justify it unde
studied by tabular compilation, as in Fig 1. By
ages and rates per ton, oz, lb, ft, or other unit, ca
against the ledger accounts, which in turn can
Reports by superintendents, managers, consulti
checked against the results of these compilatio
terminable reliability can sometimes be thus s
tions as to points requiring investigation. Dat
sometimes available and can also be profitably
are of sufficient importance, and the ledger accou
the services of an expert accountant may be adv

Terms of acquisition. The engineer
to be responsible for the effect the terms m

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Monthly Costs During 19--

and smelted	Mining		Trans		Smelting		General		Miso				Total	
	Amount $	Per ton ¢	Amount $	Per ton ¢	Amount $	Per ton ¢	Amount $	Per ton ¢	Amount $	Per ton ¢	Amount $	Per ton ¢	Amount $	Per ton ¢

Fig 1. Mode of Tabulating Working Data and Costs
(Blank columns for unexpected additions)

he should familiarize himself with these terms; if not (and the case fails to make this obvious) he should bring out the fact in an inoffensive manner.

3. Titles, Legal Assistance, and Forms

mining engineer may not be well qualified to pass upon questions nure and ownership are of such vital importance that he should thereon as fully as possible. Terms of tenure, such as operating rking bond or lease, or ownership in fee simple, and the chain of the tenure is based, should be thoroughly understood, unless the o take all responsibility, or has specifically limited the scope of n so as to eliminate these questions. In the latter cases the report ly and inoffensively state the situation. Otherwise the engineer gate along the above lines until he can make definite statements.

the engineer may have occasion to make one of the following statements In view of the client's assurances, all questions of title have been ignored; e been assumed to be perfect. B. The titles have been examined, or the them examined by competent attorneys (whose names should be given), t; or certain defects, which should be specified, have been found. C. In hich attorneys and engineer agree are remediable, recommendation should curring further expenditure on the property, or stopping it pending per- In case of irremediable defects, a joint opinion should be given as to ces of gain are enough greater than chances of loss to justify proceeding se defects.

ance (preferably that versed in local laws and practice when of er and reliability) should be secured not only on questions of cing any important agreement, as a deed, working bond, or escrow absence of legal advice, following forms (2) may be used:

Deed

, Made the.....day of.....A.D., 19.., between....., e first part, and....., the part ...of the second part; at the said part...of the first part, for and in consideration of the sum dollars, lawful money of the United States of America, to.....in said part... of the second part, the receipt whereof is hereby acknowl- ted, sold, and forever quitclaimed, and by these presents do.. grant, uitclaim, unto the said part.., of the second part, and to.....heirs he following described real estate, situate in.....mining dis- .....State of....., to wit: (Here follows description)... and singular the mines, minerals, lodes and veins within the lines of

said claims and their dips and spurs, and all dumps, privileges, and appurtenances thereunto in anywise lands, tenements, and hereditaments hereby conveye part . . . heirs and assigns forever.

In Witness Whereof, the said part . . . of the first and seal . . . . . the day and year first above written.

Signed, Sealed and Delivered
in the Presence of

. . . . . .
. . . . . .

(To be signed, witnessed, sealed, and acknowledg commissioner, to meet the requirements of the State situated. Provisions for recording should also be ob

Mining Leas

This Indenture, Made this . . . . . day of . . . . . , of . . . . . , lessor, and . . . . . .

Witnesseth: That the said lessor, for and in cons reserved, and the covenants and agreements hereina to be kept and performed, ha . . . granted, demised grant, demise and let, unto the said lessee, all the property, situated in . . . . . mining district . . . . . , to wit: (Here follows descriptio appurtenances to have and to hold unto the said les the date hereof, expiring at noon on the . . . . . day feited or determined through the violation of any co tenant . . . . . reserved.

And in consideration of the said demise the said said lessor as follows, to wit: To enter upon said p in the manner necessary to good and economical m amount of ore possible, with due regard to the devel premises as a workable mine, and to the special co insert Special Conditions and Agreements of Lease,

In Witness Whereof, etc. (See deed, except that signature of the lessee, with corresponding alteratio

Working Bon

(A lease and option, also known as a working bo of sale. At end of the mining lease above outlined proceed as follows:)

And in consideration of the foregoing lease and the and the well and faithful keeping of the covenants t right to purchase the said premises, together with a . . . . . dollars ($ . . . . .), to be dist On or before the . . . . . day of . . . . . , 19 . . , the ignate manner and place of payment), and on or b 19 . . , the sum of . . . . . (Here insert com

Time being of the essence of this contract as to of such payments the lessor will execute, acknowled and sufficient deeds to the lessee, or such person or c conveying the said premises clear of incumbrance. placed in escrow with some bank or responsible part due under the agreement, and who delivers the de

The forfeiture, surrender, or termination of the al the option void, and the above mentioned payments

It is expressly agreed and understood that this option to purchase only, and not as obligating the s

In Witness Whereof, etc. (See close of Mining Le

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Notice of Right to Water

ed claims the water running in this.....stream to the extent of inches for mining purposes to be conveyed by (ditch or flume) from this .....placer claim.

..., 19... .....

Locator

be posted near the outlet, and the following form duly recorded in the y recorder's office.)

on of Right of Way for Ditch and Location of Water

e presents may concern, know ye, that I, ..., of the county of in the State ....., a citizen of the United States, do hereby ish as a legal notice to all the world, that I claim, and have a valid right n, possession and enjoyment of all and singular, that tract or parcel of eing in the county of....., in the State of....., for the ex- ay for the purpose of constructing a flume or water ditch from..... .....placer claim, more particularly described as follows: Com- escribe the exact route for ditch or flume).

nd have a valid right to the enjoyment and use of.....inches of stream for mining purposes, to be conveyed through such flume or water m, together with all and singular, the hereditaments and appurtenances ing, or in anywise appertaining.

nd and seal this.....day of....., A.D., 19..

(Name)

.....

the stream....., 19...

ced at claim or at stream....., 19...

.. County of....., ss.

day of....., 19.., before me, a ..... in and for the county state aforesaid, personally appeared....., to me personally person who executed the foregoing written instrument, and acknowledged the same for the uses and purposes therein set forth.

nd and official seal

.....

Escrow Agreement

deed of the.....lode is hereby placed in the.....Bank in escrow. If A.B. shall place, or cause to be placed to the credit of said..... Bank of....., on or before....., 19.., ...dollars, then and in that case the said bank is hereby author- e enclosed deed to A.B., or his order. In case the said A.B. shall fail to sum of..... dollars, as above provided....., then hereby authorized to deliver the enclosed deed to the said C.D. and E.F. er.

(Signed) C.D
E.F.

9... (Place and date) A.B.

tion for purchase of a mine is desired by a third party, it is safest and er to put a deed in escrow. It saves incumbering the record, and averts ight arise concerning payment of money. The deed should be a war- , or mining deed, as agreed, fully executed and acknowledged, ready for a sealed envelope and placed in a bank, or left with a responsible person, nt written upon the envelope, as above.)

tion as to location notices and patent proceedings, see Sec 24,

Sampling; Methods of Crushing, Coning, and Quartering

s the process of securing a more or less representative part or e purpose of gaining information as to the composition of th tigation of the part (see also Sec 29, "Definition"). It is nece

sary unless the conditions are such that more
from records of operation. The process is
only by special care can the heterogeneous
gineer usually has to deal be made to yield
sentative. These may approach the truth
when used with full knowledge of their limi
the number of points from which equal and s
are taken is great enough in proportion to t
if these points are uniformly distributed, it i
actly representative of the average compositi
The greatest difficulty in securing an accura
bility of most of the interior of the mass. D
penetrated, can usually be sampled by sink
mineral masses in place, especially those ex
sampling is generally limited to EXISTING E
ordinary course of mining, and not to furni
venient points for taking samples.

Sampling is thus usually reduced to securing as
posures as circumstances admit and justify. The
determined by calculation of combinations made
the geology, and the relative position and size of
sampling vary in detail according to size, charac
of the deposit, the degree of accuracy required an

Crushing the sample. Unless a mass is
particles, or is practically homogeneous, one
sampling is
and power
feasible (Sec
But, much
sampling mu
crushers are
er's hammer
stamp head,

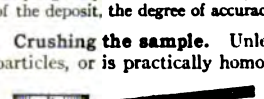

Fig 2. Ring for Crushing Samples

rock, are usually employed. Iron rings with
the piece on the anvil, so that few chips fly

Hand-hammer crushing should be done on a san
vent loss of material or introduction of foreign
operated through a hole in a canvas cover, to pre
flying chips, is good. Soft materials like coal may
in a light mortar, with iron-covered wood bottom
sible canvas sides (7) (Fig 3).

In working down a sample, the largest pieces mu
so small in proportion to the total volume that th
or rejection will have little effect on the result,
such a piece contain the maximum amount of impu
possible in the circumstances. Crushing to a max
piece weighing 0.001 of the mass to be quartered (a
in a 62.5-lb lot) is usually satisfactory. The extre
1-oz piece of chalcocite (containing say 75% Cu)
or lower the sample assay by 0.15% Cu, if it occur
sating errors in the whole series of quarterings wo
ably. If care is taken that very rich or impure pi
thoroughly mixed throughout the mass during the
quartering masses containing ordinary pieces seven
is known that no pieces of erratic value are present
increased. But, unless the work is in well-trained

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erratic values are present. Under prescribed conditions proper crushing y using a screen as is done by the U S Bureau of Mines, which provides a n with its coal-mine sampling outfit. But trained observation can deter- te the suitable size with sufficient accuracy (see also Sec 31, Art 2, 3).

l quartering (see also Sec 29, Art 3). The crushed mass is cone by shoveling all the material to one point on the quartering manner that the particles roll down in all directions from the Irregularities in composition of the mass are thus distributed ayers of a cone. The top of the cone is then flattened with the ovel, by spreading the material equally in all directions until a ned, the thickness being about 0.1 the diam. This disk is marked , and the diagonally opposite quarters are cut out, care being material in the rejected quadrants is removed, but no other. contains half the original quantity and, after reducing the large onally, coning and quartering are repeated, thus securing one- original. The cycle of crushing, coning, and quartering, is re- ne sample is of the desired size. With sufficient care that the e up of layers of each shovelful, evenly distributed about a fixed at the cones are flattened by being worked out evenly from the rections (coning about a fixed vertical rod (12) insures accuracy ts), results of any desired accuracy are obtained, provided suffi- s done either before or during the process.

mple is corrosive, the best quartering floor is a piece of boiler plate, large the disk of material prepared for the first quartering. When available, t shaft landings make good quartering floors. A well-laid floor, prefer- od, is the next best. When neither are available, a platform may be oards as solidly and evenly as possible on the ground and covering them sheet. Cloth covering is harder to keep free from residual fines than a or, but thorough brushing with a whisk, or scrub brush, usually suffices, elds better results than when working on a floor containing cracks in dge and be lost or inopportunely reappear. The whole process, includ- s often referred to as quartering.

hen using a sheet in reducing lots of 100 lb or less, time is saved ad of coning. It consists of mixing the mass by drawing one heet forward towards the diagonally opposite corner, until the en rolled over and over on itself to the limit of the capacity of e first corner being laid back in place and the diagonally oppo- ed over the sample, the process is reversed, and so back and terial then being left in a long mass across the center of the done at right angles to it, by working with the other diagonally of the sheet.

two rolls each back and forth with each set of corners, give good mixing ready for final rolling into the center by all four corners preparatory to ing may be imperfect unless the folded-over corner of the sheet is held is pulled forward; otherwise the material may slide on the sheet instead er on itself. Also, if the mass is wet and sticky it will adhere to the ing will be imperfect.

Decimating, Trench, Pit, Drill, Small-parcel, Dump, and Large-scale Sampling

he reduction of a mass by some type of splitter, or series of which alternately retain and reject a series of equally broad stream of crushed material poured over them (Sec 30, Art 3).

nes sampler (9) is recommended, as giving results equal to the best sing it, care should be taken to insure the usual proportion betwe

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the size of the largest piece and that of the mass ab
steady, full-width flow from the shovel; to see that
the apparatus is kept clean. The form shown in
cleaned with an ordinary brush. Splitting should
Quartering in such case is less objectionable than o
unavoidable, reduction of wet, sticky samples shou

Decimating. A mass in process of shipm
conveniently sampled by decimating; that is,
tenth car, barrow load, shovelful, or other uni

For accuracy, the number of units taken for sample
larity in composition of the mass, and if the units of
whole, every 5th, 3rd, or even 2nd unit may be requ
form materials, such as the product of a good coal s
accurate enough for most purposes. Ores of not very
of rich material may be satisfactorily sampled by ta
spotty ores may require 1 000 units, and it is difficu
without carefully working down the entire mass.
sample secured by decimation is crushed and reduc
splitting, or by use of a MECHANICAL SAMPLING equi

Trench sampling consists in cutting a seri
ficial deposit, and decimating or otherwise r
sampling the sides of the trenches. The sampl
of material composing the mass are cut in the

Pit sampling. A number of pits or ver
samples in a manner similar to trench samp
deposits, where large pieces usually accumul
composition are more marked with depth
good sample can therefore be secured if a suff
arly distributed as to cut uniformly across thi

Pits in dump material must be supported by tim
the vertical walls necessary to secure a uniform yie
and with short-handled tools, 36-in diam round sha
any support. Small elliptical shafts (13) only 28 i
dumps to depth of 50 ft, lining with corrugated ir
ported on ⅜ by 1.5-in elliptical iron rings, spaced

Drill sampling. Though a single pit yie
drill hole, where drill sampling is practicable
cost, many more points can be tested by drill

This is especially true in sampling flat deposits
coal seams, secondary enrichment "porphyry" cop
deposits of S E Missouri, and zinc deposits of Wisc
with the same regularity as pits, close enough tog
the deposit and the drilling must be done with care

Small-parcel sampling. Lots of a few ton
tering, splitting, decimating or, mechanically

With care any of these methods will give accur
cheapest and commonest method, is essentially th
by taking every alternate shovelful. The value m
grab sampling, particularly if the lot is first well sp

Dumps containing more than a few tons,
their value and irregularity justify it.

Decimation of rich dumps by shoveling is quite u
tionally irregular, dumps may be sampled with suffic
the latter, however, only when there are not enoug
drilling, or when the dumps are large enough to just

samples of many tons, or even thousands of tons, are taken e value of erratic ores; especially those containing metallics, as ver or copper, or very rich minerals, the average occurrence of etermined only by handling large amounts. Large samples are etermine amenability to treatment by mill or smelter tests.

aintaining efficient surveillance in taking and handling large samples sefulness that they are taken only where other methods fail, as in the tive-copper deposits, or where full-scale treatment is desirable. Such lly secured by mining from the entire exposure a sufficient, uniform sh the desired amount. If this would give too great a bulk, miniature de, of fixed size, and at regular intervals; or the tonnage may be further ng blasts at regular intervals, and taking equal quantities from the last. Judgment and care must be exercised in this work, and rules are

Interval-channel, Chip, and Grab Sampling

nnel sampling is done by cutting channels at regular inter- ength of the exposure. To secure an accurate average, the in- small enough, relative to the irregularity in composition and posure, to insure proper representation in the sample of any con- larity. The amount from each cutting must be sufficient to effect of erratic occurrence of rich spots. Thus, the metallics perior copper deposits require mill-runs instead of assays. Usu- even less per ft of channel will suffice. Channels cut in a flat bination into one sample must be equidistant, of equal length n of the exposure), and of uniform depth and width, to yield a ty per ft of channel.

urved or oblique surface, which for calculation must be considered as ther to the dip or to the vertical, the channel lengths are assumed to be of the curved surface upon such normal flat surface, and the depth and must be decreased to yield the uniform quantity per ft of projection, f actual length of channel. Thus, the curved a banded gold vein, sufficiently rich and well- rant such detail calculation, should be meas- in regular way only between *B* and *C* (Fig 4). easured on their projections *AB'* and *C'D*, and d reduced so as to furnish the regular yield per tance (5). It is generally undesirable to make s in the field. Separate samples are usually channel, and if great diversity in composition ent for curvature is necessary, from each frac- equiring such adjustment. Adjustments are f measurements taken at time of sampling. banded masses should cut across the banding. rection of individual channels may bear any rmal to parallel, to that of the line of interval measurement, provided onverge as to emphasize or neglect any particular sampling area.

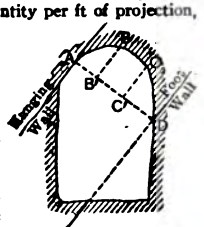

Fig 4. Drift Sampling; Narrow Vein

tting. In soft, comparatively uniform material, as coal or clay, adily cut with a prospector's or a miner's pick. In hard ore, t useless, and in mixtures of hard and soft it is almost impos- own proper proportions of each. Hence, in all but uniformly the channel should be cut with moil and hammer, either single according to hardness of the ground.

may thus be required, depending on whether a single-hand hammerman oil works on a sample sheet, or a sample box has to be held, or a double- used besides holding moil and box. A small pneumatic drill and moil if compressed air is available. In any event, there should be an a

number of hands and eyes, to insure careful, watc
octagonal steel, forged to a diamond point on a t
about 0.5 in from tip is at a larger angle, to incre
18 in long; most of them, 10 to 12 in. For ha
kept ready, to avoid delay caused by using dull
hand, of weight suited to the strength of the
single, to 7 lb for double-hand work and should
the sampler on all-day work, but, if too heavy,

Precautions. After determining where t
be taken that the ore surface at this place is
any efflorescence or decomposition, which
minerals since first exposed.

Dirt or dust can usually be removed by brushi
ing with water, but efflorescence and other altere
solid surface. In firm ground, where there is litt
side the channel, the sample may be caught on a
or large enough when spread on the floor under
including flying chips. When using a sheet, extr
the engineer's knowledge, and enter the sample-
dynamite or candle box, in a gold or a round-corn
bucket with round bottom, or a sack the mouth o
a wire ring. Chips are then prevented from fly
gloved hand over the point being cut. Objection
spill from the cutting point down past the recept
Since a sample sheet will serve to prevent contam
samples may be immediately reduced on it to a
be readily protected from salting (Art 9). Before
mark its delimitations (usually by making soot s
side and for the entire length. To lessen opportu
channels should not be marked until they are acti
may be demanded in cutting samples from expos
drifts and backs of old stopes it may be necessa
wedged like stulls between walls, or, in a wide stop

Sampling shafts. UNTIMBERED SHAFTS
from platforms on light stulls. If the mate
a vertical shaft or winze may be done while
through the center on a windlass rope, and
desired by a light cross pole hinged to the pl
back to the far side of shaft (5). The sam
his position as rigidly as desired by pressing
thus acting against the pole, which points
opposite side. The sampler's hands are thu
he can receive on a sheet spread over his le

To sample CLOSE-LAGGED SHAFTS part of
care is taken to expose but a small area at
lagging, no injury to the shaft will result.

In lagged shafts running down with the bandi
posure is made accessible by cutting out entire p
this can be done by cutting out, sampling back of
time. Shafts in the midst of a mass, as test pits
removing part of the lagging between timber sets
size as the dimensions of the opening admit. If
shaft, channels should be cut on both sides and h
suffice to cut a single channel between sets, locate
channel just above was cut, and so work around

Chip sampling. Some unstratified mas
Ducktown, Tenn, are so hard, tough and c

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difficult. Fortunately these qualities produce a broken surface, es of which are independent of composition, so that chip sampling ory results. The work is like channeling, except that the sample ts of chips moiled off along an assigned line.

ng, the chips must be so broken as to represent equal distances by equal roper proportion of fines and coarse must be secured from each point of e number and size of chips determined in accordance with the irregu- posure.

les are taken by "grabbing" pieces at random. Information haracteristics can thus be gained quickly, and by careful adap- val sampling a method of some precision is developed.

dump surface, for example, may be determined with as much accuracy aking the "grabs" at regular and closely-spaced intervals, securing at roper proportion of coarse and fines.

7. Underground and Placer Sampling

d mineral exposures are usually best sampled by interval-channel- (Art 6). As stated, deposits the values of which are largely in r very rich minerals can not be sampled and assayed in the usual e-scale sampling followed by mill or smelter runs is requisite. st always be checked against previous operating records and a te of value based on careful inspection of ore exposures. In dertaking any sampling, the peculiarities of the deposit should as far as possible by such inspection. For this purpose fresh d be made at frequent intervals, by breaking off pieces with a gh material, with moil and hammer.

ue will be saved, and the engineer's efficiency increased, if this work be od hammermen, at points indicated by the engineer, who is thus left to attention to studying the exposures. The sample interval is usually 5 bly 5 ft, which is short enough to afford means of check by calculating s on alternate samples. Precious-metal deposits should be studied care- eaks and chutes, and sample interval made short enough to include all h or low values. In case of streaks, it is often desirable to divide the e banding into two or more samples, to insure proper proportioning and to exclude (when revealed by assay) low-grade or barren streaks in subsequent mining they could be left untouched (Art 6).

pling must be done in a manner to show much smaller values ered in solid deposits. Large-scale dredging, favorably located, American or Yuba rivers, Cal, gives high percentage of recovery ut 5¢ per cu yd, and in such cases 10¢ ground is profitable when icient to keep a dredge at work for a number of years (Sec 10).

e conditions (properly sloping bedrock, abundant water under sufficient dump room) (Sec 10) hydraulic mining costs even less than dredging. etween 7¢ and 10¢ per ton (0.0035 to 0.005 mg per assay ton, or about assayer can report with any assurance of accuracy), but placer samples he nearest 0.1¢ and averages calculated to nearest 0.01¢. For this the ured and concentrated by rocking and panning before being assayed. s of a practical rocker (11).

mining amount, condition (whether coarse or fine), grade and the gold, other features should be noted at time of sampling. is to be dredged, it is desirable that the gravel (material up to s head) be free from boulders. Strongly-built dredges, with v cu ft capacity, will handle, at some additional operating cost, diam, while 15 and 16-cu ft bucket dredges will handle bould

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weighing over a ton. But, for a low operat
present. Cemented gravel increases cost as
entire equipment, resists washing, and carrie

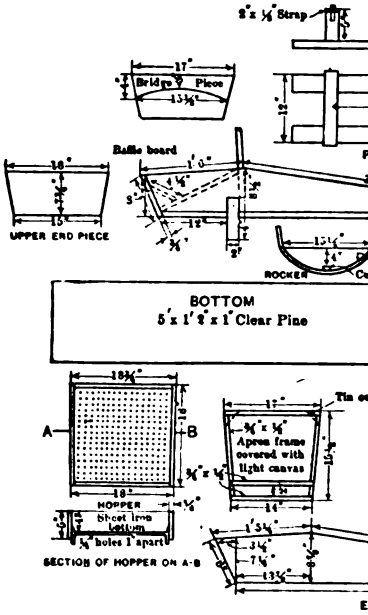

Fig 5. Placer Gold

Maximum dredging depth is limited in low-g
of which not much over 50 ft can be below water
tically long. Hence, without special drainage,
shafts or borings, should not much exceed 50 ft a
and cleaned by dredging. Exceptionally, total d
70 to 75 ft being below and 25 to 30 ft above wa

Minimum depth to bedrock must exceed t
clear the sag of the bucket chain when the ladd
Cost of removing timber, including stumps, mus
small amount of buried timber prohibits dredging

Provision for handling debris is largely a qu
settled to a clear overflow. 50 miner's inches fl
keep the water clean enough for gold saving and
Rapid-running streams in narrow valleys sub
cause of danger to the boat. A hard, rough bed
value would be lost, is equally impracticable for
that what shows in a stream bed as hard rock m
soft, decayed, but sufficiently firm layer, suitable

Methods of sampling placers. The o
placer are those made by freshets and str
probably have been done before the engine
of pits for mining by hand methods, or of sl

c sampled, and note made whether BEDROCK has been reached. drock should always be reached, to inspect and sample the gravel ately upon it. Character, strike, and dip of bedrock can then be omparison with similar observations elsewhere precluding possi- otion by a large boulder. The exposures, if large enough, should annel sampled and each sample panned (Sec 31, Art 5) or "rocked" . After recording, all the "colors" from a single pit are usually final weighing and assay (Art 5). Table 2 is useful for calculat- m fineness (2). The value of a mg of gold from 750 to 775 fine tor often used when weighing colors from placers where the aver- gold has not yet been determined.

Table 2. Fineness and Value of Gold

lue oz	Value per grain	Value per mg	Fine- ness	Value per oz	Value per grain	Value per mg
.36	$0.0409	$0.00063	750	$15.50	$0.0322	$0.00049
.12	0.039	0.00060	725	14.98	0.0312	0.00048
.60	0.038	0.00060	700	14.47	0.0301	0.00046
.08	0.0376	0.00058	675	13.95	0.029	0.00044
.57	0.0366	0.00056	650	13.43	0.028	0.00043
.05	0.0355	0.00054	625	12.91	0.0269	0.00041
.53	0.0344	0.00053	600	12.40	0.0258	0.00039
.02	0.0333	0.00051	.....	.....	.....	.....

ch has been proved by drilling can be CHECK-SAMPLED only by fresh sinking. When the water flow is not enough to prevent shaft sinking, ts on say half a dozen holes of representative location and varying value king method. Measured representative amounts of the material from ing can be rocked, and after the shaft is complete these results may be times as desired by interval-channel sampling of the sides of the shaft. the shaft must reach and thoroughly clean the bedrock, particular care easuring and rocking gravel close to and on bedrock, where most of the occurs. All the gold in a measured amount of gravel in place should be edrock into the sample, but the rich gold-bearing sand surrounding but asured sample must not be allowed to run in and contaminate the sample. of wet ground or other reason, shaft sinking is difficult and costly, and are available, adequate check can be secured by re-drilling holes instead on them. That is, a second hole is bored within a few ft of the original s possible without risk of entering ground disturbed by previous drilling. drill, such as "Keystone No 3," is usually employed (Sec 9, Art 5,7). th 6-in tools and properly used, yields 1 cu yd of gravel from approx 100 n a new outfit must be purchased for boring only a few holes, so that ge item, or when labor is cheap and unskilled, or when transportation is and-power outfit, like the "Empire" (Sec 9, Art 4) is useful. Bedrock ed far enough to determine its character, and that it is genuine and not ulder. The character and amount of ground drilled and the number of ould be noted ft by ft, and the ground-water level recorded (Art 5;).

work be done by pits or drill holes, a close check of individual largely accidental, and whether the results of a certain amount be accepted (though with modification) is a matter of judgment. ust determine if the average of the checking holes is in sufficient agree- erage of the holes or shafts checked, or if the divergence is too great in possible operating profit to permit use (after discount) of the original of combining data are discussed under Conclusions, Art 10. 75 to ecting value is usually considered to be recoverable by dredging, while ually been secured in practice.

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8. Care of Samples, Notes,

Sample tagging. A good method is to
scriptive matter) with soft lead or indelibl
bond paper, like that of ordinary loose-lea
close, to enclose and protect the writing, an
inside the container. This marking may b
tainers, by marking the number with penc
throat just outside the tying, or on the outsi
ing the sample number.

Some prefer thin strips of soft wood, about 0.1 b
ber is written in heavy pencil marks which can
Others use small paper or linen shipping tags, one
tainer; or brass number tags placed inside. The
so that the sample can not become valueless thro

Field notes should show date of samplin
samples of every one concerned in or prese
information regarding the specific day's wo
number and the data regarding it are recor
the sample itself; as, exact location referre
ably a survey station), length and characte
deposit or workings noted at or near the poi

OBVERSE

Mine.....Level.....
Working Place.....

Loc dist from	Section		Sample		Assay value
	Formation	Width	No	Width	

Fig 6. Field Sample Note

Sampling

Mine.....Level.....Wo

Date	Loc dist from	Section		Sample		Assay value	Width by assay	Progre tota	
		Formation	Width	No	Width			Commencing with sample No	Width

Fig 7. Office Record, Sample

In an examination involving a considerable an
systematized by using a tabular form arranged f
form suitable for underground interval-sampling.

. To provide against loss, and for convenience, it is well to copy field
nanent office record. Fig 7 is a form of office record adapted to field
Since field notes are made as samples are taken, samples of work in
scattered over a number of pages, but by keeping a separate sheet in
for each working (preferably in a loose-leaf binder) samples will appear
eir position underground.

tainers. Samples of ores, the moisture or other volatile con-
need not be considered, are best sacked as taken, or immediately
ction as is made at the point of sampling. If, as is often the
exist which might be served by deception, the sacks should im-
ealed or placed under lock and key. The latter can best be
d by using a locked leather mail sack, in which the samples are
of being tampered with even by injection of values in solution
given sufficient time and motive, salting can usually be accom-
en the best protected samples should as little as possible be left
hers than the engineer's confidants.

or other volatile contents is to be determined, samples should be taken
cloth or rubber sheet, and should completely fill the container. They
n screw-top cylindrical galvanized or tin cans, which can be hermetically
g the top down on a rubber gasket, or by binding the joint with several
e tape. Corrosive solids are best carried in glass fruit jars with glass
l gases in glass or earthenware bottles, or demijohns with cork, rubber
according to the corrosive character of the sample.

are the best method of permanently recording sampling results,
e results in relation to each other and to the property as a whole,
ng this data in a report. They are made by tracing a map of
ing the sample positions thereon by dot or line, and writing the
or value of each sample alongside its location (Sec 19, Art 1).

ingle value for each sample, the scale of 1 in = 50 ft is as satisfactory for
r most other work. If several constituents are to be entered for each
1 in = 20 ft is preferable. If several entries per sample are necessary,
ale maps are available (1 in = 50 ft or smaller), the sample number
ten at the point of location, and a table of all data desired placed in a
sheet. This is not so good as the direct entry of the data at the loca-
tion in value in the different workings is less obvious. Variations in
e element, or currency values of several elements, are sometimes indi-
y of shading, variety in coloring, or design symbols, the fluctuations
at a glance. This method has gained favor on the Rand.

ssays, assay maps should show the tonnage calculation blocks,
l average values. For steeply-dipping deposits, these blocks
on longitudinal elevations or on slope plans. For flat deposits,
nstructive.

9. Salting

process of intentionally or unintentionally raising the value of a
at of the exposure sampled, must be continually guarded against.
of intentional salting being attempted must be borne in mind
mstances seem to give little occasion for it.

engineer examining a copper mine, taken on operating lease with privi-
ecommended that the smelter be operated on average ore to be obtained
f uniform thickness from the floor of each level. The mine foreman,
l all the ore for the test run from a particularly rich lot which he knew
or of one of the levels and, in absence of the engineer (who had been
the single examination), the lessees received the impression from the
at workings actually averaging 3% contained between 4 and 5% cop-
n had no connection with the owners of the property, was solely the

employe of the lessees, and had no apparent ob
resident and property owner in the district, an
fraud explained that he feared that if the lessees
have completed the purchase, the property woul
the expected rejuvenation of the district depend
not have been realized.

Since the commonest cause of salting is t
the only safe practice for the engineer is to
ing, even if the vendor's reputation and th
to allay suspicion. Though it is not the ex
alyze the motives of those with whom he deal
his examination as to assure himself that de
him. On the other hand, unnecessary obt
to such assurance is a breach of courtesy, a
examination work, since the sympathetic a
the property is of advantage in promptly
such features as might otherwise be noted o
ably be entirely overlooked.

Salting has been effected by preparing the o
likely to be drawn, or by limiting the exposure
latter is usually done by timbering and close lagg
a cave (or the appearance of one) in barren sect
throughout the mine, as though blasted from ad
will be taken from them rather than the faces.
while it happens to be in good values is also emp
chance that the vendor himself may be deceive
just as large values are about to be encountered.

The commoner method of contaminating
upon the engineer at any point where he
handled by others than himself or his co
employed to cut samples can readily salt
by breaking into the sample an undue prop
which the sample is being cut, or by pick
work. Moreover, very misleading results m

Example. The early history of one of the g
U S was checkered by incorrect conclusions bas
averaging 4% copper, from ore actually containi
by the operating company for its own informatio
pling laid out by a reputable European enginee
left to the casual Mexican miner who happened t
minerlike and wholly innocently, broke down a
showing at the points sampled.

Samples may be salted by syringe injectio
cloth of sack containers. Exposure of sam
affords excellent opportunity for salting by
shaken from the hair, nails, or clothing, or
tobacco ashes or what not. By manipulati
due proportion of barren material may be t
methods which ingenuity can devise for sal
a sealed sack, are innumerable. The only
constantly both sampling and assaying, and
out of the hands of all except his confident
be barren, and purposely carried for assay
the salter's values if the lot is tampered w
placed on his guard. (See also Sec 30, Art

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10. Sampling Checks and Conclusions

accuracy of both sampling and assaying should be made in suffi- to give assurance that results are as reliable as circumstances e desirable. When possible, ERRATIC SAMPLES should be retaken precaution against inaccuracy or contamination. When assays de prior to leaving the property, it is well to duplicate 5 or 10% g, selecting the samples for duplication at random, but so as to which appear to be of high, low or average value.

uld be checked by sending for assay a small percentage of duplicates last reduction in sample preparation), and a few blanks of known barren w numbering. Also, a few duplicates should be sent to a different assayer. lways agree in detail, but the average of the duplicates by either assayer from that of their originals by more than a small percentage of the values vided the assaying is accurate, the DUPLICATE SAMPLING should check in r; that is, actual duplicates may vary considerably, but the average of riginals should show reasonable agreement. If sampling has been done rval, compared to the total length of exposure (say 5 or 10 ft on large ge of alternate samples should closely agree with that of the intervening ater the number of samples, the closer should averages agree.

from results of sampling should be based upon propor- s; that is, the quantity of material represented by an assay ich a factor in calculating an average as is the assay itself. Thus, d vein, exposed over its full width and sampled at regular inter- y is multiplied by the sample width, and the sum of the prod- the sum of the widths to determine the true average. If sample regular, the width is first multiplied by the interval to the suc- (using the last interval on both the last and next to last sample) ict multiplied by the assay. The sum of the interval-width- divided by the sum of the interval-width products then gives d average.

exposures containing large and irregular proportions of heavy minerals (as galena in quartz), a SPECIFIC-GRAVITY DETERMINATION is sometimes mple, and the interval-width product multiplied by it before multiplying he sum of the interval-width-gravity-assay products divided by the sum dth-gravity products then gives an average assay, which includes allow- ific gravities of different parts of the ore exposure. This refinement is are rare in which the engineer's knowledge of the mining value of the sufficiently increased by gravity determinations to justify the added he value usually follows the heavy mineral, the probable effect is to assay. This, unless subsequently discounted, seems more likely to decrease error in determining the mining value. Owing in part to the re of the valuable minerals, and particularly to the inevitable inclusion, some of adjacent low-grade or barren material (which can not properly sample), the yield per ton of a carefully sampled deposit is generally alculated average, and the tonnage higher. An allowance (say 10%) e made for this in reporting values.

sometimes taken against over-valuation is the ELIMINATION ATIC VALUES, substituting for them the average of the two ad- but this practice is questionable. If there are enough erratic the result seriously, there may also be enough high values to n of all. The proper disposition of erratic samples calls for each case, and for the engineer's best judgment.

it is wider than the exposure in drift, upraise or stope, and samples of ve been taken at regular intervals, the width and interval factors di- athematical average of the assays applies to the area sampled. B

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unless the workings sampled are uniformly distri
matical average should be applied only to the imm
area assays should then be combined with their
average of the whole is found by dividing the sum
of the areas. This avoids the common error of
samples secured chiefly from a small, well-devel
but little-developed area.

Calculation of average values is facili

Sample No	Width			Gold			Silver			
	of sample	Prog total		Assay	Width by assay		Assay	Width by assay		Assay
		Inc sample No	Inclusive total		of sample	Prog total		of sample	Prog total	

Fig 8. Form for Calculatin

(Blank columns for unexp

From sample numbers, assays, measurement
averages can be recorded, and final summati
which to secure proportioned averages of dif

A pocket adding machine (as the "Golden Ge
field calculations. More elaborate adding and c
which is especially for addition and multiplication

In drawing conclusions from results of sa
that the development of a deposit, or even a
that an accurate, mathematical description
of ore can not be accurately known until it i
must always be made for contingencies.

A single block of ore may be sufficiently devel
calculation closely approaching those ultimately
engineer should scrutinize the data of each block,
is of this thorough nature, or whether allowanc
knowledge of the characteristics of the deposit.
in determining his declaration of assured values
are based primarily upon his sampling and calc
important data bearing upon the conclusions rea

11. Determination of Quantity and

Assured mineral is the amount existing
nite assurance. It is necessarily a minimum
in case of mineral well blocked out relative
deposit, there may be masses of proved mi
calculated. The term ORE IN SIGHT has bee
variously used and misused that it has cease
ing. It has been suggested that ore expo
"in sight;" but, in irregular deposits ore s

lue, while in regular deposits the ore may be assured, although 3 sides. Hence, the use of the term is being abandoned by to convey a definite idea. An engineer's statement of assured tially a conclusion based upon judgment, not merely the result calculation.

calculation, though simple, must be carefully applied in each e relation of amount and reliability of data to extent and im- clusion may be fully realized. Assay maps (Art 8) best show ecially when studied in three dimensions (plan and elevations), the prime sources of information as to size, shape, value and he different ore exposures. Structural geology of the deposit died, so that isolated exposures will not be considered consecu- ing barren volumes calculated as of value (Art 2). The aver- of the exposures of a given block being determined, volumes y multiplication of average area by average thickness, by in- etween exposures, or by depth between levels, according as flat-lying seam, bed or zone of enrichment, in an inclined vein, n a deposit of vertical extent or development (Art 10).

. strong vein, of fairly evenly distributed values showing no tendency shoots, the value areas, measured normally between walls, and exposed els close enough together to preclude unexpected irregularities, are lip distance between levels to determine the volume of mineral, and xposures, combined proportionately to their respective areas, may be of the block.

developed by well-defined and sampled cross-sections, the accuracy of increased by using the prismoidal formula. A mid-section is con- g adjoining sections, connecting their definitive points, and forming ing the mid-points of the connection lines, as in railroad earthwork rt 19). Masses well developed by drilling cross-sections are advan- ted, the volume being found by multiplying ⅙ of the distance between by the sum of the areas of these sections plus 4 times the area of the

elopment and sampling of a deposit is so uniform that the d number of samples of each block are practically the same, l be as small as the data will permit, so that a minimum will be grouped into an arithmetical average before being he volume factor which they represent (Art 10). General obtained by including the volume factors.

Fig 8, by substituting "tons" for "feet," facilitates determination ge values of any desired combination of ore blocks. In laying out n, care must be taken that all available data are included, and that suffice to assure the presence of the full calculated volume and value; n, what proportion is assured. Possible influence on each block of inches, horses, low-grade spots, chutes, rich masses or other inclu- ecided only after careful investigation of the development of the and similar deposits. Minable ore of less than stoping width must the calculation by enough barren rock to make workable width, or mining of the necessary waste included in the cost estimate. The dgment should be brought to bear on each block.

actice is regrettable, flat-lying deposits (especially placers) by irregularly placed pits or drill holes, so that the mathe- f values shown gives an erroneous idea of the average value en though the depth factors are included. As the tendency eatest number of pits or drill holes in the richest spots, the ys exaggerates the true value.

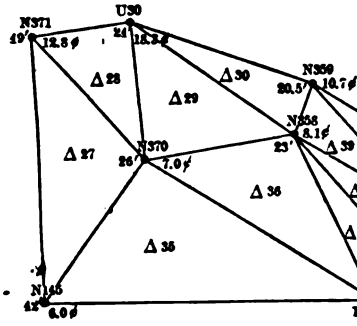

Fig 9. Example of Irregul

To minimize this error, divide the area of valu
possible, having their angles at the pits or holes
hole by centering an unusual number of triangles

Tri-angle No	Corner holes					
	Hole No	Depth			Value, ¢ per cu yd	Prod-uct, value by depth
		Ft	Yd	Aver 3 cor holes		
28	N 371	49	16.33	....	12.8	209.02
....	N 30	24	8.00	....	18.3	146.40
....	N 370	26	8.66	....	7.0	60.62
Tot &	Aver	....	32.99	10.99	....	416.04
29	N 370	26	8.66	..:.	7.0	60.26

Fig 10. Sample Holes and Values

Tri-angle No	Scaled dimen, ft		Base, yd	Half alt, yd	Base b half alt = area, sq yd
	Base	Alt			
28	410	210	136.66	35.00	4 783.1

Fig 11. Volumes

Besides natural variations from appar
occur. Abandoned drifts, crosscuts, up
by timbering or muck, intentionally or b
common in mines which have passed thr
found may reveal unlooked-for conditions

A case is on record in which a rich ore-block, d
of levels above and below and in upraises at

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winze in the floor of the upper level, which had been timbered over and tram-road muck (4).

ecision hangs on the presence *in toto* of a few well-developed reat care must be taken to insure that neither natural peculiar- s intentions deceive; possibilities which must always be recog- rded against (Art 10). But deception of this kind can rarely sufficient scale materially to affect the conclusions of a report.

wn over the property, the engineer should put directly to the executive s opportunity offers, to each subordinate independently) the question neer may be assured that he has seen every known opening or exposure, such exposures are incorrectly shown on, or entirely omitted from, the Unintentional oversight will usually be thus discovered, and, if in- s being practiced, some one reply may be sufficiently evasive or con- se suspicion and lead to discovery.

ctors. The factor by which the volume in cu ft should be ce it to tons is determined by dividing 2 000 lb (or 2 240 lb for 62.5 lb (wt per cu ft of water) × sp gr of the mineral mass (Sec 1, Sec 28, Table 1).

sp gr take the average of a number of sp gr bottle-tests on average Fill a marked narrow-neck bottle to the mark with distilled water at *a*). Introduce a weighed amount (*b*) of sample, restore the water level insuring removal of all air accompanying the sample (done by boiling, ently insoluble) and take final weight (*c*). Then, sp gr of the mineral - *c*]. (See also Sec 1, Art 3.)

ing average ore composition, the average tonnage factor is Table 3 (1).

Table 3. Weights of Minerals and Rocks

Mineral	Wt, lb per cu ft	Cu ft per ton (2 000 lb)	Metal	Mineral	Wt, lb per cu ft	Cu ft per ton (2 000 lb)
ive.....	417.8	4.8	Copper...	Tetrahedrite	305.5	6.5
nite.....	286.8	7.0		Atacamite....	236.9	8.4
iment....	361.6	5.5		Cuprite.....	374.1	5.3
lgar.....	218.2	9.2		Chalcanthite.	137.2	14.6
ite	280.5	7.1		Malachite....	243.1	8.2
herite....	268.1	7.4		Azurite.....	230.7	8.7
bon.....	93.5	21.4		Chrysocolla..	137.2	14.6
cite.....	168.4	11.9		Dioptase.....	205.7	9.1
gonite....	187.1	10.7	Gold.....	Native.....	1184.7	1.7
psum.....	143.4	13.9	Iron.....	Pyrite.....	318.0	6.3
orspar....	199.4	10.0		Marcasite....	299.3	6.7
tite.....	199.4	10.0		Pyrrhotite...	286.8	7.0
hracite...	93.5	21.4		Arsenopyrite.	374.1	5.3
uminous..	81.0	24.6		Hematite....	311.8	6.4
naite.....	305.5	6.5		Menaccanite..	299.3	6.7
altite.....	424.0	4.7		Magnetite....	311.8	6.4
paltite.....	386.6	5.2		Limonite.....	236.9	8.4
thrite....	187.1	10.7		Siderite.....	236.9	8.4
ive.....	554.9	3.6	Lead.....	Galena.....	455.1	4.4
lcocite....	355.4	5.6		Cerussite.....	405.3	4.9
lcopyrite.	262.0	7.6		Anglesite.....	399.0	5.0
site.....	311.8	6.4		Crocoite.....	374.1	5.3
rgite.....	274.4	7.3		Pyromorphite	436.4	4.6

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Table 3 (*Continued*). — Weights

Metal	Mineral	Wt, lb per cu ft	Cu ft per ton (2 000 lb)	
Manganese	Pyrolusite	299.3	6.7	Ti
	Psilomelane	262.0	7.6	
	Wad	218.2	9.1	Tu
	Rhodochrosite	224.4	8.9	
	Rhodonite	224.4	8.9	Zi
Mercury	Native	897.9	2.2	
	Cinnabar	561.1	3.5	
Molyb-denum	Molybdenite	286.7	7.0	
Nickel	Millerite	349.2	5.7	
	Niccolite	467.6	4.3	
Platinum	Native	1091.2	1.8	
Silver	Native	174.6	11.5	
	Argentite	455.1	4.4	
	Hessite	530.0	3.8	
	Petzite	561.1	3.6	
	Sylvanite	498.8	4.0	
	Pyrargyrite	361.6	5.5	
	Stephanite	392.8	5.1	
	Proustite	342.9	5.8	
	Cerargyrite	336.7	6.0	
Sulphur	Native	130.9	15.3	

Note. — For compositions and specific graviti

Example. A ton of ore in place, composed of 23% schist, will occupy a volume = (70% of 8.2 cu ft; or, in practice, call tonnage factor 9 c

As in the example, it is advisable to allo tonnage factor than calculated, regardless mineral masses, 1, 2 or more cu ft may hav factor. In important cases, where uncert rately measured excavation should be take

Roughly, cu ft of anthracite coal may be re bituminous coal to short tons by dividing by 25; by 13 (or by factors down to 10, if sufficiently su by 10, 9, 8, or 7, according as they consist chi Fe_2O_4, or PbS, or contain more or less gangue, or than 5 cu ft of pure PbS make 1 ton. In dump to 1.9 times the volume of same material in pl and values, as of individual exposures, are best n

12. Prospective Values, Percentage o

Prospective quantities and values. trained judgment than, and is of as much vi part of an engineer's report. Having coll yield and present showing of values, geologi occur and records of similar conditions and careful sifting, balancing, and digestion of form as definite an opinion of the prospe opinion in practical form, the engineer may questions: 1. What are the geological ar change of formation, well-determined fau

itions), and to what extent do these preclude possibility or prob-
er development of values in depth or horizontal extent? 2. Are
ly probable continuations or indications of independent values
y? If so, what is the least development and cost which will
nformation regarding such values, and what development and
ry to prove thoroughly these values, or prove them sufficiently
profitable industry thereon? 3. In addition to highly probable
opes may reasonably be entertained as to extent and value of
ineral resources of the property, what scale of outlay is likely
and justified, and what, if any, definite limitations do local
e on such hopes? In general, deposits not definitely limited
ntain at least 25 to 100% more ore than can be actually assured.

(Question 1) A deposit was limited at one end by a property line across
dy passed to an adjoining mine, while the structural geology showed
main workings bottomed in and were limited by a synclinal trough.
mparatively small outlying developments on what could be traced as
an anticlinal fold succeeding the syncline, exposed a small but excellent
lower level than the bottom of the syncline. The property being in
and equipped for diamond drilling, $1 000 spent in boring in the floor
e was likely to add many times this amount to the value of assured ore
on 3) The drilling was also likely to trace ore to sufficient depth to
winze, so as to render a new level acccessible to drilling. The winze,
at the lower level, would cost $5 000. Granting the first favorable dia-
ng, the general features of the deposit and the reputation of the district
pe that enough ore could be proved and indicated to justify opening this
direct from the surface (replacing the long crosscuts and hoisting winzes
revious development had been made) by an efficient main shaft to cost
This shaft in operation, a low-grade outlying deposit on the strike
as hope, because of the 25 to 50% increase in value with the depth of
developed) could readily be proved and exploited in depth. This
e proved valuable, in addition to the main deposit, the mine would have
eat future, notwithstanding the limitations of property line and fold
nce the mine was equipped and in profitable operation, its purchase at
ue of all assured ore, figured on the highest likely average market, was
se skilled in the business and looking for promising mining ventures,
is question whether a cash purchase was not advisable at a much higher
, all expectations were so largely fulfilled that the true value proved
slightly forecasted by the value of the assured reserves.

(Question 1) An obviously secondary enrichment, lying between a
nd underlying limestone, was found exposed by an open-cut working
concentrating mill, the operation of which investigation proved could
. Test pits to bedrock at different points about the open cut showed
on 3 sides and proved considerable value. At no point on the prop-
cial values found in the underlying limestone. Property lines were
tances in all directions from the open cut. Post-hole-digger holes to
ds of which in many cases showed ore, were found scattered about the
any of them were reported barren, it was doubtful whether the deposit
considerable area of the property. (Question 2) Though the records
area proved by test pits showed the values to be exaggerated, continual
ds of barren holes suggested that the exaggeration was due rather to
than to intention, and that the presence of ore could at least be pre-
ing a record of ore value. Therefore, systematic prospecting could be
siderably to the values already assured. The facility with which both
been sunk through clay and ore to the limestone proved the practi-
ploration, and it was obvious that for $1 500 the entire property could
lowing simple manner. The area was laid out in 500-ft squares and a
corner; then holes were bored at corners of 100-ft squares laid out
howing ore indications. The 100-ft square system was then extended
developed were completely surrounded by barren holes on this 100-ft

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system. Ore areas thus defined were finally p
squares. (Question 3) Though expectation of
those actually assured at time of examination w
showed that such increase would be moderate,
must be considered as forming a large proportio
far as the secondary enrichment was concerned.
lacking in the exposures, they received little con
value. The property was therefore thoroughl
values, as described. The above inferences bein
equipment and scale of operations were propos
secondary-enrichment values, and the highly pro
not overburdened with necessity for returning la

Percentage of gross value recoverab mates of recovery and costs are based on the property; on results of investigations ore-testing works; on the operation of a s erected for the purpose; and on practice elsewhere. ACTUAL ACCOMPLISHMENT on t by investigating the records, and checkin tailing-dump samples and, if feasible, again Equipment should be carefully inspected tion, and whether general appearances, co that the management could be improved. assist in suggesting possibilities of treatm WORKS treatment of larger lots may be of st ROASTING AND LEACHING TESTS in the lab treatment other than by concentration. O ing heavy capital outlay for equipment befo customary to erect a PRELIMINARY OPERATIN times hundreds of tons per day) for ore tr working conditions. Such a mill is operated though at a loss, simply to secure data and ment, preliminary to designing a large eq design is as flexible as possible, so as to i treatment which preliminary tests in labora ACTUAL PRACTICE in mining and treating s factor in estimating operating costs and pe

Conclusions drawn from practice must be b conditions, and of the mineral to be treated. R scale of operations at successful properties, as co question, should be thoroughly understood. Sca ing costs. Thus, concentration costs in the gre each treating thousands of tons per day, are ab ment in a 100-ton mill might cost 80¢ or even $1 ant estimates of percentage of gross value reco made only after analyzing all information attaina ance must be made for general overhead charges, ket and contingencies, the last usually being put

13. Market Price of

Theories and difficulties. Gold, bei of $20.6718 + per troy oz. Fluctuations products form one of the most uncertain essentially a commercial question, busine in the product in question can often est gineer the average price it will command

ngineer himself is often forced to choose the standard of price f his estimates. Current prices may be adopted, but this may npressions, resulting in conclusions that become absurd before the veloped. Hence, the engineer should consider at least one other ably several others. In case of metals or coals having a standard estionable whether an engineer should recommend purchase of a n he estimates is incapable of making a small profit, when mining erves and selling its product at the MINIMUM PRICE of say 25 years ue of a property which would fail to make an attractive profit under on, at the AVERAGE PRICE for a like period, is even more doubtful.

ineral products having no standardized open market value, especially s depend on empirical gradings as to color, texture, etc, there should be n in acquiring a property which, though meeting other requirements, an output comprising enough of the highest grade product to insure a of output of other grades. This is due to the uncertain market for low dardized products, both as to price and to ability to absorb at any price. a mine is in producing the highest grade, the lower grades being pro- largely as unavoidable by-products.

n of SIZE OF MARKET, available within a radius of feasible freight nportant, since the opening of a large new property in a restricted may demoralize existing prices. Facilities for sale of product is consideration. Marketing a peculiar product may constitute f cost; sometimes the market can be secured only through chan- on practical surrender of control of the property itself.

in touch with sources of more detailed information, Table 4 has been from the U S government reports. Prices for 1914, though extant at on, are omitted because they are generally abnormal.

eral Products of the United States. (Yearly Quantities, Prices, and Averages. Imports or Exports given, if informing)

Metals	Highest price			Lowest price			1913		Aver, 25 yr, 1889–1913	
millions of pounds short tons	Year	Quantity	Price	Year	Quantity	Price	Quantity	Price	Quantity	Price
mil lb (except '89 in										
.....	89	47	205	11	46	18	72	19	15	22
mports since '91, mil										
.....	06	8	18	97	1	4.1	15	5	6	7.5
ad, since '06, thous										
.....	07	10	134	08	14	93	17	100	14	98
mill lb & ¢.....	07	87	20	94	36	9	122	15	68	14
oz & $	10	5	21	97	3	21	4	21	4	21
¢	06	700	5.7	96	376	2.8	873	4.4	541	4.3
s since '93, mil lb & ¢	09	12	34	94	1	20	29	33	10	31
ons & $.....	02	18	21	94	7	10	30	15	16	16
orts since '94, thous										
.....	13	118	42	99	188	8	118	42	105	22
ous flasks (75 lb) & $	90	23	52	94	31	30	20	40	28	41
oz & ¢	90	55	105	09	55	52	67	60	66	66
ail lbs & ¢.....	13	104	45	96	45	13	104	45	72	29
¢.....	12	648	6.9	95	179	3.5	675	5.6	316	5.3
—metals										
t & ¢.....	03	283	124	98	167	80	479	118	250	107
il long t & ¢.....	13	82	239	98	48	158	82	239	59	196
bbl (42 gal) & ¢....	00	64	119	92	51	51	248	96	104	77

Table 4 (*Continued*). — Mineral Pr

Structural Materials mil lb = millions of pounds sh t = short tons	Year	Hig pr Quantity
Cement, mil bbl & ¢	06	
Glass sand, since '01, tens thous sh t & ¢	06	1
Gypsum, tens thous sh t & ¢	03	1
Lime, since '03, mil sh t & ¢	07	
Sand, etc, since '04, mil sh t & ¢	05	
Abrasives		
Corundum & emery, tens sh t & $	92	1
Garnet, since '95, tens sh t & $	00	
Pumice, since '01, thous sh t & ¢	07	
Chemicals		
Arsenious oxide, since '00, tens thous lb & ¢	07	
Bismuth, imports, thous lb & ¢	05	
Borax, thous sh t & $	93	
Bromine, thous lb & ¢	89	
Calcium chl'de, since '09, thous sh t & ¢	10	
Fluorspar, thous sh t & ¢	98	
Marls, '89–'08, thous sh t & ¢	01	
Phosphate rock, tens thous long t & ¢	90	
Pyrite, thous long t & ¢	01	
Salt, mil bbl (280 lb net) & ¢	90	
Sulphur, since '89, thous long t & $	94	
Sulphuric acid, since '10, thous sh t & ¢	12	
Pigments		
Barytes, thous sh t & ¢	89	
Cobalt oxide, imports, since '89, thous lb & ¢	11	
Mineral paints, '89–'11, thous sh t & $	07	
Miscellaneous		
Asbestos, tens sh t & $	92	
Asphalt, thous sh t & $	07	
Bauxite, thous long t & ¢	09	
Chromic iron ore, long t & $	08	
Feldspar thous sh t & ¢	99	
Fuller's earth, since '94 thous sh t & ¢	13	
Graphite, since '95, tens thous lb & ¢	96	
Magnesite, since '92, tens sh t & $	98	
Manganiferous ore, since '05, thous long t & ¢	09	
Manganese ore, thous long t & $	09	
Mica, since '95 tens thous lb & ¢	96	
Mineral waters, mil gal sold & ¢	98	
Potash salts, imports since 1911, mil lb & ¢ (not including kainite & manure salts)	13	
Silica (quartz, flint) thous sh t & ¢	94	
Talc, fibrous, thous sh t & $	92	
Talc & soapstone, thous sh t & $	02	
Tungsten ore, since '99, tens sh t & $	07	

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ver price of copper for 25 yr ending 1913 was 14¢ per lb, on annual aver
000 lb. Minimum price (1894), was 9¢, on output of 361 000 000 lb.
15¢, on output of 1 224 000 000 lb. In view of these facts, an engineer
recommend purchase of an independently operated mine, producing
ss it were likely to maintain an output from best ore reserves for at
ss cost of less than 9¢ per lb. Such a property would be undesirable
ke good profit while mining its average ore and selling in a 14¢ market.
t that as recently as 1907 (13 yr after the 9¢ minimum) the aver price
tput of 869 000 000 lb, should be remembered in considering the possi-
r mine.

Aver price of barytes for 25 yr ending 1913 was $3.20 per ton, on aver
41 000 tons. Minimum price (1897) was $2.24, on output of 26 000
1913 was $3.45, on output of 45 000 tons. An engineer should therefore
mend purchase of a barytes mine unable to produce for at least 1 year,
ss than $2.24, or which would fail to make good profit at $3.20 per ton.
all production of barytes, of the physical and chemical peculiarities re-
de in this mineral, and of the high relative transport cost, due to its
n, the engineer should also hesitate to make recommendation before
character and location of available markets. He would probably find
es, saleable in quantity only in particular parts of the country and
stablished channels, yielded more than the aver price, all other grades
y at figures low enough to reduce the aver prices to those given. The
hest price, $4.96, occurred in 1889, at beginning of the 25-yr period,
ly declining market; but, the lowest price was in 1897, rising in 1913 to
it of 45 000 tons, as compared to the 25-yr aver of $3.20, on output of

g Scale and Profits, Capital Requirements, Amortization

eration is determined, on the one hand, by the minimum rate
roperty can be worked at a low enough cost per unit to insure
he other hand, the maximum scale of operation is limited by the
and probable ultimate mineral resources from the exploitation
the capital, interest, and ultimate profit must be secured; also
vailable for development and equipment, and unless the product
ed market, as that of gold, by the available market. These
termined as far as practicable, and estimates made of operating
oposed scale of operations, the profit per unit is estimated by
cost of production from the assumed market price or prices
The profit per unit, multiplied by the proposed annual produc-
sual basis of 300 days actual operation), gives the estimated
NG PROFIT (Sec 21).

irements. Estimates should first include the probable cost
he property sufficiently to give a safe amount of assured mineral
is, there should be ASSURED MINERAL RESERVE at least sufficient
its that will more than return the capital required to put the
uccessful operation. Assured reserve should be so large that
the proposed rate can not exhaust it in a reasonable maximum
ecessary to prove and block out additional reserves.

d reserves being secured, there should be ample capital to develop them
h will insure regular production at the desired output. Capital must
or EQUIPMENT necessary for proper handling of output to the point of
ial, concentrate, metallurgical product, or whatever form conditions
plant, smeltery, employes' houses, etc, must be considered, and capital
when necessary (Art 2). There must be sufficient OPERATING CAPITAL
general and commissary supplies, mineral in process of treatment, and
or to sale and settlement. In absence of more exact data, 3 months'
y be allowed for these purposes. In estimating total capital requ-

ments, the cost of securing the capital (underwriters' incidental financial expenses) should be included. U an ownership in touch with strong financial interests, may not be more than 5%.

Amortization of capital is its return with i exhaustion or "death" of the property. If th operating profits must include an ultimate profi

Amortization of equipment is usually effe depreciation or write-off on the equipment ca charged to operating expense. 10% of the gro ordinary mine, mill, or smelter plant. If this c accumulating at 4% compound interest, funds 9 yr for complete replacement of equipment.

Amortization of entire property is usuall dividends, without discrimination between an mate profits. It is then the stockholders' resp proportion of the dividends to amortize the i the engineer must make allowance for amortiz to the value of the property. In less usual case have been determined and the price of product tuate, amortization can be calculated by the us

Table 5. Amortizat

Annual dividend rate	Number of years life required to yie nish annual instalments which, if original investment at end of perio		
%	5%	6%	7%
6	41.0	.....	.....
7	28.0	41.0	.....
8	21.6	28.0	41.0
9	17.7	21.6	28.0
10	15.0	17.7	21.6
11	13.0	15.0	17.7
12	11.5	13.0	15.0
13	10.3	11.5	13.0
14	9.4	10.3	11.5
15	8.6	9.4	10.3
16	7.9	8.6	9.4
17	7.3	7.9	8.6
18	6.8	7.3	7.9
19	6.4	6.8	7.3
20	6.0	6.4	6.8
21	5.7	6.0	6.4
22	5.4	5.7	6.0
23	5.1	5.4	5.7
24	4.9	5.1	5.4
25	4.7	4.9	5.1
26	4.5	4.7	4.9
27	4.3	4.5	4.7
28	4.1	4.3	4.5
29	3.9	4.1	4.3
30	3.8	3.9	4.1

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A thoroughly proved gold placer deposit in successful operation by te of 1 000 000 cu yd per year, at cost of 6¢ per cu yd, contained 10 000 000 ing 13¢ per cu yd recovery value. Annual operating profit was therefore he life 10 yr. Table 5 shows that an income having a 10-yr life will yield esides a sufficient amortization charge, accumulated at 4%), when the in- ich a sum that the annual dividend is between 13 and 14% thereon (in posite to "10.3 to 9.4 yr" in column 2, will be found "13 to 14%)." To turn the dividend rate must be between 18 and 19%. Dividing $70 000 by d 19 respectively, shows that the property will yield 5% net, besides amor- ween $500 000 and $538 000 is paid for it, or about 10%, at a cost between $389 000.

e of assured mineral reserves, the whole question of value is one of d amortization tables have little application. But, when part of 's value lies in assured mineral, it is often instructive to carry out ation idea as far as possible on the assured reserves; and then rs of work, tons of ore, etc, required beyond the reserves to return n placed upon the property with appropriate profit.

A copper mine, having assured reserves of 750 000 tons ore yielding, at per ton profit, is operating at rate of 150 000 ton per annum, and its secu- pen market at a price aggregating $2 000 000. At the annual profit of yield is 15%. Assured life, 5 yr. Table 5 shows that a 15% dividend 3.6 yr to amortize the principal, if 5% per annum return is to be had, or sk is such that a net 10% return is required. The $2 000 000 market valua- roperty therefore assumes a life beyond the assured reserves of between a sufficient rise in the aver price of copper above 14¢, or a combination of e additional ore required, independent of change in price of copper, would 2 250 000 tons. At an aver ore yield of 2 000 tons per ft of depth, these s would involve the continuation of the deposit with average strength for h of 270 and 1 125 ft, respectively.

on of Mines Having No Assured Mineral, or in which the Assured Mineral is of Unproved Value

nines of this kind is simply an attempt to discount the future, and almost entirely a matter of personal judgment. A great variety nd conclusion, according to temperament and viewpoint of the st be expected. But the engineer should not make this an excuse give his client the benefit of his best judgment on this vital ques- tements in Art 12 as to prospective values and quantities apply qual force. All possible information should be secured regarding in question, neighboring properties and their relative location, the neral, and results secured elsewhere from apparently similar prop- strength and character of outcrop; extent of, and values shown ound development and drill records or cores; apparent tendency pinch or swell, to become enriched or impoverished, or to assume nore complex character with depth or with horizontal extent; like- ling a feasible treatment process, and the total values per ton in mass, whether known to be recoverable or not, should all have ching an opinion.

ogical conditions should be considered, as to whether they are favorable to of quantity and value, or the reverse; and also respecting their similarity cessful properties elsewhere. As to geographical location, see Art 2. ints having been studied and digested, the engineer should allow himself d by his general opinion, or his instinctive feeling in the matter. Analysis this opinion should be worked out and recorded, to enable the engineer to fend his position to others and steady his own resolve during periods of certainty in subsequent developments. But an instinctive sense of con- developed by exhaustive study and restrained by realization of its li

tions, is of great value to the client; it should be cultiv
strength rather than suppressed as a weakness.

Example I. An enrichment of a highly acid rock
was exposed by underground workings, near the cent
about 500 000 tons of ore. Property included an are
about 0.75 mile wide by 1.5 mile long, generally iron
of value similar to that of the proved ore was revealed
of a few hundred ft depth or less had been sunk. Bott
and the main shaft sunk below it showed distinct tende
and the workings as a whole showed an enriched zone
about the same thickness of leached overburden. T
dike was noted in one side of the underground working
dike in connection with showings of value elsewhere on
be found, discouraged the theory that the dike was the
indications pointed to the existence of a secondary en
at varying depths beneath the entire porphyry area.
of the values quite near the surface suggested cheap
from the surface, and the rock being both hard and ba
promised to be best. Judging by the thickness and
the exposed enrichment zone, probably millions of ton
But, the assured ore was too poor for profitable oper
and none of the outlying shafts promised that further d
richer ore. The further objection had to be met that
richer deposits of same character doubted the possibili
grade. Altogether the property could be given no
been achieved elsewhere on even poorer ore (though
ally) by large-scale work employing extensive equipm
promise of vast tonnage provable by drilling, at a fracti
an attractive feature. If successful treatment could
justification for venturing the comparatively small am
the tonnage required to justify capitalization for large
consideration with another, the engineer was finally co
certainties, the chance of large profit was sufficiently g
attractive venture. The client was so advised, but t
mately be required was not available. The engineer
devote available funds to investigating processes, to p
tonnage development by drilling, to meeting the con
of the property, to interesting heavy financial interests,
which would give such independence of position as wo
final negotiations for the transfer of the property to stro
8 years elapsed between the forming of a small compar
of regular dividends. But in half that time the client
vestment from sale of part of the securities received o
financial interests, and, the property becoming a mark
were eventually valued at a much greater amount.

Example II. A gold placer property on a small stre
the stream by property lines, while outcrops and drill
the bedrock cut off the gravel on each side. Records
experienced placer prospector, showed yardage and va
ing profit of several times the cost of a dredging
engineer was responsible for the drilling, the records
there were too few holes, and they had been placed too
if correctly drilled and recorded. Value calculations
conclusive, but indicative only. The likelihood that t
ing was emphasized, however, by the fact that the v
dredged, while drift-mining had been done above. As
a leading builder for a well-designed dredge, which
direct and cheapest mode of checking the drill records
operations. Meanwhile the property had no proved
enterprise could scarcely be considered a good gambl
The client (owner of a minority interest) was therefo

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only on the basis of a very limited valuation of the property, and then speculation and in case satisfactory assurance could be given of proper perating of the company. These conditions could not be fulfilled and, as d advised was probable, a receivership and ultimate sale of the property avoid losing his original investment the client bid in the property at a cost of the dredge. Sufficient value was ultimately proved to cover the lus a reasonable profit, but only a trifle of the original hopes of the enter- ed, and only those willing to contribute to the purchase at the receiver's al loss.

Conduct of Examinations and Outfit Required

ies. All information furnished by the client (which may range tatement respecting location and supposed value of the property, d maps by skilled engineers) is reviewed, to gain an idea of the aracter of the examination, and the time and facilities required to e extent of development of the property, especially that of the ex- eral values, largely determines the time and equipment necessary and the importance of sampling results in reaching a valuation.

ation and accessibility from the engineer's headquarters depend the time uired to reach and return from the property. Technical periodicals and d books on economic geology, sometimes yield useful information, through ecial mining indices. The relations of the client to the property are usu- ee access is taken for granted; if not, the limitations must be understood f the promised report restricted accordingly. If any considerable outlay ain access to underground workings, the question of providing for this nderstood.

nination, regardless of time or money, would be one made wholly eer personally, who would thereby become thoroughly familiar tions affecting the problem. But reports must be made within a ne, and are generally needed as quickly as possible. Moreover, engineer is too expensive a man to devote time to unimportant ome assistance is always required, if only for clearing out old case of a large, operating property, the whole mineral exposure t be sampled, and its records and operations reviewed, a corps necessary for preparing a reliable report with reasonable outlay of ey.

must be selected with great care. Their integrity must be be- n, the capacities of each well understood, and none should be sponsibility that he is not well able to carry.

hould be given work that is new to him only under supervision of an ex- ior. Bright young technical graduates make excellent raw material to g, surveying, drafting, and similar work. But an inexperienced graduate sent into the field alone, except to gather preliminary information, his checked by an experienced man before any important decision is reached.

As this is usually the most tedious part of the examination it run promptly; though not until enough information has been ably by inspecting the mineral exposures) to determine if the gh sampling is justified.

ained assistant is required, he usually does the sampling, thus allowing devote his time to more general investigations. For extensive sampling ay himself require trained assistance, or at least unskilled help. The y obtainable at or near the property, though it is sometimes desirable to the outside to lessen possibility of salting (Art 9). But the chief sampler y retake enough samples to detect fraud, thus making it feasible to em-

Expert accountants sometimes assis
analysis of the accounts, though the e
is generally sufficient for him to understa
explanations, who can, when so checked,

Legal assistance is sometimes require
situations arising in connection with title
planations of the owner's legal adviser a
to make unnecessary the expense of ind
unable to grasp the full purport of any im
petent attorney whose interest in the mat

Estimation of time and expense fo
of an unknown property is difficult. Th
vestigations, and in state of developme
frequently requires much time to bring h
conditions of the property, and to get h
thought-out position that he can submit a

In general, the examination of an unfamiliar
careful consideration, can rarely be made in le
mum fee, it is customary to charge 10 times th
commands. As time spent in the engineer's o
his routine work which he has the advantage
gineers count only half to three-quarters time

Contingent fees, dependent on or p
though just and desirable in cases where
with those of his client, are undesirable
cause the engineer can not maintain the
in making valuations. But, if a client
offers a satisfactory contingent fee, the en
his report shows that he is not entirely di

Specific features to be noted in examining
OF SAMPLES for other than the principal valu
posite samples, or to special samples the appe
plete analysis might yield useful information.
moisture, heat value, coking quality, sulphur,
able metallurgical uses, should be determined
gas in the deposit, character of roof and floor
burden (particularly quicksands) through whi
practicability of mining several seams simult
their bearing on operation of the property.
should be made of Fe, Mn, S (for Besseme
quantity of Fe present), P, SiO_2, and Al_2O_3; a
CO_2, CaO, MgO, Ti, Cu, Zn, etc. OTHER META
Co, Fe, Mn, As, Sb, S, SiO_2, CaO, MgO, and
circumstances. If several economic metals are
ration by concentration or smelting must be co
offered by the smeltery where the ore will be
earthy, ferrous, and zinc determinations. PRE
Au, Ag, and the proportion of value recoverab
ing, or direct smelting are the principal deter
listed above may be made for their indicative
covery (Sec 28, 30, 31, 32, 33). PLACER DEPO
concentrates of Sn or monazite should be dete
Amount and head of water supply; elevation
character, depth, and slope of bedrock; quant
timber, cost of removal of surface vegetation
must all receive especial consideration in exam

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red varies greatly, but the following suggestions will be found useful:

General

id barometer
ket compass
lic" tape in leather case
pick
de
struments, with pencils, ia and red ink and tracing
ink, and pencil erasers
elible pencils
d white crayon pencils
n and ink in traveling filler
ote books (quadrille-ruled
nery for drafting reports
cale of 6 scales, 1 in = 20 to
ding rule
g machine
c camera

For Sampling

6 by 12-in and, 14 by 20-in canvas sampling sacks
Twine for tying above
Sample envelopes, with flexible metal-strip mouth
Seal and sealing wax
6 by 6-ft canvas, ticking, or enameled cloth (for moisture samples) sampling sheet
Jones sample-splitting outfit
Large iron mortar with pestle and canvas cover, or collapsible mortar outfit for coal (Fig 3)
Galvanized iron screw-top cans or glass jars for moisture samples
Adhesive tape for above
Leather mail sack, with lock and keys
Whisk, clothes, or scrub brush
Single and double-hand hammers
Miner's pick
¾-in moils, 6 to 18 in. diamond points
Gold pan and vanning plaque

Additions for Placer Sampling

ape on reel
d
pencils
ottles, with corks, for gold
d balance, sensitive to 0.5
Mercury
Annealing cups
Nitric acid
Alcohol and lamp
Gummed labels
Short-handle shovel
Galvanized-iron panning tubs
Retort stand with alcohol stove

17. Writing of Reports

rms. The form of a report depends largely upon circumstances at which has gained favor in recent years is one in which an consisting of the mere formalities of submission, is followed ry and brief statement of conclusions, followed in turn by the s, conclusions, and advice. This form places the gist of the situately before the reader. A busy man gets the information he minimum time, and the interest of any reader is more promptly when the entire report must be read before its conclusions are

d more usual forms lead up through description to conclusions. This is n a report which cannot be brought to an unequivocal conclusion, and impression can be given only by the atmosphere created by detailed de- convenience of reference, the subject matter is divided by headings, zed, underscored and centered on the line, with sub-headings at the side, l and underscored. Below is a simple form of introduction, giving de- tion as to date and duration of the examination. This is followed by n have been found useful under various conditions. These, and the ragraph headings of this section of the book, furnish suggestions for de- est form for a specific case.

.....Esq., President,
.....Copper Co.,
.....Broadway, New York

Dear Sir:

Pursuant to your request, I have carefully exam
ing.....days, from.....to.....
my report thereon.

Summary and Conclusion.

Location of the property, names and relative po
areas, and titles.

History, including shipments, profits, and difficu

Economic geology, topography, vegetation, clim
topographical maps.

Governmental conditions (when below standar
and police protection.

Sampling methods, checks, limits of accuracy, a

Tests on character and extractable value of the

Assured mineral, calculations, and amount.

Prospects and limitations of the property.

Present equipment (photographs), mine devel
of working and of treatment.

Supply and cost of labor, water, power, fuel, ti

Operating costs, general conditions, and profits.

Market (for mine product) conditions, and assu

Advice, respecting development, scale of operat

Estimated returns on investment.

Acknowledgments (of courtesies received and to

18. Estimating Standing Timber

Estimates of timber tracts are based on a
logging and of transportation, and value of
the degree of accuracy required in an estimat
work of the CRUISER, or timber estimator, v
spection to an exhaustive and detailed tree co
by running once through each 40 acres (1/16 s
the timber in a general way or estimate the
COUNT is made by running through each fort
counting the trees on either side of a comp
cover the whole area (23).

Cost of cruising is from 10¢ to $1 per acre,

Rules observed by timber-cruisers are general
and with the proportions of different kinds of tim
is much timber that runs only 3 to 5 M ft per acre.
out heavy undergrowth, making it possible for a cr
ing an acre at intervals, taking the aver result, and
tract for total count. In the northwestern U S, n
of ridges and canyons; the undergrowth is heavy,
not uniform in stand (10-acre spots carrying pra
having 300 M ft or more); also the trees vary great
tree. Under these conditions, a cruiser must count

Methods of estimating the stand are: (
of circles, computing the contents of the aver
trees to obtain stand per area covered. T
multiply the stand per area covered by a
acreage actually estimated. (*b*) Determinin
tree in selected strips or circles, and multiplyin

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ghly estimating entire acreage by much criss-crossing of an area
ass lines (practiced mostly by men of long experience).
iffer but slightly; in the one case the number of trees is multiplied by an
e other the trees are estimated separately and added for the total. There
ifference between circle and strip methods of counting, but strip method is
r total tree count.

individual trees is found from a volume table, based on an ideal
n taper. The table gives board measure contents of trees ranging
in or more in butt diam, and carrying a different number of logs
taper for each diam. (A partial table follows.)

ne Table, Showing Contents of Logs of Uniform Taper (23)

Diam at top of 32-ft log or fractional log, in							No of logs	Aver taper, in	Volume, bd ft	
1st	2d	3d	3½	4th	4½	5th			From scale of individual logs	Volumes for same aver taper
18	16	11	..	..	..	..	3	4	879	889
20	17	13	11	..	..	..	3½	3½	1 180	1 082
19	17	15	..	12	..	..	4	3	1 294	1 281
20	18	15	..	14	12	..	4½	2½	1 563	1 596
18	15	11	..	..	..	..	3	3½	831	789
20	18	16	..	15	12	..	4½	2	1 657	1 406
29	27	24	..	20	..	..	4	5	3 656	4 556
37	34	31	..	27	..	19	5	5	6 650	6 476
44	38	34	..	29	..	24	5	5	8 762	9 398
Diam at top of 16-ft logs										
1st	2d	3d	4th	5th	6th	7th				
16	15	14	11	..	..	..	4	1¾	475	451
17	16	14	13	11	10	..	6	1¾	670	556
17	16	15	14	12	10	..	6	1¾	727	556
30	29	28	25	23	..	..	5	2	2 662	2 466
31	30	28	25	23	20	15	7	2½	3 164	2 537
35	33	33	32	29	25	..	6	2	4 298	3 718
34	34	32	31	30	27	23	7	2	4 699	4 059
33	32	31	31	31	27	26	8	2	4 908	4 335

standing trees is determined by the use of a DIAMETER TAPE, length and
d by measurement. Due to natural variations, the sum of the scale of the
loes not usually correspond to the volume given in the table for the aver
; a tree rarely has the same taper for each log; trees of same diam vary
and taper; and top and butt logs usually have a much heavier taper than
dle of the tree. The estimator must decide upon taper and number of logs

"two times run" through a forty is made at intervals of 2 tallies or 660 ft,
unting trees for a distance of 25 paces (about 66 ft) on each side of the
d multiplying the amount of timber by 5 for the total. A "four times
intervals of 1 tally or 330 ft and multiplying amount of timber by 2.5 for
n "eight times run" (intervals of 164 ft), the estimator sends a survey
cate corners, set tally stakes along the section to keep the compass-man
d do the pacing, while he counts trees.
runs through a forty are made, the estimator also gathers data for a map,
opographic details as will give the general characteristics of the whole
as for contours are based on barom readings. As the cost of logging a
rtant to know as the amount of timber on it, a good map is essential for
ng problems.

A cruiser's report may vary from a simple
and the total amount in bd ft inserted on ea
forty the number of trees of each species, their
tents per tree, and percentages of different gr
species and the logging chance, all compiled
detailed reports on forties or sections are usual
the tract as a whole, describing character of t
bility of the country, general plan of logging
A buyer wishing general information as to t
care to spend much money, and so sends an
report is satisfactory, a more detailed estimate
higher, the buyer will probably want a compl
his timber on the market needs an estimate a

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SECTION 26

ERIAL TRAMWAYS AND CABLEWAYS

BY

EDWARD B. DURHAM

SSOCIATE PROFESSOR OF MINING, UNIVERSITY OF CALIFORNIA

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. An AERIAL TRAMWAY transports small loads in carriers suspended
s forming the tracks, between fixed points, usually a long distance
are 3 classes: Double-rope, reversible, and single-rope tramways.
and third, a continuous series of loads travel in one direction,
rriers returning in the other. With the second, a single carrier
nd forth on a single cable.
r transports a load for a short distance, in a single carrier traveling
h on a single rope, a hoisting operation being combined with the
operation is intermittent.

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GENERAL FO

1. Cable Fo

Catenary vs parabola. Theoretical cur
when suspended from its ends is a catena
polygon
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polygon
hung eq
2 suppor
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tramway
by its us
OF PARA

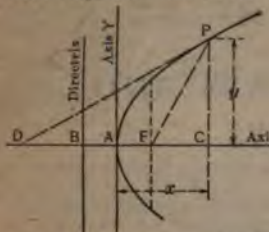

Fig 1. Parabola with Rectangular Axes

Empty cable; chord horizontal. In

s = span, ft
h = deflection at center, ft
h_1 = deflection at any point of curve, as p, ft
t = tension in cable at center, lb, = horizontal component of tension at any point
T = tension in cable at any point, lb
w = weight of cable per ft, lb
W = total weight of cable in span, lb, = ws
R = total vertical reaction at a tower; for cab
m and n = sections into which point p divide
ϕ = inclination of tangent to the curve at a

Deflection at center. Take moment
center, then: $\frac{1}{2} W \frac{1}{4} s - \frac{1}{2} W \frac{1}{4} s = th$, o

$$h = ws^2 \div$$

Deflection at any point, as p. Take
left of it, then: $\frac{1}{2} wsm - wm \frac{1}{2} m = ht$, b

$$h_1 = wmn$$

Tension in cable at any point is:

$$T = t + o$$

It is also equal to tension at any other
distance between the 2 points and the wei
tical distance between points B and p, the

$$T' = T -$$

Simplified working equations (Table 1)
tained by substituting numerical values
any one kind of cable w and t bear fairly
with locked-coil track cables, the weight
per sq in of cross-section, and the working
substituted in Eq 2, give:

$$h = \frac{ws^2}{8t} = \frac{3.6\,s^2}{8 \times 30\,000}$$

es weight and strength of cables used for tracks; by substituting different cables in Eq 2, the following table is obtained:

Working Equations for Center Deflection of Horizontal spans

e of cable	w, lb	t, lb	Working equation
able.....	3.6	30 000	$h = 0.000\,015\,s^2$
able.....	3.4	30 000	$h = 0.000\,011\,s^2$
l rope.....	*	⅓ ult strength	$h = 0.000\,014\,s^2$
	*	⅕ ult strength	$h = 0.000\,018\,s^2$

locked rope from 1 to 2.25 in diam, the mean value of weight + ultimate o 029.

Table 2. Weight and Strength of Track Cables

ed-coil cables		Smooth-coil cables			Patent locked rope		
Ultimate strength, lb	Working tension, lb	Weight, lb per ft	Ultimate strength, lb	Working tension, lb	Weight, lb per ft	Ultimate strength, lb	Working tension, lb
56 800	15 780	1.55	49 360	13 710	1.88	72 000	14 400
71 300	19 820	2.03	64 460	17 900	2.50	86 000	17 200
86 100	23 910	2.57	81 580	22 660	3.15	110 000	22 000
108 700	30 210	3.18	100 890	28 020	3.80	134 000	26 800
131 600	36 560	3.72	118 270	32 850	4.75	160 000	32 000
156 600	43 500	4.68	148 500	41 250	5.70	190 000	38 000
198 000	55 000	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	7.65	258 000	52 000
.....	.....	.....	.....	.....	10.00	340 000	68 000
.....	.....	.....	.....	.....	12.50	440 000	88 000
.....	.....	.....	.....	.....	15.50	550 000	110 000
ty.....	3.6	.....	.....	3.6	.....	.....	5.0

ion of L-C and S-C Cables, see Fig 15, and for P L Rope see Fig 44.

ngent to curve of empty angent at end of span, when ntal, has a slope (Fig 3) of

$$= 2h + \tfrac{1}{2}s = 4h + s \qquad (7)$$

alue of h from Eq 2:

$$= ws + 2t \qquad (8)$$

he reaction R, hence:

$$= R + t \qquad (9)$$

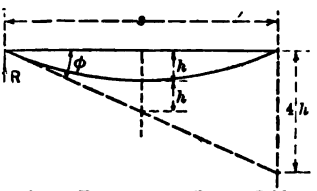

Fig 3. Tangent to an Empty Cable

case, w and t are constant; hence Eq 8 can be simplified by sub- erical values. Thus, for locked-coil cable, $w = 3.6$ lb and $t =$ ng $\tan\phi = 0.000\,06\,s$.

ed loads. Let Fig 4 be the funicular polygon resulting from 3 oads, g_1, g_2, g_3, and a, b and c be their respective distances from l. Then the reaction at left-hand end is: $r_1 = \frac{g_1 a}{s} + \frac{g_2 b}{s} + \frac{g_3 c}{s}$.

ents about o gives: $r_1 m = h_0 t$ or $h_0 = r_1 m + t$

If weight of cable be included, moments a

$$R_1 m - \tfrac{1}{2} w m^2 = Hi, \quad \text{or} \quad H$$

In Eq 11, R is reaction at left end, when b

considered, and equals $\frac{g_1 a}{s} + \frac{g_2 b}{s} + \frac{g_3 c}{s} + \tfrac{1}{2}$

Hen
load
force
horiz

Fig 4. Funicular Polygon Due to Loads Only

Ex
ing

spaced, deflection at any point is practically
cable of which the weight per ft is equal to
load divided by the spacing in ft. If value

$$H = \frac{R_1 m - \tfrac{1}{2} w m^2}{i} = \frac{r_1 m + \tfrac{1}{2} w s m -}{i}$$

but $(s - m) = n$, whence $H = \frac{r_1 m}{i} + \frac{wmn}{2i}$

to concentrated loads (Eq 10), and $\frac{wmn}{2i}$

(Eq 3). Hence, total deflection at any poin
cable alone and that due to load alone.

Deflection due to one concentrated lo

One load AT ANY POINT. Deflection at th

$$H = (ws + 2$$

One load AT CENTER. Deflection at cent
$m = n = \tfrac{1}{2} s$, whence:

$$H = (ws + 2$$

One load AT DISTANCE y from one end.
same end, by Eq 11, is:

$$H = (wms +$$

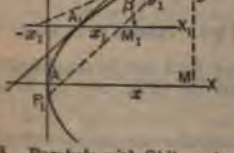

Fig 5. Parabola with Oblique Axes

Fig 6.

Parabola with oblique axes. Let Fig
lytical geometry:

$$y y_1 = \frac{2p}{\sin^2 \beta}$$

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ble; chord inclined. Let Fig 6 represent such a case, l being ord. Applying principle of moments:

ON AT CENTER, $$h = \frac{wls}{8t} = \frac{ws^2}{8t\cos\alpha} \quad (17)$$

ON AT ANY POINT, $$h_1 = \frac{wmn}{2t\cos\alpha} \quad (18)$$

rying concentrated loads; chord inclined. Total deflection at he sum of deflection due to cable alone and that due to loads alone. ue to cable alone is given above. Deflection at any point due to equals moment of vertical forces due to loads on one side of that d by horizontal tension (similar to Eq 10), thus:

NY POINT of an inclined span. DEFLECTION AT LOAD is:

$$H = (wl + 2g)\frac{mn}{2ls} \quad (19)$$

ENTER of an inclined span. DEFLECTION AT LOAD is:

$$H = (wl + 2g)\frac{s}{8t} \quad (20)$$

ISTANCE y from one end of an inclined span. Deflection at a point m same end, is:

$$H = (wml + 2gy)\frac{n}{2ls} \quad (21)$$

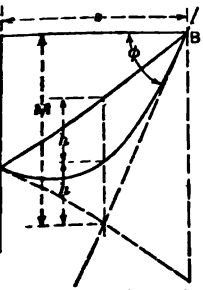
tangent to inclined empty cable. Let Fig 7 be inclined span. f tangents to the two ends are:

$$\tan\phi = \frac{M}{\frac{1}{2}s} = \frac{4h+V}{s} \qquad \text{At } A_1 \ \tan\phi_1 = \frac{M-V}{\frac{1}{2}s} = \frac{4h-V}{s}$$

of any tangent is given by:

$$\tan\phi = \frac{4h \pm V}{s} \quad (22)$$

vertical distance between ends of span, ft.



ngent to an Empty Cable

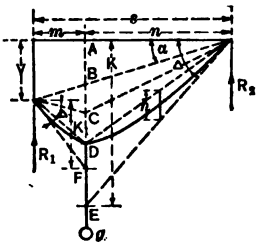


Fig 8. Tangent to a Loaded Cable

ble. Let Fig 8 be an inclined span with concentrated load at some of tangents to the two ends are: $tan\ \Delta = K \div n$ and $tan\ \Delta_1 = K_1 \div m$. are made up of a series of known parts. Thus, $K = AB + BC +$

$$\frac{V}{s}n; \quad BC = \frac{gmn}{st}; \quad CD = \frac{wmn}{2t\cos\alpha}; \quad DE = 4h = \frac{4wn^2}{8t\cos\alpha}$$

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Substituting these in above, and simplifyi

$$\tan \Delta = \frac{gm}{sl} + \frac{}{2}$$

$$\tan \Delta_1 = \frac{gn}{sl} + \frac{}{2}$$

Length of parabolic curve. The leng
L, when the rise is small, is closely app
"Theoretical Mechanics," trans by E. B.

$$L = s + \frac{8h^2}{3s}, \quad \text{or} \quad$$

If chord be inclined, the length, for taut
may be taken as equivalent to a symm
length and same deflection *normal* to th
nearly equal to: vertical deflection $\times \cos$

Horizontal tension (t). In tramway
tension applied to one end of span (usu
tension of cable. This tension is used in
makes the actual horizontal tension in cab
calculations, and hence the sag will be a li
to flatness of the curves, this error is not
angle. But on inclined spans it may ma
can be partly compensated by calling the
its direction is parallel to inclined chord
tension, to be used in the formulas, will be

Plotting of parabolic curves may be done by
BY POINTS: Series of offsets is calculated by
Eq 16 or 18, if chord be inclined. These are
their lower ends.

BY TANGENTS: Chord of span and center def
Fig 9 or 10 (same method applying whether ch
is prolonged to a total length of $2h$; lines dra

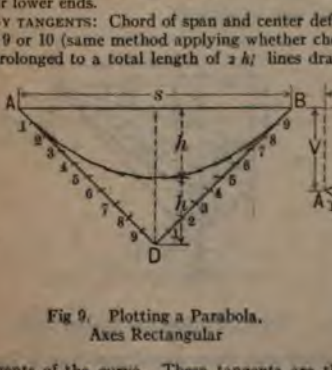

Fig 9. Plotting a Parabola.
Axes Rectangular

tangents of the curve. These tangents are t
parts, numbered as shown. Lines are next dr
these are tangents of the curve, and the curve
making a sufficient number of divisions in the t
formed by the tangents will be short and pract
and 10, where for clearness only the end tangen

2. Operating Formulas

f carriers. For a system operating a continuous series of carriers,

t = time interval between carriers, sec
d = distance between carriers, ft
l = load in one carrier, lb
s = number of tons (2 000 lb) to be carried per hr
v = velocity of tramway, ft per min

$2\,000 \div 3\,600 =$ lb per sec to be carried by tramway, and this l, gives time interval in sec, reducing to,

$$t = 1.8\,l \div v. \tag{25}$$

ween carriers is, $d = tv \div 60$, reducing to,

$$d = 0.03\,lv \div v. \tag{26}$$

n traction rope. Tension in a rope holding any body at rest on eglecting friction, is $T = W \sin \alpha$. Friction of a body moving on $F = fW \cos \alpha$; in which,

T = tension parallel to the plane, in same unit as W
W = total weight of bodies on the plane, preferably in lb
α = inclination of plane
f = coefficient of friction
F = total friction, in same unit as W

body extends whole length of incline, as a rope lying on it, then is $W = wL$, where

w = weight per ft, in same unit as W, preferably in lb
L = inclined length of the plane, ft

this value of W in general equations for T and F above, gives:

$$T = wL \sin \alpha = wV \tag{27}$$

$$F = fwL \cos \alpha = fwH \tag{28}$$

V = vertical distance between ends of plane, ft
H = horizontal length of plane, ft

traction rope of a double-rope tramway, or in endless rope of tramway, at its upper end, taking friction into account, will be:

$$T = wV \pm fwH \tag{29}$$

erm is + when rope is being pulled up the incline and − when it own. If rope be empty, w is the weight per ft of rope alone; but attached to it, w must be increased by weight of the carriers uniouted over whole length of incline, or else the effect of carriers ulated separately and added to value of T.

3. Transmission of Power by Band Drives

sion is applicable to any type of band or rope drive, or to band tension is known.

rive, tension on slack side must bear a certain relation to tension on taut p sufficient friction to prevent slipping. The formula for this relation, l mechanics, is: $T = Se^{f\pi\alpha}$

in which: T = tension on taut side } Bot
S = tension on slack side }
e = base of Naperian logarithms
$\pi = 3.1416$
f = coefficient of friction (values
n = number of half laps on drum

The useful effect, or force transmitted by th
on the two sides, that is, $T - S$. If S be subt
comes, on reducing: $T - S = S(e^{f}$

which is an expression for force transmitted by
S for a given drive when power to be transmi
$e^{f\pi n}$ for a number of combinations, for saving

Table 3. Coefficients of Friction, f, fo

Condition of surfaces	Iron
Dry	
Wet	
Greasy	

Table 4. Values of $e^{f\pi n}$

f	n = Number of half l		
	1	2	3
0.070	1.246	1.552	1.934
0.085	1.306	1.706	2.228
0.100	1.369	1.875	2.566
0.120	1.458	2.125	3.099
0.130	1.504	2.263	3.405
0.140	1.552	2.410	3.741
0.150	1.602	2.566	4.111
0.170	1.706	2.910	4.964
0.200	1.875	3.514	6.586
0.205	1.904	3.626	6.904
0.235	2.092	4.378	9.160
0.250	2.193	4.810	10.551
0.265	2.299	5.286	12.153
0.300	2.566	6.586	16.902
0.350	3.001	9.017	27.077
0.400	3.514	12.346	43.376
0.410	3.626	13.146	47.663
0.450	4.111	16.902	69.487
0.495	4.716	22.425	106.194
0.500	4.810	23.140	111.318

Most calculations for tramway drives and bra
for wet ropes, as this often occurs; neither rope

Grip sheave, of type shown in Fig 29, has
which are tightened by the rope's pressure; the
sheave by the ratio of lever arms of the jaws,
friction. For a grip sheave with iron or steel j
$e^{f\pi n} = 2.228$, and where the leverage is 1 : 3
which $f = 0.085$.

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DOUBLE-ROPE TRAMWAYS

4. General Considerations

A double-rope tramway consists of: 1, two track cables stretched
nsion; 2, an endless traction rope for moving the loads; 3, numer-
or the loads, each fitted with carriage or trolley to run on track
amp or grip for seizing traction rope; 4, a station at each end to
trol the traction rope and provide places for loading and unloading
termediate towers for supporting track cable and traction rope.
n, each load is placed in a carrier while standing on a track in
ial; the carrier is then fastened to traction rope, and is hauled
k cable to discharge terminal; there it is released from traction
ents are discharged. Empty carrier is then fastened to traction
side and is hauled back to loading terminal.

Economical limits of capacity for double-rope tramways are ap-
o tons per hr as minimum, and 100 tons per hr as maximum.
tons per hr is to be carried, it is more economical to operate the
hours or on alternate days, than to reduce its carrying capacity.
o tons per hr requires quite heavy loads at frequent intervals, and
ut the maximum that track cables will stand without excessive wear.

is seldom wise to use double-rope tramways for distances less
because terminal machinery costs as much as for a longer line
first cost excessive. Maximum limit, under favorable conditions,
es; the resulting 8 miles of traction rope is as long as can be
ssfully. If distance be greater, a series of tandem tramways can
the carriers switched from one to another; but each division is
independent tramway. Thus, a tramway at Chilecito, Argentine,
comprises 8 divisions. In any tramway, the track cables are
ections of 3 000 to 5 000 ft each, one end of each section being
tension weights attached to the other end. The carriers pass
rage and tension stations without interruption, so lengths of cables
peration nor possible length of line or division.

t weights of 500 and 1 500 lb are the usual lower and upper
s on ordinary tramways; below 500 lb there is no saving in cost
d over 1 500 lb the wear on cables is great. The time interval
rs is conveniently about ½ min. Then, 1 500 lb every ½ min =
, and is near the maximum capacity of double-rope tramways.
vorable profile conditions, 2 000-lb net loads can be carried, and
time interval can be cut down to ¼ min, but it is seldom feasible
val and increase load at same time.

5. Tramway Surveys

proposed route, with surface elevations, must be made and a
er-line must be plotted before a tramway of any type can be de-
mmits, or wherever sudden changes in profile occur, elevations
rmined at intervals of 25 or 50 ft. The stations staked out may
, 2, 3, etc, for distances of 100, 200, 300, etc, ft from the origin,
veying; intermediate distances are noted as + ft from preceding
ery rough country, distances and elevations of important points on
be determined by stadia measurements, and the position of addi-
on each side by taping and leveling from the stadia points. In
ata collected must be such that an accurate profile can be plotted.

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Location line of tramway should gener
distance between terminals, and is therefo
exception occurs where tramway passes ove
file; then, for a line of large tonnage, or whe
steep, it is advisable to search for a lower pa
profile is made from loading terminal straig
discharge terminal, making a horizontal ang
It may be found, on laying out the two rout
more advantageous. If the ground has a ste
cross-sections should be made at intervals,

Additional data. Besides profile of center-lin
site of both terminals, with contours; and it sh
which tramway must be connected. These pla
larger than the proposed terminal, say 50 ft on ea
it. Similar plans should be made of sites along the
likely to be located, in order to place all conditions
Relation of plans and cross-sections to points o
reference marks on each, and permanent marks must
can be located when construction is begun. It is s
points near terminals, and if these are likely to be d
have, other reference points must be set where they w
be plotted by profile, cross-sections, and plans, befor
all necessary data have been secured; but survey no
of, each point should be sent in with the plots, so th
saved, without having to scale the surveyors' drawing

6. Design of Double-rope

Economical proportioning of interrelated
established by experience, as given in Table 5.

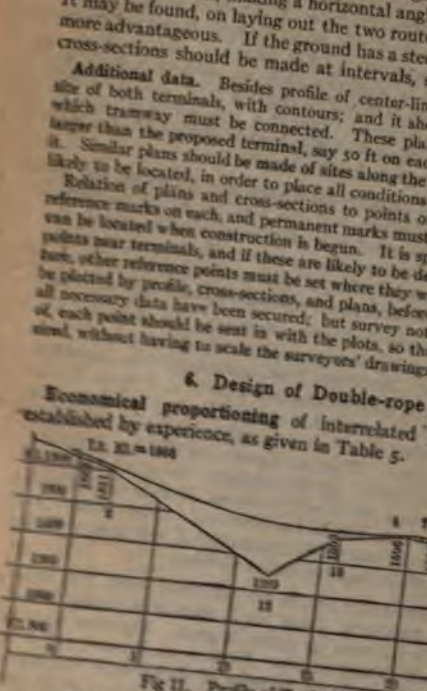

Fig 11. Profile of Typical Double-rop

Table 5. Relations of Cable Sizes, Loads,

Tons per hr	Diam of cables, in		Net loads, lb
	Loaded side	Empty side	
8 to 20	1	¾	500
15 to 40	1¼	⅝	600
25 to 50	1¼	¾	800
30 to 60	1¾	¾	1 200
40 to 75	1½	1	1 500
50 to 100	1¾	1	2 000

* For lines carry ing 25 to 50 tons and for 50 to 100 tons, lengths in small for lines over 2 miles long, where driving mach for lines that carry bulky material requiring large carriers.

binations are based upon the fact that the size of load for a given ermines the number of carriers on the line; if load be small, the g stock is increased, due to the number of carriers, since part of unit is constant; if load be large, the size of track cable increases n to the individual weight carried, although the loads may be few The empirical rule used in Table 5 to determine SIZE OF TRACK allow 1 sq in of cross-section for each 1 600 lb gross weight of one allowance is decreased a little on smaller sizes and increased a larger. The GAGE (distance apart of track cables) may be 6 ft riers, but must be 8 ft for large ones, and in some cases may need Though varied to suit unusual conditions, bulky materials, or special interrelations apply to all ordinary cases.

speeds have constantly increased. Speed must be high to carry e without increasing weight of moving parts, nor the stresses due ht, but when speed is too high it is difficult to handle the carriers The practical speed is about 8 ft per sec, with 10 ft per sec as ith favorable conditions and careful designing; equivalent to 480 er min respectively.

e a typical profile on which is to be constructed a tramway to carry 60 000 lb r hr, weighing 100 lb per cu ft. By Table 5, the cables will be 1¼ and e carrier capacity will be 800 lb or 8 cu ft each, and gage will be 6 ft. If n at 480 ft per min, the spacing of carriers will be (Eq 25 and 26): time $.8\,l + s = 1.8 \times 800 + 30 = 48$ sec, and distance apart $d = tv + 60 = 48 \times$ 4 ft. The weight of an 8-cu ft bucket (Fig 17) is 420 lb, making gross loaded carrier = 420 + 800 = 1 220 lb.

the line. The profile is plotted to a natural scale, say 1 in = 40 ft " profile paper, 20 in wide, ruled 4 by 20 lines to the inch. If differ- tion between ends of line be great, several strips may be joined to- required width. If line is too long to be operated as one division points are determined and each division is laid out as a separate ne junction stations are designed (Art 17).

n of the profile, points where a rope stretched from one terminal to the st touch the ground can be seen; in Fig 11, these will be at Sta 21 and r. Sta.

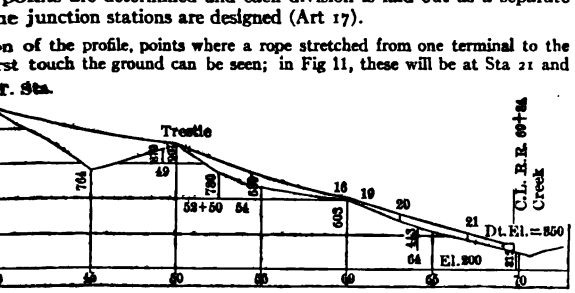

Fig 11 (Cont'd) Profile of Typical Double-rope Tramway

le is then tacked to a long table, exposing one portion of the line, say from al to the first high point; and the terminal structure is plotted in outline the conditions imposed (Art 16). A pin is set in the terminal at point ves the horizontal rail, another pin is set at the minimum height of tower ay 12 ft if snow does not lie there over 3 ft deep); a silk thread is looped assed around the other and led over the edge of the table where the spool's having been taken in thread to prevent unwinding) draws it tight. If er be ruled in green, and red thread used, there will be no confusion with ncil lines that may be added. The thread represents a chord, from which can be plotted by laying off deflections at as many points as are necessary.

At crest at Sta 21, a similar group of towers oc
of problem. A point *A*, at Sta 21 and El 1600,
drawn to Tower No 3. It is evident that a cur
ground, hence Sta 20 is a more suitable terminus
set at Sta 20 and El 1606, and a new chord drawn
Sta 2 to Sta 20 = 1 800 ft horizontally. Next, t
computed and the tangent *B* to *C* is constructed.

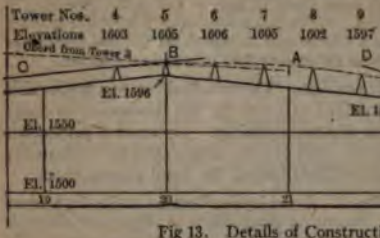

Fig 13. Details of Construct

that the cable will follow slope of ground to Sta 2
and 15 ft above ground. A trial of several circula
(which, at scale of 1 in = 40 ft, corresponds to 800
and *DE*, conforms to the profile. By Table 6, the
apart, for deflections of 5% on each. This arc is th
points, with a distance between them equal to a
plotted up to the curve with a spacing of 40 ft, thu

Tramway trestle is a structure with a ser
of 5 to 20 ft, with a slight deflection of cabl
summit where a group of towers would be ob

Fig 14 shows an application of this constructi
The figure also gives, for comparison, the graphic

Fig 14. Details of Construction

towers, similar to arrangement at Sta 21. A tria
and, as the spans are long, the tangents to loade
and *AC*. A curve of 800-ft radius is drawn tange
40 ft apart giving the 8 towers, shown by dotted
are undesirable, owing to their heights (upper one
cutting the summit of ridge. Next a trestle with
5% on each is investigated. Under these conditi
200-ft radius (Table 6). This curve, placed tan
necessarily high at the left, hence line *AB* is lowe
of tangents to the right. The structure adopted i

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final design of a trestle, the elevations of end points are scaled he inclinations of tangents to loaded cables are computed and the he angle of deflection between the two tangents; this divided by esponding to desired deflection, gives number of saddles required. slope of chord from saddle to saddle is found and the elevation e calculated to nearest 0.1 ft. A similar process may be used to ations of individual towers of a group, after locations of end een determined by scaling.

es are kept taut by a weight attached to one end of each section, being anchored (Art 11). On short lines, the anchorage is at one he tension is applied at the other. A section from 3 000 to 5 000 s can be controlled from one tension point, hence long lines must o sections of approximately 4 000 ft. Exact lengths of sections by locating, along the line, suitable sites for stations. It is cus- ce anchorage at the high end and apply tension at low end of the weight of cable aids in working the slack down to the tension rection is also facilitated. Intermediate stations are necessary (Art 12). The typical tramway (Fig 11), being nearly 7 000 ft one intermediate tension station, placed on the convenient ridge It is constructed as an anchorage and tension station (end Art 12).

7. Track Cables

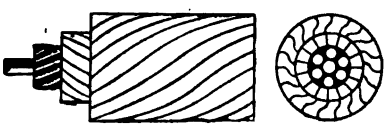
s for all types of aerial tramways should be either smooth-coil or les. They are made of large wires (Fig 15) in order to have the surface wear of the carrier wheels. Ordinary stranded ropes are not practical because the wires quickly wear through.

l cables have a smooth exterior, the shape of outer wires re- ngular oblique figure 8, so that they interlock and are prevented loose if one breaks. ature is operative if ose contact, but is ire in manufactur- keeping the layers . Locking is suffi- e broken wire, but near together the l unravel.

l cables are made d wires. They are s locked-coil cables, ly smooth by wear. lo not interlock, a orks out at once ust be made more with l-c cables. easier and cheaper the l-c. Either ory for tramways.

(*a*) Smooth-coil Cable

(*b*) Locked-coil Cable, ⅞ and 1 in diam



(*c*) Locked-coil Cable, 1¼ in diam and up

Fig 15. Track Cables

ce of cable is limited by length of wire that can be manufactured. Sec- quired for long tramways are joined by couplings to form a continuous orage to tension station. Each coupling (Fig 16) is made in halves, wedged to end of each section, the two halves then being connected

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a plug having right- and left-hand threads. Co
consistent with strength, and the carrier wheels
them. At the ends, the cables are attached to
used for couplings. The end sockets are designe
is rotated. Couplings should not come within 15
other, so that the cable will be free to give as a c
receive sharp bends at its ends.

Attaching couplings and sockets. Cable is
from end to prevent distortion; it is then sawed

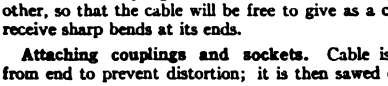

Fig 16. Coupling for Track Cables

annular wedges driven between them, spaces betw
wedges and any small spaces remaining are filled w
process grips each individual wire and makes a s
prevents it from pulling out of the taper. Wedge
hammer and punches, but a screw press does a q
the fitting is placed on cable as above, thimbles an
way down; then the fitting is placed in the press,
and the bulged end of cable pushed firmly into the

Rotation of track cable. Every week
through ¼ of a revolution. This distributes
prevents flattening of cable and displaceme
be such as to tighten the outer wires of cabl
lay (Fig 15), the cable should be rotated to t

Maximum wear on a track cable occurs i
of a saddle on a summit. Directly over the s
est, due to pull of traction rope combined w
the saddle the cable gets a reversed bend b
especially on the side where loaded carriers
ward pull of traction rope can be reduced
traction-rope rollers (Art 10), and bending mi
as its strength will permit.

Cables are oiled at intervals of about a m
rusting, to provide internal lubrication, and
may be done by men riding slowly over the
oil on cable with a handful of waste, or it ma
device, so equipped that it can be hauled
saddle grooves should be greased to facilitate

To repair a cable having broken wires,
cable, tackle is attached on each side of bre
and break is cut out. If break is short, half
and they are united with the right- and left-t
new piece of cable is inserted in the same m
never be less than 10 ft long, or wires will l
stiffening of cable by couplings.

8. Rolling St

Each piece of rolling stock consists of: c
for attaching to traction rope.

Carriage runs on track cable; its wheels, 8 to
with a connection for the hanger supporting the c

ılk materials is a bucket; for other materials it may be a tank, a plat- For timber or similar long articles, the hangers terminate in hooks, to es are attached by slings; these are used in pairs, one at each end imes workmen are transported by a tramway during certain hours of rs like a long bucket, each holding 2 men, are best for this purpose. each end on a low block, with his legs between those of the other man. or bucket is often used for the complete piece of rolling stock.

ar steel, with cast-iron or cast-steel parts to facilitate connection with ainer, and to hold clamp for traction rope. Its form is largely governed hich it connects.

ction rope may be a GRIP, which can be applied to or released from nals, thus allowing carrier to be shunted to loading or discharge point; IP, permanently attached to both carrier and traction rope.

grips. OTTO type, in which the pressure to hold traction rope a screw operated by lever (made by 2, 4, 5, Art 27); WEBBER ressure is obtained by a toggle operated by lever (3, Art 27); in which the weight of container and its load, operating through ws on the rope (1 and 3, Art 27). Webber and Bleichert grips ined with carriages to meet special conditions. Webber and be attached by hand, but are usually closed by an attacher; released automatically when carrier enters terminal. Bleichert pecial system of rails for lifting weight of container, in order to ratory to inserting traction rope on outgoing side, and a similar ing side to open grip and release traction rope.

by most of the space, not needed for clearance, between carriage and car- which support traction rope at towers must be placed below the carriers. s good on most lines and carrier is free to swing sidewise through consid- ls are often used to fend the rope on to the rollers. Other grips allow sup- be placed close under their jaws, so that the traction rope is lifted but ses the sheave. This requires carriers to be guided, to prevent colliding ıd to make the rope drop back on sheave after grip has passed. Guides hen applied to every tower, but to have the sheave close under the grip us in passing over crests, as it reduces downward pressure. Guides are n summit structures. (See discussion regarding Fig 18, Art 10.)

ıped band of steel, bent around traction rope and permanently bolts. As clip can not be released from rope while tramway erminals must be so arranged that carriers can pass around buckets being loaded and discharged while in motion, by an r. Tramways with clips are radically different from those For description, see Art 19.)

Dimensions

Capacity of bucket, cu ft		
; .	10	20
with traction rope roller be- carriers; 5′–0″ with traction e sheave under grips		
3″	1′–10″	2′–5″
7″	2′–11″	3′–2″
1″	2′– 3″	3′–0″
5″	1′– 9″	2′–3″
00	420	550

oximately applicable to buck- 15, 16 to 25 cu ft, respectively

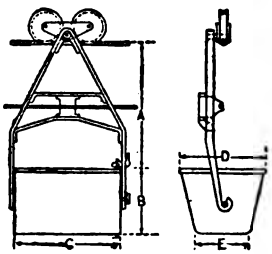

Fig 17. Ore Carrier

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Dimensions of 3 sizes of ore carrier ar
design these may be used for several sizes
those noted. The distance *A* differs co
minimum is about 3½ ft, when the tract
carriers. *A* must be increased to clear sh
under the grips, to admit special latches
ance in terminals, and for loading purpo
grip is kept constant on each system, at al

9. Traction

This is usually of cast steel with 6 stra
lay. This provides large wires, to resist
reasonable flexibility. Where stress is ve
down weight, the rope may be of wire
cast steel.

Lubrication. Traction rope should be oiled
terior. Lubricant must be carefully chosen, so
A thin oil may be used in abundance, to penet
off to leave surface dry, or a thick sticky lubrica
between wires and working into interior, while
clean. Admixture of linseed or other drying
causing exterior surface to harden, and enabli
lubricating the interior.

10. Tow

Stresses from three sources are provid
saddles, due to vertical component of tensi
carriers, and vertical component of tensio
tion parallel to line of tramway, due to ch
the cable when a load approaches and leav
to wind.

1. **Downward pressure** on saddles is ofte
must be of oak to resist bending stresses and cru
installations it has been necessary to make them

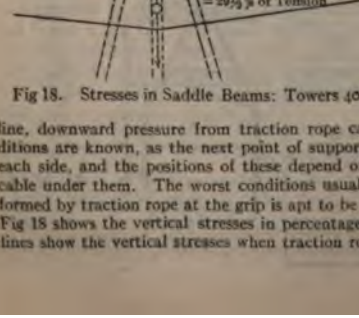

Fig 18. Stresses in Saddle Beams: Towers 40

line, downward pressure from traction rope c
ditions are known, as the next point of suppor
each side, and the positions of these depend o
cable under them. The worst conditions usua
formed by traction rope at the grip is apt to be
Fig 18 shows the vertical stresses in percentag
lines show the vertical stresses when traction r

lines show stresses when traction rope is supported on rollers 8 in below in the grip.

The problem of supporting the saddle beams and providing rigidity against difficult by necessity clear passage for the

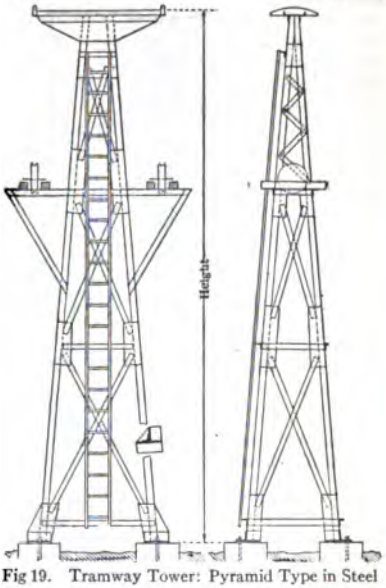

Fig 19. Tramway Tower: Pyramid Type in Steel

wer base is spread er line of tramway, ressure of cables on within base, even moderately inclined; y resist overturning o line. High towers r in other directions 22), and should be esistance to wind at ne.

er (Fig 19) is eco- terials and is satis- of steel; when of ll area at top of almost imposs ble le beams so that ck and work loose, between a large side of a wooden mall.

er (Fig 20) is rigid ut contains about per per ft of height h is nearly double timber used in Sometimes, on light lines, the outside center posts are omitted k.

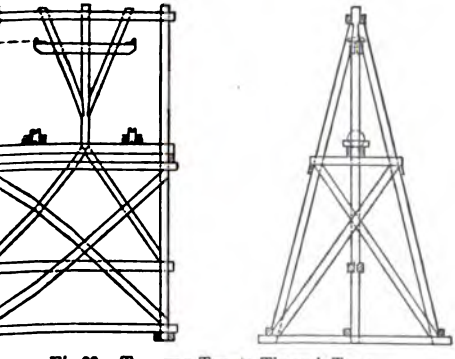

Fig 20. Tramway Tower: Through Type

wer (Fig 21) has a top that is identical for all towers, the base give desired heights. Base stands with its diagonal on center

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line, hence its long dimension is in line with
be rigid against racking. Framing is simpler
towers contain less timber than those of thro

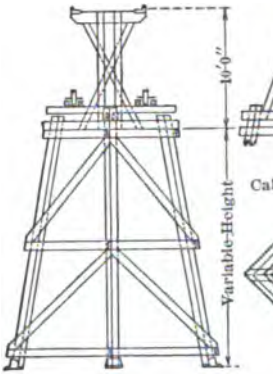

Fig 21. Tramway Tower:

Side-hill tower (Fig 22) is used on steep
sure shall fall within base.

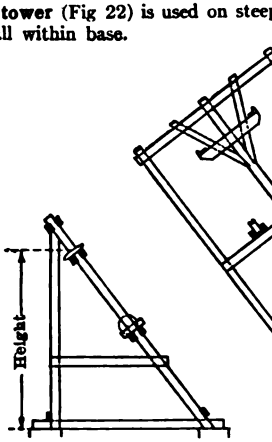

Fig 22. Tramway Tower

11. Anchorage and Tensio

Track cables must be taut enough to supp
when carriers pass tower saddles.

Tension weights are usually wooden boxes
tonnage lines, scrap iron to reduce bulk. Fo
for closely-packed scrap iron, 350 lb. Box p
insure working tension in agreement with str

x are influenced by its allotted space. A tension weight is capable
3 000 to 5 000 ft of cable, depending on profile of line and the care
lle grooves are kept clean and well greased.

t opposite end of cable from tension point must be designed
to fact that the slope of line may increase the weight-box tension.
cables terminate in station framework. For steel construction
lest arrangement; for timber it is often difficult to design the
e these stresses, and it may be more economical to deflect cables
rry them through station and attach them directly to a masonry
und. The anchorage block must be a monolith, equal in weight
component of the tension, allowing 125 lb as weight per cu ft
oncrete, and the front must have sufficient area to give a bear-
gainst the earth equal to horizontal component of tension.
tance of average earth with level surface may be taken as 1 000
t on a down-hill slope a shallow block has little holding power
age must be carried deep into the hill by a tunnel. For an
k, a T-shaped trench can be excavated and the anchorage bars
ce around them being filled with concrete. Metal parts of the
be designed for required tension, and also to allow the cable
rt 7), while under tension. Anchorage bars must be enclosed
a point above water level, and must extend above dampness
to cables, to avoid injury to the wires from rust.

12. Intermediate Stations

ded to apply working tension to track cables which are too
l apparatus to be effective throughout (see end of Art 6).

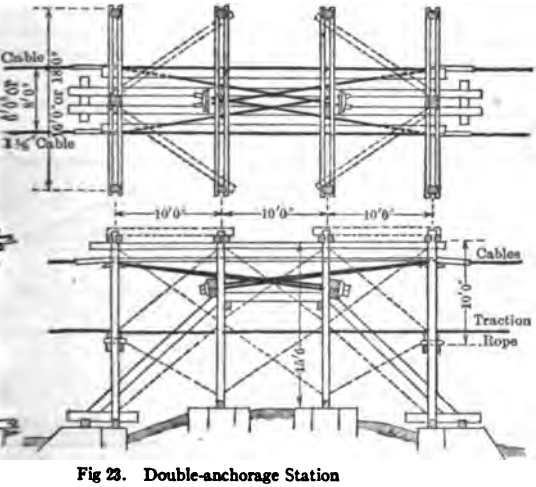

Fig 23. Double-anchorage Station

: Double-anchorage; double-tension; anchorage and tension.
c cables are deflected to center of structure, out of the way of
s downward to anchorage or to tension weights. Space be-

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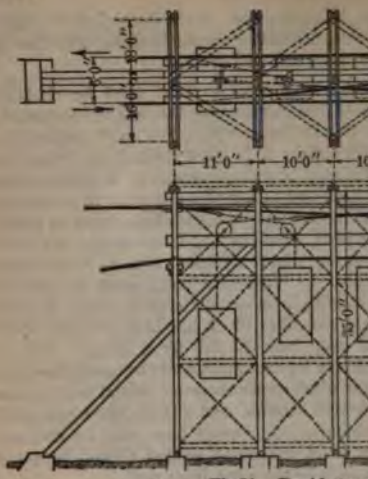

Fig 24. Double-tens

Fig 25. Angle

of track cable is bridged by rails, to allow carriers to pass through detached from traction rope.

irs of cables are in place and under full tension, the pull of those on one ill counteract the pull of those on other side, but the structures must be be secure if cables on either side are released while the others remain n, a condition which may occur during erection, while making repairs, ident. These stations are similar on all tramways, but must be decial conditions occurring on each profile. They are usually of wood; eel, in which case the bents of structure can be simplified and all posts the cables, somewhat like the steel tower, Fig 19, or wooden summit

horage stations can be as low as profile conditions will permit, is ample clearance for carriers above the ground. Where there need not be over 10 ft high. Fig 23 shows essential features, hored to the structure.

ion stations must be built high enough to allow tension weights n of 6 to 8 ft, and still have their tops below path of carriers. weights, this makes stations 20 to 30 ft high to the rail (Fig 24).

ached to chains which pass over wheels mounted on short axles, resting izontal pull is transferred to main beams by gains in the timbers, into s fit. Main beams support the vertical load and transfer horizontal d braces, which in turn transfer them to foundations. Inclined braces pward thrust when design is such that the full weight of tension box end of inclined brace, or when all the cables are not under tension. ces are resisted by giving the foundations a mass equal to the uplift, sts to them.

nd tension stations provide for anchoring the cables of one lying tension to those of the other. They are practically half ion station combined with half of a double-anchorage station of

13. Angle and Summit Stations

s are required where a horizontal deflection must be turned. terminated, traction rope passes through and is deflected by are detached from incoming traction rope, run over a rail tween cables, and are reattached to outgoing traction rope. imple angle station, with both pairs of cables anchored. (They mplicated by addition of tension apparatus for cables.)

hown, the carriers detach automatically, but 2 men are required to traction rope. Occasionally, angle stations are designed so that carreattach automatically, or are arranged so that carriers make the turn eased from traction rope. Either arrangement complicates details cost of the whole tramway, but reduces operating expenses by saving e station. An angle station may be introduced at a junction, between tramway, without increasing first cost or operating expense (Art 17).

ons are trestles introduced at crests in a line to save erection vers (see near end of Art 6) and carry a series of saddles with a pe on each.

are similar in purpose to trestles with saddles. The main ed through the station, but rails are mounted above them, on rs run, and so relieve ropes of wear. Each end of the rail has he carriage from cable to rail and let it down from rail to cable Rail trestles are used where rolling pressure is heavy, due to

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heavy loads, limited space for structure, l
where conditions prevent a reduction of v
struction.

Traction rope, in both forms of summit
distance below the track. The larger the ra
will be the deflection of traction rope at the
sure and wear of the carrier wheels on the tr

Fig 26 shows a trestle with saddles set on an ar
the carriers. Downward stresses due to traction

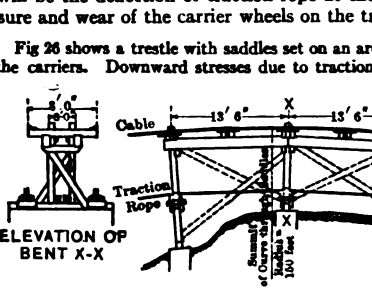

Fig 26. Summit Station, Tract

by increasing radius of curve through the saddl
will allow traction-rope sheaves to be placed cl
component and rolling pressure will be less. Ver
methods to those shown in Fig 18.

14. Power Required or Dev

Tensions. Given: weight of individual
$= e$; spacing of carriers $= d$; weight per f
weights per ft of uniformly distributed load
plus weight per ft of traction rope, are:

For loaded side, $$w_l = \frac{l+e}{d}$$

For empty side, $$w_e = \frac{e}{d} +$$

If w be the total uniformly distributed we
sion in traction rope at upper end of line, b
is descending, the sign of last term is minus
in rope equals the tension thus found plus
weight applied to a sliding sheave. Let t
tension weight $= t_w$.

Power. Let gross tension on taut side =
$= S$. Then $(T - S)$ is the force to be applie
opposed to traction rope, to restrain it. T
rope in ft per min (v) gives the work done
hence, power required or developed (omiting

$$\text{H p} = \frac{(T -}{33}$$

Friction losses. Power thus computed
required, or decreased, when power is dev

rminals. If friction coefficient of this machinery be 0.33⅓% of its the power to overcome friction will be

$$\text{Friction h p} = \frac{0.00\frac{1}{3} Wv}{33\,000} = 0.000\,000\,1 Wv \qquad (37)$$

= weight of moving terminal machinery in lb, and v = velocity of min.

Eq 36 and 37, gives total power of tramway as

$$\text{H p} = \left(\frac{T-S}{33\,000} \pm 0.000\,000\,1\,W\right) v \qquad (38)$$

t instant of starting, inertia of the line is to be overcome, and ance is higher than when line is running. To insure sufficient r, the following empirical rule may be used: in determining the ed by a gravity line in starting, divide l by 2; or for the power art a power-driven line, multiply l by 2, before substituting it in

ght may be carried on a tramway. This at times increases tension on ndition which must be taken into account in power calculations. On a siderable up freight can be carried when line is running at full capacity oads, but to carry up freight when there is little descending material, ery must be supplied, for use at such times.

q 36 and 38, T and S must be known. But when a drive is first these factors are unknown, since each is composed of the stresses lus half the tension weight, the amount of which has not yet been however, as half the tension weight is applied to each side, the S) will be equal to the difference of the stresses on the two sides ds, as determined by Eq 29, without considering tension weight.

eight. A sheave which will furnish friction necessary to drive or action rope is found by Eq 31, by substituting the value of $(T - S)$ the values of f and n suitable for the friction surface, and number nder consideration, and then solving for S, the tension on slack less than the tension for this side as found by Eq 29, no tension e applied by sliding sheave to provide the necessary friction. r than the tension on empty side, the deficiency must be supplied veight equal to twice this difference, applied to a sliding sheave. ations may indicate several possible types of drive with their re- on weights, and the merits of each can be considered. If S be ension produced by loads on slack side, a sliding sheave is needed e up slack in traction rope, and the weight applied need be suf- keep the rope taut near sheave. If the resulting tension, after t is added, is greater than necessary, it shows that the power ble of doing more than is required. Having found the weight to sliding sheave, T can be found by adding h_w to tension on taut ne loads as found by Eq 29. This is the stress that finally de- f traction rope required.

f these principles to the typical tramway, Fig 11, illustrates method of Fixed data for this line are:

Weight of load = l = 800 lb
Weight of carrier = e = 420 lb
Spacing of carriers = d = 384 ft
Weight of traction rope = r = 0.89 lb per ft
Horizontal length of tramway = 6 900 ft
Difference in elevation of ends of tramway = 1 866 − 350 = 1 516 ft.

Let Fig 27 represent CONTROLLING MACHINERY arrangement for this tramway. Calculation of t and required for ascending loads

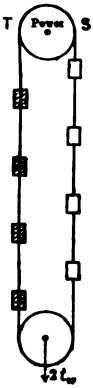

Fig 27. Power Machinery at Upper End

LOADED CARRIERS DESCEND

Loaded side; descending.

$$w = w_l = \frac{l+e}{d} + r = \frac{1\,220}{384} + 0.89 = 4.$$

$$t_l = 4.07 \times 1\,516 - 4.07 \times 6\,900 \times 0.$$
$$= 6\,170 - 562 = 5\,608$$
$$(T - S) = t_l - t_e = 5\,608 - 3$$

LOADED CARRIERS ASCEND

Empty side; descending.

$$w = w_e = \frac{e}{d} + r = 1.98$$

$$t_e = 1.98 \times 1\,516 - 1.98 \times 6\,900 \times 0.$$
$$= 3\,002 - 273 = 2\,729$$
$$(T - S) = t_l - t_e = 6\,732 - 2$$

In above case, for descending lo line, controlled by hand brakes, (stresses being too great for wood fi and $n = 2$; hence from Table 4, $e^{f\pi n} = 1.706$, which $S(e^{f\pi n} - 1)$, or $2\,333 = S\,(1.706 - 1)$; hence $S = 2\,3$ equals value of t_e; hence t_w might be zero if there power margin of 25%, new value of S is $4\,125$; subst gives $t_w = 4\,125 - 3\,275 = 850$ lb; therefore the should be 1 700 lb, which is reasonable.

To haul the same loads up the line, a grip sheave (Fig 29) would be necessary to develop required jaws have a ratio of 1 : 3; then $f = 3 \times 0.085$, $n = 2.228$ (Art 3). Substituting these values in Eq 31 $S(e^{f\pi n} - 1)$, or $4\,003 = S(2.228 - 1)$ and $S = 4\,003 \div 1$ which is greater than t_e. Hence, with no margin o sion weight of $2(3\,250 - 2\,729) = 1\,042$ lb is essen purposes. It will be well, however, to add 25% make up the increased tension on slack side by add sion weight.

Another combination occurs with CONTROLLING LOWER END of tramway (Fig 28). Then the tensio due to loading will be zero, and the only tension o driving sheave will be that given by tension weig sliding sheave; that is, $S = t_w$. Tension on taut si

$$T = (w_l - w_e)\,V \pm (w_l + w_e)\,fH + t_w,$$

the plus sign being for ascending loads, the minus f From these equations the force to control move $(T - S)$ can be determined. For ascending loads, p required, but for descending loads there are two c term may be the larger and the value of $(T - S)$ posi 2, on a gentle slope, second term may be the larger force is required to pull the loads down. With larg it is not practicable to place power machinery at l side must be obtained artificially, and by so doing t hibitively. Value of $(T - S)$ is substituted in Eq 3 and the value of S is determined. Since $S = t_w$ for t the required tension weight is known at once.

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15. Power Machinery

ves on plain sheaves may have one or several half laps about the cast-iron sheave with a single groove, giving a half lap on the es a satisfactory drive, but the pull exerted is small. Two or more cast-iron sheave, where the grooves are positively connected, is an e arrangement, because the rope in first groove has a heavier tension the others; hence this groove wears faster and becomes of smaller an the others, which produces a differential stress of large but un- unt. Multiple-groove drives are satisfactory if the grooves can p to compensate for this differential stress. Elliptical-grooved several laps about them, as used on capstans, are not applicable work, as the continual rolling of the rope down the slope in one l twist it and give trouble by tilting empty carriers.

ed sheaves can be used for transmission of considerable power, S) is large but actual tensions are small. If tensions are heavy, the cut out.

nd leather-filled sheaves give maximum friction for rope drives, y cut and not applicable to tramways.

ves give a strong hold on rope for a single half lap on sheave, and uce differential stresses in rope. The Hallidie pattern (Fig 29) is

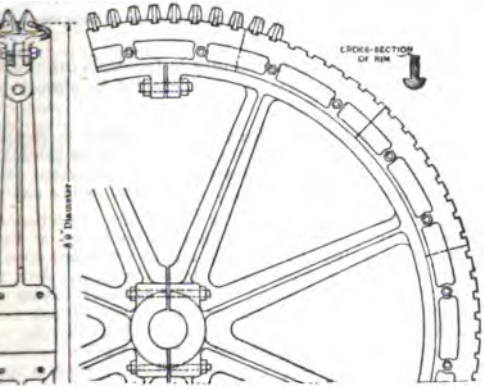

Fig 29. Hallidie Grip Sheave

to transmit considerable power on tramways, and does not injure e. The jaws multiply the rope pressure by 3 or 4, and hence in- n by that amount. This has effect of increasing f in Eq 30 and 31 ount, whence the power transmitted by a single grip sheave is o that of a 3 or 4-grooved iron sheave. The grip sheave costs me as the multi-grooved sheave and its idler.

kes may be used with machinery running constantly in one direc- orce absorbed by the brake is $(T - S)$ and ratio of T to S is given Brake should be so arranged that the greater tension T will come then S is the pull exerted on other end in applying the brake. If ns are reversed, the maximum stress comes on movable end and

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the brake tends to release itself. The bands blocks of maple to increase friction. Const absorb too much power per unit of area, or destroy the blocks. A satisfactory rule is to surface for each h p to be absorbed.

Automatic regulators are advantageous for cont save brakeman's wages, and the tramway runs smoot speed. (On power-driven lines, the governor on The surplus power of a gravity tramway may b only a small amount of power is generated, a bra more than the maximum power, thus preventing t condition, a motor being connected to drive the lin the power and extra wear must be cheaper than l radial blades, may be rotated by the controllin This offers a resistance which increases with sp much power. 3. Hydraulic regulator, consisting controlling machinery, may be made to pump a flu and thence back to reservoir. The throttle is so that its opening decreases and resistance inc absorbed from the fluid by circulating cold water device is very effective in absorbing surplus power machines are made to absorb 20, 30 or 50 h p a there is other machinery running at uniform spe connected to it. Its speed will then be controlled erated by tramway it will be turned into the m generated by a tramway may be utilized to pump be used or wasted.

Arrangement of controlling machinery circumstances, since traction rope can be after carriers have been detached.

Brakes may be operated by levers, handwheel two. Levers are quicker to operate, but are not The operating stand may be on the terminal floo brakes are operated by levers, the latter can be ex

Power-driven tramway. When using less th shaft is conveniently made by bevel gears plac With a gear ratio of 1 : 5, the horizontal shaft m motor or engine. When power exceeds 20 h p, to This is avoided by bolting main sheave to a s mounted on a parallel vertical shaft, the latter b a horizontal shaft. On heavy drives the she Bevel gearing is avoided by deflecting traction ro causing it to pass under main driving sheave mou bolted to a spur gear, driven by a pinion on a par is belted. As the ropes tend to lift main sheave pressure of the shaft in its bearings is less than by

Gravity lines. Where power is absorbed by a purpose, connection to main sheave is made as for

Tension sheave at lower end usually slides in (Fig 33), or the rope may be deflected so as to m any other direction.

16. Terminal S

Purposes. Every terminal station must b 1. To secure track cables. 2. To operate t riers from cables to rail, lead them to loadin them to cable.

s are secured in same manner as at anchorage and tension stations th changes in details to suit conditions.

on rope is operated by power drive, or controlled by brake system, usually the upper, and is tightened by passing over a sliding sheave, ed at lower end.

power machin- ghtener are at Type and ar- f controlling and hanism depend ditions, and are nsiderable modi- 14 and 15).

deflecting and on rope in termi- ed by the process rope into or out d of attaching or iers. For a grip DE, the rope must hrough a point be opposite the the instant that nto the vertical

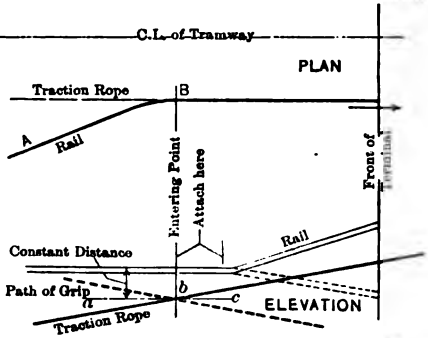

Fig 30. Attaching Point for Grips Opening Sidewise (Dotted lines show alternative position of rail and traction rope)

the rope, as the carrier moves toward rope in running along rail. Thus, in rrier moves along rail from *A* to *B*, the grip travels in plane *abc*, and when *B*, the traction rope must be at *b* so as to enter the grip, which is then closed, e carrier supports the rope. In detaching, these operations are reversed. OPENING DOWNWARD, the carrier moves so as to bring grip into plane of ewhat above it, then advances toward inclined rope, or is lowered onto orizontal, until it bears in the grip when jaws are closed. Thus, in Fig 31,

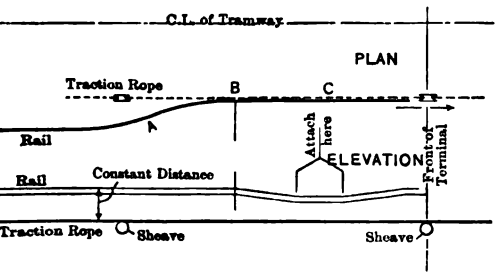

Fig 31. Attaching Point for Grips Opening Downward

from *A* to *B*, with grip above rope, then advances until point *C* is reached. l rest in jaws of grip and it can be closed. In detaching, the process is re- ip jaws OPEN UPWARD, the grip will pass under the rope before taking it.

er of carriers, from cables to rail, or vice versa, must be done as possible, to prevent lunging of carriers and hammering of cable r leaves the rail. Experience shows that wear on cable is least s run on to cable outside the terminal saddle, where cable is free nen load comes on it. This necessitates a terminal point movable When carriers take the cable at a rigidly supported point, the mered and the wires are displaced and worn.

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materials, carriers are filled from bin chutes, and at into which they are dumped.

Fig 32 and 33 show typical arrangements of c tramways with grips opening at side, and illustra traction rope from grip. Fig 34 shows a terminal f the horizontal curves are not necessary for gripping carriers to clear the center posts used to support th

17. Junction Sta

These occur where two divisions of a long tr two terminals placed back to back. An ar lines of the two divisions without increasing c

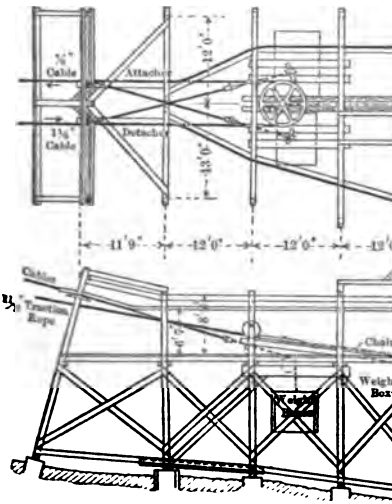

Fig 35. Junction S

slight complication of design. Fig 35 shows which contains the driving machinery for one ior traction rope of the other division of tram sion are anchored, those from the other being

18. Cost of Equipment

Cost of equipment comprises: 1. First c distributing point at the tramway. 3. Co [illegible] distributing point. 4 Cost of c [illegible] material along the line. 5. Cost o [illegible] machinery, including cables, sa [illegible] metal work for a trawway wit [illegible] steel construction. The l

t of machinery for a double-rope tramway, for average conditions and nent, is approximately as follows:

Table 7. Price and Weight of Machinery

	Tons per hr	Specifications for line				
		Cables, diam in	Carriers, capacity cu ft	Spacing, ft	Price per ft	Wt per ft, lb
ery, includ- equipment, es and carers taken as line speed, in	10	1 and ⅞	5	750	$1.00	8.0
	20	1⅛ and ⅞	6	450	1.25	10.0
	50	1⅜ and ⅞	12	360	1.50	12.0
	100	1⅝ and 1	20	300	2.00	16.0

	Price	Wt, lb
or two terminals, including all metal-work and no motor or automatic regulator	$2 400 to 3 600	24 000 to 36 000
station machinery	1 000	10 000

y R R, boat or team, can be estimated from above weights and the pre-

nd foundation materials are usually obtained locally. Timber needed: 12 in wide; timbers from 3 by 6 to 10 by 10 in; a few beams and struts, d intermediate stations, from 10 by 12 to 12 by 16 in. No timber need ng. Foundations should be of masonry. Quantities of materials are ap- follows:

Table 8. Quantities of Materials in Structures

	Timber, ft B M	Foundations, cu yd
inal, no bins	10 000	50
	8 000	10
minal, no bins	15 000	50
	80 per ft of height	10 to 15
eight, 30 ft	2 400	12
station:		
nchorage station	8 000	25
ension station	16 000	45
e and tension station	12 000	35

litions require an intermediate station every 4 000 ft, which may be ain same material as an anchorage and tension station.

stribution is a local matter. Weight of machinery and quantity of timber . Length of haul to get materials to destinations depends on topography, with local charge per ton mile for hauling, form basis for estimate.

aming and erecting of timber depends on local rate of wages and quality e. A very general figure for framing and erecting is $20 per 1 000 ft B M ins, and intermediate stations and $25 per 1 000 ft B M for towers. cluding excavation, can often be built for $6 per cu yd.

stallation varies greatly with conditions. Rough estimate is: for heavy er lb; light cables and traction rope, 2.5 ct per lb; machinery, 0.75 ct gures probably include cost of distribution of machinery along the line

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On a 4-mile tramway in Tennessee, erected 190 of the line were accessible for teams, and the dis costs were:

Material	Unloading from R R a hauling
Cables	$2.76
Traction rope	Unloade pulled thro and installe
Machinery	$1.02

In general, the cost of timber, framing and er chinery, on lines carrying 40 tons per hr and over, on lines of less than 40 tons per hr, from ¾ to on tions under which tramways are built are so varia conditions; hence all the above prices must be conditions, and may easily differ 25% from a quo

Cost of operating consists of labor, re a sinking fund allowance.

Operating labor for a line carrying 50 ton man, one laborer at each terminal and an over 2 miles long. On heavier lines, 2 labore and the oiler will give his full time. The oil oiling bearings, inspecting machinery, tighte repairs; when his whole time is given to the man. Automatic regulators, or a power d light lines, the loader may act as brakema dumped automatically and all labor at disch

Repairs (S. S. Webber, *Cassier's Mag.* D be light, due mostly to accidents; after 3 axles and grips, will show wear, and renew first cost of machinery; after 5 years, repair

19. Clip Tran

Essentials. In this variety of double-r manently attached to traction rope by clip the grip type, as the carriers are not released terminal sheaves without stopping, being lo while in motion. In interval between carri with enough material for one bucket, delive passes, by traveling with it for a few feet. Th at discharge point, and dump through botto

Some clips are loose, and are clamped to rope by to stop the carrier for loading. These are object through the clip while carrier is stationary, and b

Advantages and limitations. Clip tramways per hr, but they have many small parts subject t mechanical difficulties for heavy tonnage lines. grip type. As only one man is needed at loading t the operating cost is low. They are practically miscellaneous freight can not be loaded or unlo Loops can not be used at either terminal.

SIBLE AND SINGLE-ROPE TRAMWAYS

20. Reversible Tramways

on. These, sometimes called "jig-backs," consist of : 1. One or bles stretched at required tension from end to end. 2. Traction ving carriers. 3. One carrier on each track cable, moved to and on rope. 4. Station at each end for operating the machinery, and mping carriers. 5. Sometimes intermediate towers are used to k cable and traction rope. Towers are disadvantageous, as they to 1 000 ft per min, while without them carriers may be run at min. In operation, a carrier is filled at loading terminal, and runs terminal where it is dumped automatically. The machinery is l and carrier runs back on same cable. One man fills the car- rates the machinery.

n. Cost of erection and operation is low for small tonnages and es, but, owing to the reciprocating movement, limit of capacity is . Roughly, this limit occurs, with a 2-cable system, when product er hr multiplied by distance in feet = 20 000; that is, a tramway can handle 20 tons per hr, but one 2 000 ft long can handle only r. With a single track-cable, capacity is less than half the above.

es are seldom weighted, but are tautened by turnbuckles or wire-rope cable tension is judged by deflection of empty cable at center of span.

hen there are 2 track cables with 1 carrier on each, are clamped to trac hat one will be at loading terminal when the other is at discharge termi- is loading while the other is discharging. When loads are descending l often operate by gravity, like a gravity inclined plane. A reversible be operated with 1 track cable and 1 carrier. This is practically half a and always requires power.

rsible tramway is often used to transfer freight or passengers across a m. If there are no intermediate towers, the carrier hangers can be ex rom both sides of carriage, thus preventing carriers from jumping off each end, carriers stop just before terminal saddles are reached, for load- ging.

21. Design of Reversible Tramways

vs profile of a 1 000-ft tramway, to carry 20 tons per hr, at average ft per min. Allowing ¼ min for loading, a load will be delivered

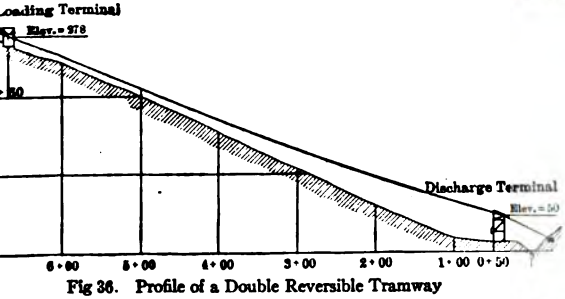

Fig 36. Profile of a Double Reversible Tramway

n, or 40 loads per hr. Bucket is of 12 cu ft capacity, holding his bucket (Table 9) weighs 500 lb, making gross weight = 1 5

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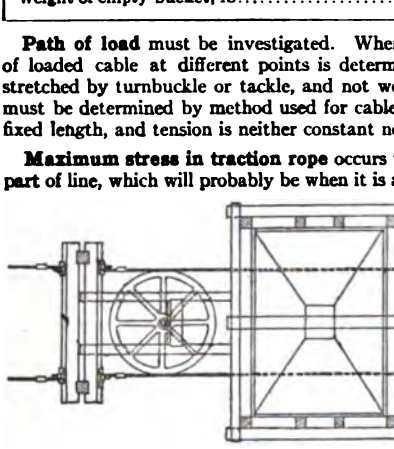
Size of cable is proportioned to load as in Art
ersing cable per day is less than with a contin
per sq in sectional area of cable may be allowe
loads 25% larger than given in Table 5, hence
here.

Table 9. Weight of Buckets for

Capacity of bucket, cu ft	
Weight of empty bucket, lb	

Path of load must be investigated. Whe
of loaded cable at different points is determ
stretched by turnbuckle or tackle, and not w
must be determined by method used for cable
fixed length, and tension is neither constant n

Maximum stress in traction rope occurs
part of line, which will probably be when it is



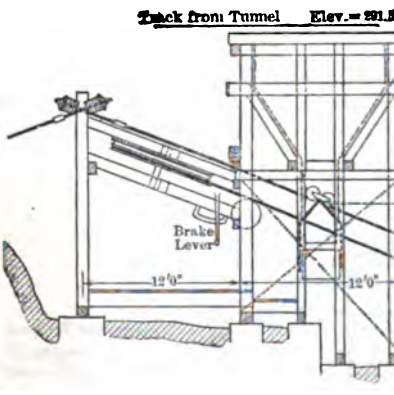


Fig 37. Loading Terminal, Double

by: Tension = (*wt loaded carrier* × *sine incli*
which wV comes from Eq 27, and ½ tw = h
sheave. Wear of rope results chiefly from i

e diameter, the wear will be mostly on surface, and a rope of 7-wire y be used; but if sheaves must be small, a 19-wire rope is better.

y lines, relation of operating force to resistance must be investigated at trip, especial attention being given to the conditions at discharge end, for pe of cable is flat and little power is generated, while the other carrier is pest part of its track and offers maximum resistance. Momentum of car- mplete the trip after loaded carrier has ceased to generate power, but this nted on with caution, as it requires a skillful operator to run loaded carrier t speed to take it into its terminal and stop without undue shock. On or with a heavy ck cable, the low- ath of carrier may able distance from terminal, and mo- arrier may not be complete the trip. cumstances, this way should not be power should be complete the trip

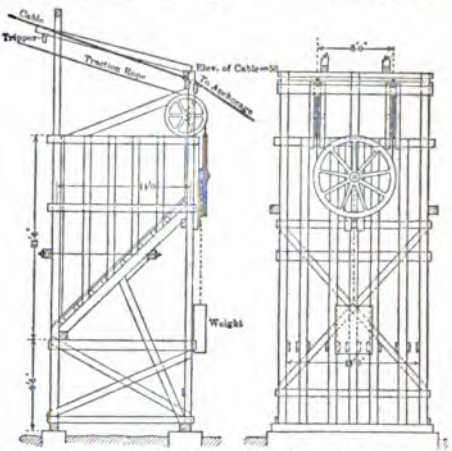


Fig 38. Discharge Terminal, Double Reversible Tramway

ng machinery either brakes or ve. The drive le for stopping ning in either Shifting belts satisfactory for ng mechanism. t wear rapidly, and easily re- nd brakes may with bands ap- osite directions, so that one will be efficient, whatever the direction and the most advantageous one will be used for main control, while left set throughout run. Power calculations are analogous to those Fig 37 and 38 give important features and dimensions of typical r this type of tramway.

22. Single-rope Tramways (or Ropeways)

nsist of: 1, single endless rope, running around sheaves at each d over sheaves at towers; 2, carriers or buckets suspended at in- n running rope by clips; 3, terminal machinery for driving the r loading and unloading carriers.

while in motion, are filled from a measuring hopper by mechanical e attendant is required. At the discharge end, buckets are tripped their loads automatically. This loading and dumping, without e buckets, limits this type of tramway to carrying ore or similar al. Fig 39 shows the profile of a short line, and Fig 40 the general of the standard terminal, and how it is used at either loading or nd of line.

on. Single-rope tramways give good service up to capacity of 15 , a length of 2 miles, and a difference in elevation of 2 500 ft. For acity, length, or inclination, running rope becomes heavy and

excessive. The limited field for this type of tramway has result almost dropped by manufacturers in the U S, although for a fi cheaper, in both first cost and operation, than a double-rope tr

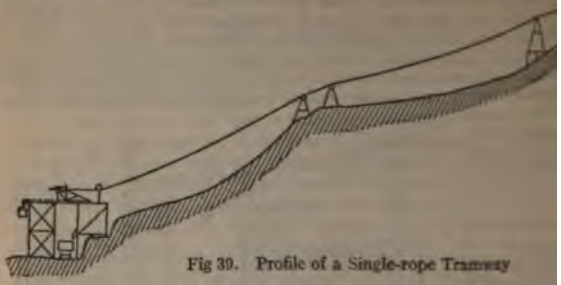


Fig 39. Profile of a Single-rope Tramway

Roe system (made by 6, Art 27) belongs to this type of tramway, but supported by a "box head" which rests on running rope. This limits

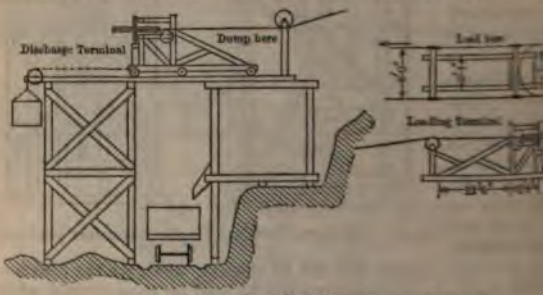


Fig 40. Terminals, Single-rope Tramway

line can be operated, but allows more flexibility at terminals, because removed from running rope and can be taken to any point by a rail sy or discharging at rest.

CABLEWAYS

23. Description

Essentials. A cableway consists of: 1, fixed cable; 2, carri the cable; 3, hauling rope to move carriage back and forth al 4, hoisting or fall rope to raise or lower the load by pulleys s

Fig 41. Diagram of a Small Cableway, Canon Type

wo towers to support cable at necessary height; 6, hoister at one ting hoisting and hauling ropes. Fig 41 to 43 show usual ar- plant. CARRIAGE has 2 to 4 wheels, depending on weight of load, oisting sheaves to match those in the fall block. FALL BLOCK nber of sheaves to give necessary purchase, and terminates in a is attached a carrier suited to material to be handled. HOIST- OPE is usually cast steel, ⅝ to ¾ in diam, with 6 strands of 19 AULING ROPE may be the same, or it may have 7 wires per strand.

rriers. Tension in fall rope is sometimes small, and provision to prevent rope from sagging between carriage and head tower. t span cableways (Fig 41, made by 9, Art 27), sag may be pre- ng weights to the fall block, but this increases dead weight to

Fig 42. Diagram of a Flory Cableway

large cableways, fall rope carriers are used for this purpose. buted along cable between carriage and head tower, and are y chains hanging in loops (Fig 42, made by 8, Art 27), or by rent sizes on a button rope, which pull carriers off the carriage another (Fig 43, made by 7, Art 27). The buttons are graded

Fig 43. Diagram of a Lidgerwood Cableway

t the first and smallest pulls off outer carrier, the second and utton the second carrier and so on. On return trip, the carriage ers on a horn at its forward end. The button rope is about same oisting rope and has 7 wires to the strand.

bulk material may be a skip or a bucket. Massive objects may directly from hook of fall block by slings. Bottom-dump buckets handling concrete or mortar in masonry construction; skips for pping, and general excavation work. Skips are of wood or steel, preferable. They are suspended by 2 chains permanently con- r corners, and a third attached to a knock-off hook at middle of In dumping, the knock-off hook is tripped by a man, load dis skip returned to loading point, swinging from the rear chains. oaded in the pit while another is being hoisted.

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Automatic dumps are sometimes arranged to trip
cases where material is dumped into a waste pile, and
tant. The dumping may be done by a clamp on hois
raising load until the trip engages an arm extending do
running from skip back to a special drum on hoiste
arrangement lifts rear end of skip and discharges con
at any elevation; first two arrangements necessitate d

Main cable is usually "patent locked rope
end is held in a step socket (Fig 45), having

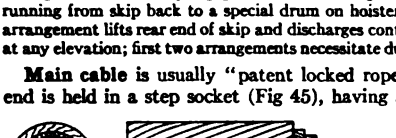

Fig 44. Two-in Patent Locked Rope

tower. At first only a part of take-up tackle is used
and tension has increased, the remaining sheaves a
final position. The tackle is then clamped and any
used is removed. The take-up tackle must be strong enough for full working tension of main cable.

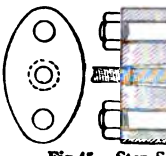

Fig 45. Step S

Towers usually have rectangular bases, set with their long axes parallel to center line of cableway, and tapering to a small square top (Fig 42). They are usually of woo
50 to 80 ft high. Height of tower depends on
be raised, and elevation of ground where tower
on record is at Tunkahannock viaduct on D L

Towers may be stationary or mounted on a
tower can be moved along the bank at right an
may be movable on parallel tracks (Fig 43), o
other movable along a curved track, with the
Thus one cableway may be used to excavate a
for constructing parallel sections of a structure

Shape of tower depends on direction of forces a
chored in the ground, the resultant may be nearly v
cal, but where towers are movable and cables are
and at base of tower, the front legs are steeply incli
base. A simpler design is used on light portable cal
towers are single bents held upright by guy lines (Fi

In designing towers, EFFECT OF WIND at right an
into account. It acts on tower and on half the leng
tower, and on cable from tower to center of span.
ropes are small in themselves, there are several of t
combined area is considerable, and as point of app
tower, a strong overturning moment is exerted.

Hoister for operating the cableway may
steam engine or electric motor. The latter is i
designed for either alternating or direct current
one plain drum for operating fall rope and an ell
8 in wide for the hauling rope which moves car
dumped by engineer, an extra drum must be prov

24. Design of Cableways

quired are: 1, plan and cross-sections, or contours, of volume to be the sections showing probable maximum depth of excavation; and elevation of discharge points; 3, kind, size, and weight per r cu ft, of material to be handled; 4, quantity to be handled in tons nature of power to be used.

load is sometimes governed by size of individual pieces or units. In strip- ally determined by quantity of loose material to be handled per hr, although gle blocks of stone may be the controlling factor. Net load equals tonnage d by number of loads that can be handled per hr.

load is made up of time to hoist, transport load, dump it, return the skip to ge connections to another load. Speed of hoisting and lowering may be as per min. Carriage can be hauled along cable at 1 000 to 1 500 ft per min, with l fall rope carriers, and at about 800 ft per min, with chain-spaced fall rope

nce time for one load $= \frac{2 \times \text{depth of pit}}{\text{hoisting speed}} + \frac{2 \times \text{distance hauled}}{\text{hauling speed}} + \frac{1}{2}$ min for dumping. Time to lower may be a little less than that required for hoisting. stance hauled may be taken as 80% of span; it can rarely exceed 90%, and is be over 70% for any length of time. Round trips often average 2 or 3 min.

able depends on load, span, and allowable deflection. In beginning is usual to allow a deflection of 0.05 of span, when load is at center, ition is covered by Eq 14. If size of cable be given, all quantities ember are known, and equation can be solved at once for deflection. is determined in terms of span, and size of cable is to be found, then unknown; but, from Table 2, t is about 7 000 times w, for safety factor t or w can be expressed in terms of the other quantity and equation iat quantity. It is usual, however, to substitute a value of w which is ely that of some similar installation, or that corresponds to typical is (Table 10 and 11). If this deflection seems reasonable, it is adopted, of towers are made to conform. If resulting heights of towers are too ger cable may be used and the increased tension then allowable will ction. Table 2 gives working tension for safety factor of 5, but if danger of cable being overloaded, the factor is often reduced to 4, ll deflection is desirable. Deflection of 0.05 of span was assumed for study only; on long spans deflection increases faster than span, due cable, hence deflection is often more than 0.05 of span; on short spans ay be taut and deflection less than 0.05 of span.

ad. Tension is maximum when load is at lowest point, and decreases as ither way. This variation in tension is unknown, hence the equations of ection, which involve tension, can not be used for determining deflection with us points. Path of load may be found approximately by assuming that the

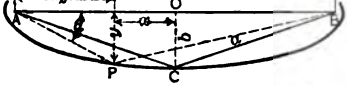

Fig 46. Path of Load on Cableway

A and B (Fig 46) are the foci Then lines $AC + CB$ equal s of ellipse, and the ellipse can be at center of span, and OC due to wt of cable and concen- center of span, then $OC = H$ semi-minor axis, while $CB =$ or axis of ellipse. The ordinates of any point P are x and y. The equation referred to its center is:

$$a^2y^2 + b^2x^2 = a^2b^2 \quad \text{or} \quad y^2 = b^2(a^2 - x^2) \div a^2 \tag{39}$$

d, from Fig 46, $a^2 = H^2 + (s^2 \div 4)$; hence, by substituting a series of values ponding values of y can be calculated and path of load determined for a ints.

If s be distance of load from end of span, the defl
gives slope of chord AP when load is near tower, ar
From last expression, or from path of load, it is se
load approaches either tower, that for a distance
end it is not feasible to run the carriage.

This method of finding path of loaded point a
AP and PB is constant for all positions of load,
proaches tower, the long section of cable will sag
position indicated by Eq 39. The deflection is fur
and hauling ropes, which take a varying part of le
part of cable. These errors are all on safe side,
to make sure that the load will clear some object.

25. Typical Cabl

The largest loads that have been carried a
20 tons on a single cable, on twin cableway of

Table 10. Typical Sp

Span, ft	Load, tons	Cable diam, in	Operating rope	Hoister, size cyls, in	Maker (Art 27)
800	1	1.5	⅝	8½×10	7
300	2	1.5	⅝	8½×10	3
950	5	2	¾	10×12	3
500	5	2.25	⅝	50 h p (Elec)	8
700	8, or 5 cu yd rock	2.25	....	10×12	7
1 200	4	2.25	⅝	110 h p (Elec)	8
1 350	7	2.5	¾	10×12	7
900	7	2.25	¾	11×14	8
800	6 at hook = 68 cu ft concrete in tub	2.25	....	150 h p (Elec)	7
1 420	10 to 15	....	....	300 h p (Elec)	7
1 500	12	2.25	....	12×12 and 12¼×15	7

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span, 1 140 ft. Several cableways have been built to carry 15-
Ordinary loads in quarrying are 3 and 6 tons; these are also common
construction, and correspond to 1 and 2 cu yd of concrete respec-
r heavy masonry, 10-ton blocks of stone are often carried.

ans thus far built are: 2 180 ft, for 2-ton loads, Hamilton Coal Co, Tarentum,
o ft, for 6-ton loads, Hudson River Power Co, Spier Falls, N Y. Spans of
ft are common for open-pit work, and 1 200 to 1 500 ft for big dam work.
aximum load and maximum span are not possible on same cableway. The
ed when the span in ft multiplied by load in tons is approximately 15 000,
have been built in which this product is between 15 000 and 21 000.

Table 11. General Specifications for Cableways

ns	Cables, diam in	Operating ropes, diam in	Engine, 2 cyls, in	Carriage and fall block, approx wt, lb
	1.25	½	8×10	600
	1.5	⅝	8×10	1 000
	1.75	⅝	10×12	1 600
	2	¾	10×12	2 000
	2.25	⅞	12×15	2 300
	2.5	⅞	12×15	2 500

operate consists of: 1, engineman at hoister; 2, fireman, when an inde-
ler is used; 3, signalman, to give instructions to engineman for moving
ll block; 4, dumpman, to trip the carriers; 5, one or more hookmen in
tach fall block to the load and to give signal to hoist; 6, rigger or oiler.

not needed if there is a central steam plant, or for an electric hoister, but
power will partly offset his wages. Signalman usually stands in a balcony
ge, in view of both loading and dumping points; he communicates with
signals or telephone, giving orders in accord with signals of men at hook-
mping points. Sometimes the engine house can be located so that engine-
nto the pit, and receive signals direct. Dumpman is needed when dumping
or car, but not for automatic dumping. Hookman must be reliable, to
d securely, and start it properly from the bottom; he also sees that other
out of the way of the swinging carrier. Rigger inspects machinery, and oils
fall-block sheaves and tower-sheave bearings, at beginning and middle of
ring installation or repairs he has charge of the work.
he skips is chargeable to excavation, and not to cableway. It is often done
por; on Chicago Drainage Canal, 12 to 16 cu yd of rock were loaded per
r day. Loading is sometimes done with steam shovel or drag scraper.
e pit may serve to move the skips from loading point to a place where
icked up by the cableway.

26. Light Cableways

leways are used for trench work on sewers, subways, and similar
, shallow excavations (Fig 41). Towers are single A-bents, 25 to
nd the whole plant is designed for portability. After an excava-
n made for the length covered by one span, plant is moved ahead
section excavated. Spans are about 300 ft long and fall rope
usually dispensed with, as the empty bucket will keep hoisting
But if loads are being hoisted near center of span and are being
ower, a single fall rope carrier may be run out on track cable and
a chain dropped to the ground. It is set where it will not inter-
rriage, but where it will hold up center of hoisting rope when car-
ne tail tower. (Made by 9, Art 27.)

. Machinery for a light cableway for a net load of 2½ tons on a 300-ft
of: 1⅛-in main cable, of cast steel with 19 wires, with anchorage and take-

up fittings; 5/8-in hoisting and traversing ropes; wi
and 16-in sheaves; fall block with 16-in sheave;
dumping and self-righting tubs, 48 in wide by 45 in
hoist, cylinders 8¼-in diam by 10-in stroke, drum
drum 25-in diam by 26 in concave face; vertical tub
engine car with 10 by 16-ft timber platform, hou
wheels; 180 ft of 25-lb T rail for moving car; 2 tow
with 1-in galvanized steel guy ropes.

Price of machinery for a 300-ft span, to
for the tower and engine car, f o b Boston, is
are 40 ft high, cables and ropes are longer, and

Capacity. These plants will hoist a net loa
per min, and convey it at 450 ft per min. I
trench, from 30 to 40 loads can be handled pe

Operating cost. Labor to operate: 1 engine
dumpmen when loading wagons or dumping into
buckets when machine is back-filling rear end o

On a sewer in Washington, D C, in 1894, the co
day was $1.50; 30 laborers were used to fill the b
handled per hour and 280 cu yd were excavated in

Moving of such cableways can be done in 2 c
provided the new anchorages, consisting of a d
vance. Force required: 5 laborers at $2, one man
say $2, total, $15 per day for moving.

27. Manufacturers of Tramway

1. Adolf Bleichert & Co, Leipzig-Gohlis, Germany
2. J. Pohlig, Cologne, Germany
3. American Steel & Wire Co (formerly Trenton
4. Broderick & Bascom Rope Co, St Louis, Misso
5. A. Leschen & Sons Rope Co, St Louis, Missou
6. Ropeways Limited, London, England
7. Lidgerwood Manufacturing Co, N Y City
8. S. Flory Manufacturing Co, Bangor, Pennsylva
9. Carson Trench Machine Co, Boston, Massachu

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   Soc Indus de l'Est, 1896. Pub. A. Nicholl
   mathematics, 40 pp descriptions of tramway
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   N Y, 1905 ($10). Chap 16, of 76 pp, is on t
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   & Min, Feb, 1908; *E & M Jour*, Dec 16, 18
   & Eng World, Sept 7, 1912; *Eng News*, Mch
   July 24, 1902, Feb 11, 1904; *Eng Rec*, Oct 26
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   1903, May 12, 1904, Feb 21, 1907, Feb 16, 19

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SECTION 27

MECHANICAL CONVEYERS

BY

LINCOLN DEG. MOSS

NT PROFESSOR OF MECHANICAL ENGINEERING, COLUMBIA UNIVERSITY

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nd Flight Conveyers	
ixed Troughs.....	1599
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or wire-rope tramways, see Sec 26.

1. Chain and Flight Conveyers

stics. SCRAPER CONVEYERS (Fig 1) are cheapest, but least efficient and ough must have inclined sides, to center the hauling chain. Sides of flights of the trough. Bottom edge of flight should be above bottom of trough ar. SUSPENDED FLIGHTS (Fig 2) do not touch sides or bottom of trough, but

Chain Flight

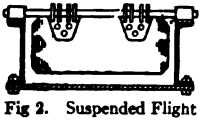

Fig 2. Suspended Flight

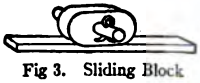

Fig 3. Sliding Block

locks (Fig 3) riding on upper edges of trough. Duplex chains are preferable. s are best made to capsize on meeting an obstruction on the track. ROLLER s wheels instead of sliding blocks. Wheels may be cored and packed with l.

re usually of sheet steel. For heavy service, C-I flights with rein- are better.

are preferably of sheet steel, bolted, with joggled joints. Rolled- ions are most economical for the sides. C-I bottom plates are used for long life. Ample clearance between bottom and flights permits a s to accumulate and reduces wear on plates. Renewable tracks are eferably of flat bars, tap-bolted from below so that bolt will hold to

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thread (Fig 4); countersunk bolts will let go
troughs lined with steel sheets may be used f

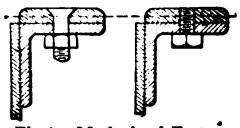


Fig 4. Methods of Fastening Track

Discharge is t
These gates are o
and fines accumu
and rack is desira
frequently, rust
discharge on the
flight must be us

Chains are most troublesome at the pins and bea
as speed falls. Tension may be abnormally high w
this is the prime consideration. Bushings are d
saving chain which otherwise would have to be sc
cause of oval holes. Steel tubing, C-I or brass bu
be used; brass is best. Closed joints are essential
grit, and are always desirable. Malleable-iron cha
within the limits set by makers, are best for gene
Drop-forged bushed steel links of eye-bar pattern ar
est and strongest. PIN PRESSURES in some succes
are 8 000 lb per sq in. Sliding friction takes place onl
and sprockets. A lower unit press will reduce tim
expense of renewing bushings and pins. Long-link
subject to pulsation than short-link, due to the ch
Compensating multi-lobular wheels are made by sor
originally made by Link-Belt Mach'y Co. The H
used for the same purpose. Short-pitch chains are
temporary use, and where abrasive grit is absent w
for hauling and driving gives good service at low cos
a closed eye. Pins should lock in the thin outer lin

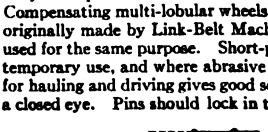
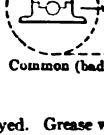


Fig 6. Sprockets

Sprockets a
longer service.
rately with ch
used. Cast-st
well as those o

Shafts and
can not be nech
tion loss and cost of bearings; turning down cuts th
ing the shaft to spring after removal from the lathe
not exceed 8 000 lb per sq in, to provi le for max con
For steady service, the bearing press should not exc
metal. Bearings should be set more nearly normal to the resultant load than in common practice, to retain grease and prevent loosening of bolts. Grease lubricant is best when there is grit to abrade, or water to float out oil; also when machines are occasionally out of service. Oil will drain and float out. Grease is also preferable where cheap labor is employed. Grease



Common (bad

forms a collar which excludes grit and water. Cap
ings larger than 10-in, 2 cups are preferable. Ball-a
elastic structures, but it is better to build stiff, rig

Speeds range from 60 to 100 ft per min.

Capacity ranges from 30 to 300 tons per h

Materials handled. Flight conveyers are
load in sliding through the trough. Thus, for

onveyer is needed. For 100 tons bituminous coal, which has higher
, 7 to 8 hp is required. Some kinds of sand can not be handled
of excessive friction. Ashes cause excessive corrosion, which be-
gh joints. Power consumption per ton-mile on a level ranges from
r. Advantages in cost, wt, friction loss, and durability always rest
r machines and higher tonnages. This is true of all mechanical
he belt and the moving-pan conveyers are therefore displacing the
.

o 30° are feasible, with directly proportionate reduction in carrying
at 30° due to imperfect filling. Power for up grades is readily
he formula: hp = tons per hr × ft lifted ÷ 990; or, divide by 1 000
Add to this net hp a friction loss in the drive of 0.15 to 0.30, de-
ount and quality of gearing in the drive.

est results, should be set on rigid frames, with rigid pillow blocks. The
om motor should be (*a*) cut gears, (*b*) silent-toothed chain. Involute teeth
drives usually get inferior erection, and wear causes pitch circles to lose
t-tooth gears, with 20° obliquity, have given good results, and permit a
rrangement, due to superior strength of the teeth. The standard involute
o 15° obliquity, will wear longer, and produce less noise. For durability,
el pinion with a cast-steel gear, and a cast-steel pinion with a C-I gear.
eferable to molded gears, being stronger, more
s noisy.

Chain and Bucket Conveyers

pans form a continuous moving trough,
the heavy friction loss of hauling material
l trough. Buckets are bolted to the chains,
the head sprockets by inverting (Fig 8).
aterial determines the shape of buckets.
be sloped back to permit buckets to pass
kets; for sticky materials, all sides are
mediate delivery is occasionally effected
trays at their ends, attaching guide wheels at the other ends, which
depressed track to spill the load (Fig 9).

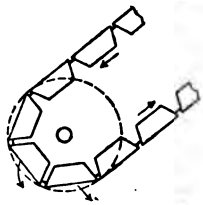


Fig 8. Bucket or Pan Conveyer

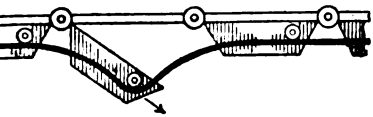


Fig 9. Mode of Dumping

The dumping ends of trays must be open, and, owing to shock of dumping, the speed reduced to about 50 ft per min. With fixed pans speed may be 80 to 100 ft per min.

h closed and bushed bearings are preferred. Roller chains only
ployed (Art 1).

up to 1 000 tons of stone per hr may be had.

handled: earth (if not too sticky), rock, ore, coal, ashes, molten iron
chines (using smoked or lime-sprayed buckets), stones, and gravel.

ired is from 2 to 4 hp per hr per ton-mile on level track. Best re-
large conveyers and heavy materials. Clean tracks and large,
d rollers in the chains are essential. Conveyers on steep grades
y, because a large part of dead wt pulls on the head-shaft bearings.

y be up to 30°. Inclines do not cause decrease in capacity, as in
rs. To equate for grades, see Art 1.

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Variations. A flat deck may be used, with slots curved arms for barrels, or with a steel beaded a

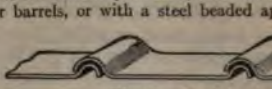

Fig 10. Steel Beade

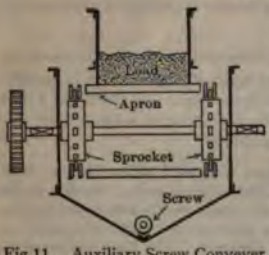

Fig 11. Auxiliary Screw Conveyer

(Fig 10). Ligh shape under h Some fines will run, which may veyer (Fig 11). in a trough (Fig with vertical p conveyer. This used in small po to deliver coal head bunkers, up ashes on the l elevating them t head bin.

3. Chain and Tilting-bucket Convey

This machine, as designed by C. W. Hunt, i provement on the form shown in Fig 12, in h troughs. The buckets normally hang upright, and for dumping can be capsized at will.

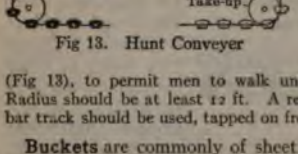

Fig 13. Hunt Conveyer

Chains bearing are slow, 40 to flanged, and yd, dependi be chilled C used, and i loaded on t attached to

Curves 6 ft diam. (Fig 13), to permit men to walk under in safety Radius should be at least 12 ft. A renewable stee bar track should be used, tapped on from under sid

Buckets are commonly of sheet steel, No gage. For hot or heavy abrasive or corrosiv substances, malleable C I is superior, but mo expensive, and requires more power. Dumpin cams are attached to the sides of bucket. Variou methods are used to prevent fines from falling b (*a*) close contact; (*b*) buckets separated, bu filler; (*c*) overlapping lips, using a skirt-board offset suspension. All retain their normal gra must be mechanically reversed once on each entering the rising run in wrong sequence. Bu carrying corrosive materials.

Fillers are required with *a* and *b*, Fig 15. opened and closed automatically; 2, wheel c to give quick discharge; 3, rotating series of

buckets. With styles *c*, *d*, and *e*, filling is done by loading between with a scraper to prevent overloading.

re C-I frames with inclined ed on 4 wheels and moved at d dumping position on a track. cessary. The shock of dumping o 40 to 60 ft per min.

o to 200 tons per hr.

uilt, the bearing value of pins conveyers to 70 to 80 ft rise, and and proportion of vertical load hain stresses and bearing loads. aring loads carefully for all high

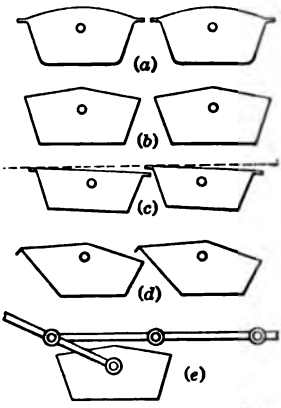

Fig 15. Modes of Preventing Fines from Falling between Buckets

arried: coal, ashes, pebble phos- nent clinker, and small-size ores.

drive. Following formula fits installations, although peculiar s, quality of material, workman- and maintenance cause excep- e at variance. Let T = net tons hr; H = horiz length of loaded h = horiz length of empty conveyer, ft; C = a constant, see below; ft; Hp = motor input.

$$\text{Hp} = \frac{T\,(R + 0.1\,H + 0.05\,h)}{1\,000} \times C$$

lly are mere sprocket wheels. Specjal sprockets with varying radii are used form linear speed and avoid pulsation 8 shows Hunt driver; Fig 17, Link-Belt Driver should be placed where the chain

Values of C for Different Values of T and R

Conveyers	T	R	C
Small.....	30–40	40– 60	2.2
		65– 90	2.4
		90–120	2.6
Medium...	45–60	40– 60	2.0
		65– 90	2.2
		90–120	2.4
Large.....	70–150	40– 60	1.8
		65– 90	2.0
		90–120	2.2

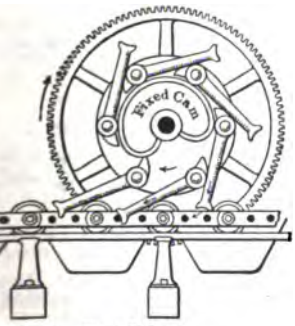

Fig 16. Hunt Driver

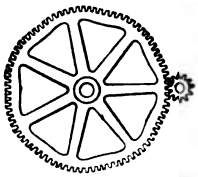

Fig 17. Link-Belt Driver

. It is sometimes more convenient to place it over the vertical rising por- to be farther along on the upper horiz run. TAKE-UP is located below, to corner sheaves on planed C-I rails, in unison.

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of buckets; width or face of drums, 2 in more than
width per ply, should not exceed 22 lb for high-grad
Reduce stress to 20 lb for 28-oz duck (Art 8, Belts).

Drums are pulleys with heads bolted in, to e
COEFF OF FRICTION is ordinarily 0.2 to 0.3 for a
lagging, bolted on, will raise the coeff to 0.35 or
the coeff, others lower it; sulphur dust is the
kept away from belt, as they dissolve rubber.
bucket elevators are that they are simple to in
chain and sprocket troubles in gritty materials.
than chain elevators. ARC OF CONTACT betwee
depending on angle to horiz. Slack side is perm
curve, which is modified by the tension of low
out, give ample clearance for the slack side, wh
foot. The actual curve is more like a parabola

Frame should be well braced for transverse stiffn
beam load. As a column, 90 $(l \div r)$ is a safe limit.
on head shaft and pillow blocks is the sum of the taut
slack tension is usually 0.3 the taut tension, and may
and millwork (Art 1). Put GUIDE IDLERS under tau
under slack side.

Ratio of drum diam to belt thickness is i
the most expensive item in total cost. It is b
5 in diam per ply of belt is advisable for perm

The tendency is to use a smaller, cheaper drum, w
gear reduction. As drums and gears do not wear ou
do, real economy lies with the high ratios. Lower
because driving tension in belt is almost nil at th
remaining. Guide idlers are often too small, do not t

Boots (Art 4).

6. Belt Conveye

In 1893, Thomas Robins introduced the first
cotton belt with idlers of like diam and no slip.

Troughing. Originally, the belt ran flat b
give a large load section. The sag between idl
under the skirt-board at *A*, which were dragged
B, cutting the belt. Rubber belts carry loose

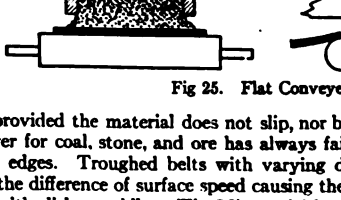
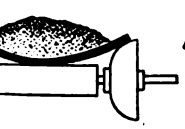


Fig 25. Flat Conveye

steel, provided the material does not slip, nor b
conveyer for coal, stone, and ore has always fa
of belt edges. Troughed belts with varying
grain, the difference of surface speed causing the
belts, with dish-pan idlers (Fig 26) are fairly
free to revolve on the shaft. These are being
more than point contact between belt and outer
faulty design (Fig 27) permits the belt, if well lo
cut; it also steers badly.

ley Robins idler (Fig 28), with all pulleys in one plane, has been very success-
ferior to those having 5 pulleys (Fig 29), which reduce the angle of transverse



Belt with Dish-pan Idlers

Fig 27. Faulty Design of Troughed Belt

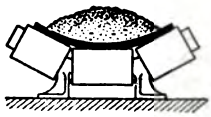


Fig 28. Three-pulley Idler

elt. In the Robins system, the shaft is hollow cold-drawn and stationary, with e lubrication. This is done in a slightly different way by the Link-Belt Mach'y ing perfect protection against entrance of grit. Oil lubrication requires less n idlers, but no oil drops must be ach the returning-belt underneath, on carrying side.

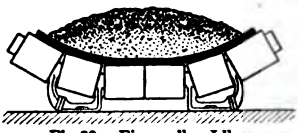


Fig 29. Five-pulley Idler

ng idlers have been used to a small reduce power consumption about ⅓ 4). Dust stops must be carefully o exclude grit; fine mineral dust ise speedily ruin the bearings.

nearly always of canvas and rubber, the canvas weighing 28 to 32 oz, cotton. The "friction," of rubber gum cement between plies, must uality to prevent separation of the plies. Testing a 1-in strip, cut by al knife-cut (wet knife with water) and peeled loose at the top, a pull of 16 to 20 lb should shear the strip, if of best quality, 10 to 14 lb and 7 to 8 lb for inferior. The carrying cover is usually $\frac{3}{16}$ in thick; gh for medium service; more than $\frac{3}{16}$ in is rarely used. The other in thick. Edges should be well molded and reinforced. Greatest e center, where the load is greatest and where the first fall from chutes s led to the thick-centered Robins belt (Fig 30), which incidentally increases lateral flexibility, with edge stiffness. An ordinary straight-ply belt will trough under its own wt, if 30 in wide or more. Long belts for a given tonnage are presenting more surface than shorter belts for same tonnage. When ded, belts wear better than if lightly loaded, as the bombardment e is lessened by keeping full load. (See Fig 38, 39.)

ick-centered Robins Belt

PICKING BELTS is about 30 ft per min; for GENERAL SERVICE, 350 o to 500 ft per min is ideal for lump materials. For fine granular er min has been used successfully, but is not advised; at 500 ft, a olves nearly 320 rev per min. Impact at deliveries increases as V^2 ad coming on at chutes also increases with speed.

riving should be of tough grey iron, with ribbed spokes, as the pull may be eccentric. Extra high crowns are desirable to keep belts centered. Con- nder double arms are advisable, also reinforced edges of rims to prevent ny good conveyers have only 3 in of driving pulley diam per ply of belt; ended for long belt life. Pulleys do not wear out, belts do. Tail-pulleys carrying less total tension. The snub or bend pulleys near drive should be s drivers. Troughing idlers are commonly 4, 5, 6, and 7 in diam, depending ortance of conveyer. Since greatest percentage of power lost is in turning n are preferable. Idlers with high rotative speed should be turned and bal- rs are best, resisting corrosion, and are worn through only by belts carrying t coke breeze. GUIDE IDLERS are unnecessary with good erection, and well- y are a safeguard against accidents which might cause belt to leave its trou

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Drivers are best located at the head, but may be
tion. It is often convenient to drive 2 conveyers fro

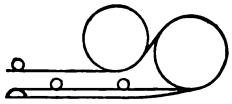


Fig 31. Snubbed Drive Pulley

The load on head s
veyer is doubled.
tensions, that on sl
side; tight take-ups
is generally 0.2 to 0
it to 0.35 to 0.4.
of same diam, will
Tandem drivers a
bending with one reverse. About ⅔ of the work is d
ence in diam will cause slip of belt and wear, unless
tandem cable drives. A tandem drive permits use
a small head pulley, with a shorter connection betwe
2 conveyers in series (Fig 32). But, for long life, th
head pulley should be as large as the drivers. Servic
statistics favor the single-pulley drive, provided th
bend pulley is as large as the driver. Best results
driving are obtained with stiff framing, and rigid pillo
blocks set normal or nearly so to the loads. Avo
angle, the leg of which deflects under load (Fig 33

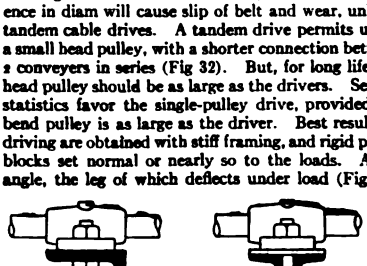


Fig 33. Wrong and Right Support of Pillow-blocks

motors, silent-chain (Link-Belt or Morse) prevents
are very desirable, equated for bending and torsion
necked down at bearings. Bearing press on Babbitt
per sq in. A shaft properly calculated for stiffness,
reduced at journals, has too large a friction radius fo
be saved by using smaller pillow blocks, if not overl

Trippers may be fixed, hand-propelled, po
versible. Scraping the load off by a rubber-ed
only satisfactory method of un-
loading a belt between its head
and tail is to invert belt (Fig 34);
water jets being needed for sticky
material. Adjustable revolving
brushes are the best cleaners.
Sometimes a few fixed trippers and
one end delivery over-head pulley have been us
cry. A flapper gate is needed at each tripper t

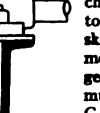


Fig 34.

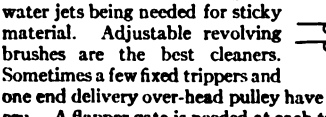


Fig 35. Double Tripper

instead
hopper
on 4 C
ning o
spur g
propell

Track grips are desirable to prevent creeping fro
The frame should be rigidly cross-braced. Pulle
results. They should have fully 4 in diam per ply

reversible, a double tripper is needed, made of 2 standard trippers linked 35).

ould have generous pitch (usually 45°) to give speed of flow. Para-, starting with 60°, would be best (Fig 36). The object is to land on the belt at high speed and in the same prevent slip of the load and wear on belt. teel is best; or wooden chutes with steel edges should be well above the belt, other- ill lodge and cut belt (Fig 37). Let thin roject 1 in below the wood; lumps will derneath the thin edge. With run-of- materials it is desirable to form a cushion on the belt and let lumps fall thereon. Fig 38 shows the simplest will be drawn through the slot between ledges, by the bed of fines; d is a screen with by-pass chute (Fig 39).

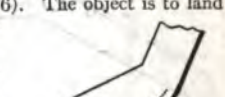


Fig 36. Parabolic Chute

rop between conveyers, the chutes at Frank- J, were made as in Fig 40. Whenever the

rangement of Bottom ge of Chute

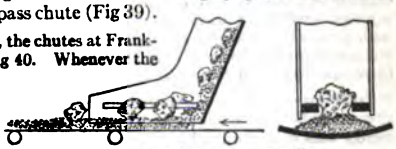
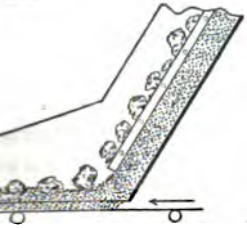


Fig 38. Loading Coarse Stuff on Belt

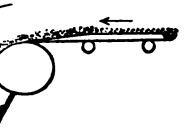
vy or abrasive, or both, it is well to put flat ledges in chutes ators (Fig 41) and trommels (Fig 42). The material banks es and forms a cushion, thus preventing wear on chute.



Screen with By-pass Chute

Fig 40. Franklin Furnace Chutes

hould never deliver across a belt. It will push the belt out of line, ntrically, and fail to deliver in the direction of travel. When a movable unloading machine delivers to a belt underneath, it is well to use an independent traveling hopper towed by unloader. This preserves uniform height of chute above belt. Make sure that chutes do not deliver load upon an idler, but at least 6 in beyond. Belt and idler will not endure pounding.



helf in Elevator Chute

Fig 42. Ore Shelves under Trommel

rves in belt are often necessary, and may be freely used. ge of direction can be permitted in a single conveyer nsverse bracing is always essential. The minimum radius of a concave ver- 250 ft; sometimes more is needed, to make sure that the light belt will lie

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its roller bed, and not lift off under tension; in som other cases, the change is as in Fig 43, convex to t

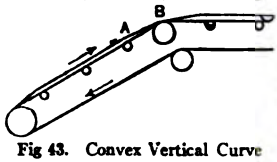


Fig 43. Convex Vertical Curve

bend pulley i 1.74 at *A* to will result. with edges of ing between b veyers, longit to shape up a not be run be belt flattens tening the heap (Fig 44) and causing spill, by dec of load section. The pile is also slightly agitate every idler, as its progress is a series of waves.

Spacing of idlers depends on the sag, which in depends on wt carried and the span. Following spa is recommended: for heavy ores, 3.5 to 4 ft; sand, coal, 4.5 to 5 ft; maximum spacing, 6 ft. Return i are spaced from 10 to 20 ft. No troughing idler sh be nearer than 3.5 ft to an end or bend pulley.

Trajectory of discharge, as determined by Re Conveying Belt Co, is shown by Fig 45. On start merely rolls over the head pulley. This will caus belt and chute beneath the pull belt until the nor

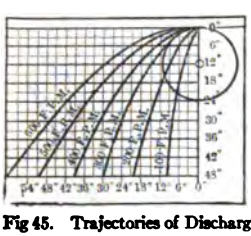


Fig 45. Trajectories of Discharge

Capacity is per hr on 48-in b on 32-in belt, ha Size of lump has can not be hand 30 in. Coke, re belt. Fig 48, 4

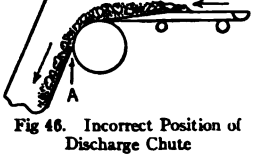


Fig 46. Incorrect Position of Discharge Chute

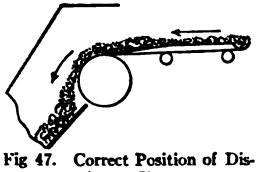
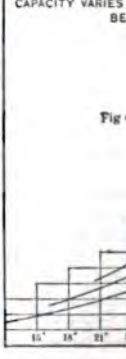
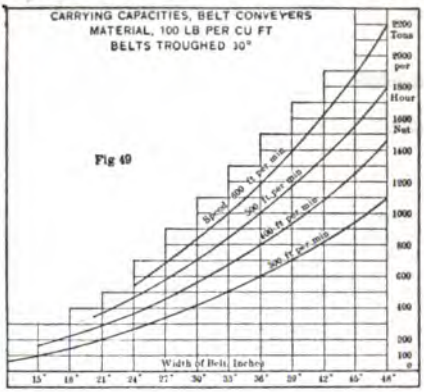


Fig 47. Correct Position of Discharge Chute



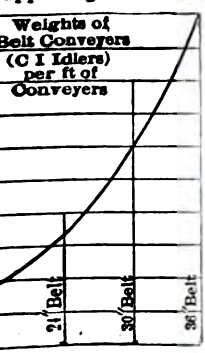
veyers, for material weighing 50 and 100 lb pe These are for belts troughed 30°; if troughed deduct 16%. Carrying capacity varies with

to drive on level: for 36-in conveyer, material 100 lb per cu ft, hp-hours
les = 0.81; for a 20-in conveyer, 1.08. With 50-lb material and 36-in
1.22; for a 20-in
1.68. In general,
hp-hr will move 1
, on a level. Trip-
1 to 5 hp, depend-
ze. Equation for
ons per hr × ft lift
net hp; add 0.15 to
ction.



cy depends mainly on
material carried and
Total hp is low, but
is small. Effic of a
eyer carrying sand,
e conveyer carrying
20-in conveyer carry-
5%; the same carry-
9%; 30-in conveyer
o tons coke per hr
y 0.1 effic.

ts (Fig 50). Belt conveyer is light and compact, and, in estimating costs, supporting structure should be credited. Trippers weigh 800 to 4 000 lb.



Structures and housing. Most belt conveyers are exposed to the weather. Conveyers on trestles often have low corrugated steel wind shields, to prevent the empty belt from being blown off the idlers, and starting up in that condition. Many are covered with low dog-house structures, just large enough to enclose the conveyer, with a railed walk alongside and hinged roofs. Trusses under trippers should be counterbraced. Idlers are usually mounted on planks or channels, with a continuous deck between them, built in removable panels, to keep fines off the lower return belt, and from falling farther. Stiff cross-bracing is essential to keep belt in a straight line horizontally.

Grades should not exceed 20°, to prevent slow creep of the load on face of belt; 18° is better. A 27° grade will work, but steep angles lessen belt life. On tripper lines a depressed tail is desirable, to prevent the tripper from lifting against the loading chute and cutting it (Fig 51). The angle should be an 15°. When steep grade is unavoidable, wear can be reduced by deep and steady stream of rom loading chute.

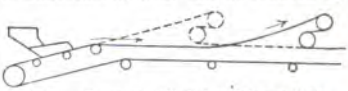


Fig 51. Depressed Tail for Tripper Belts

s carried: ore and rock from wn; gravel, sand, mixed con- ed materials, packages, and loose-bulk materials. Noth- han 160° F can be carried, nor any oily substances. Wet material is more than dry, because of interior rotting induced by absorbed moisture.

her than rubber and canvas are balatá, and waterproofed canvas. Balatá is rubber, but a trifle stronger, as less heat is used in curing, than in vulcanizing aterproofed canvas belts may be worked at 20 to 22 lb per in width per ply.

Various sections of rubber and canvas belts are sl

Splices. A working stress of 20 lb per in per ply is belt. This is necessary, to allow for strength of splices, its width under tension. Metal hooks are most satisf

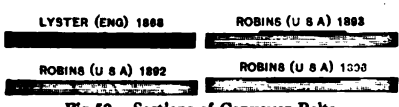


Fig 52. Sections of Conveyer Belts

7. Helical Convey

Characteristics: low first cost, compactness, small for useful work done. To the friction loss of load sli the friction of screw turning within the mass. Low c load line below the shaft center and bearings. For tons 100 ft per hr, with 40 hp. This is about 20 hp per as for a belt conveyer. The disparity increases wit friction: for bituminous coal, about 25 hp-hr per to rock and ore (with clean, lightly loaded trough) 25 t ground stuff is to be carried up an inclined belt (100 length of screw conveyer is required as a feeder, to Hot material is sometimes cooled in a screw conveye heated in transit; or the screw may be cut into p Ordinary grades of concrete can be cheaply mixed; mixer. Shafts are preferably tubular, for transverse ings should be babbitt lined, with grease cups. Spa lateral bending of shaft, and may vary from 7 to 15 should have flexible couplings; otherwise, angle or b (Art 1). Screws are usually of heavy sheet steel; ashes, smoothly molded C I is far more durable. steel, curved to conform to screw. Generous clearan mits accumulation of a layer of fines, which protec boxes, of rectangular section, lined with sheet steel, troughs. Careful lining up, and stiff supports are bearings are plain shoulders on end of shaft, with C-I b cation and are generally too small for long life. Long of line on timber supports, where beam loads cause ch ction affects the good running qualities of long overh

8. Feeders, Grizzlies, Gates, Chutes. Me
(See also Sec 28, Art 3

Feeders for continuous conveyers are essentia ... skips, and buckets. They act as eq ... walls, and may also be used advanta ... to prevent clogging, steady the motor

Types. ... reciprocating troughs, inclined ... plungers, with stationary ... gates; (f) rotary plates; (g) ... roll feeders. SIMPLE INCLI ... 0° to 15°, suspended by 4 adj ... way in an oval orbit and st ... SLIDER obviates this by placi ... inclined chute on 4 wheels (... It can be made to empty ... adjustable rods at front. ... 6 ...

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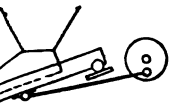
hese, there is considerable vibration, requiring stiff supports. Crank shaft otate more than 80 rev per min. All bearings should have bronze bushings. ing rods are desirable; best made of pipe with forged ends. Throw is usually k disks can be tapped for 3 radii of crank pin, to vary stroke. Stresses on



C. K. Baldwin Feeder

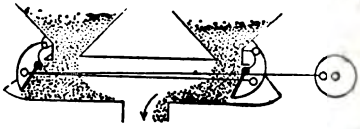


Fig 55. Duplex, Balanced Feeder (L. D. Moss)

anism and crank-shaft bearings are doubled for impact. Counterbalances s lessen vibration. Gear teeth may be cut on rims of disks to keep torsion shaft and save 1 gear wheel. Single and double swinging plates have been used r, to reduce vibration of large masses. s a duplex balanced form for a track riz chutes should be as short as possible, material will not fall beyond angle of l, 100 rev per min. Dempcey's plunger 56) may be horiz or inclined. It can be under a large hopper. Plunger is holly of C I, and slides on rails, which ily replaced, as they lie in the material, unprotected; speed, 100 rev per min. feeders can be varied while running, as in Fig 54.

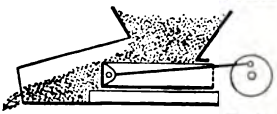
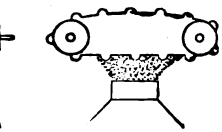


Fig 56. Dempcey's Plunger Feeder

y can be calculated by displacement, main factor being breadth. Re- feeders, if made as shown with a back plate, are really stationary pusher feeders. The incline does not increase capacity, but lessens work.



57. Apron Feeder

Apron feeders (Art 2) make an excellent gate for ore (Wellman-Seaver-Morgan Co) and may be power-driven (Fig 57). No skirt-boards are used. Rotary plates, as in the Chal-

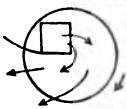
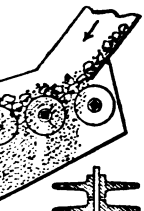
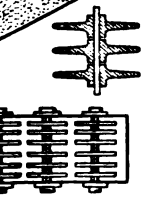


Fig 58. Action of Plate Feeder

and Trump mixer, consist of circular plates, level or slightly h a plow. They will not sweep off theoretical volume, owing to slip backwards on the plate (Fig 58).





Stockwell Feeder

Revolving grizzlies were introduced by E. V. Stockwell, chief engr, Braden Copper Co (Fig 59). A single set or a series of rolls may be used. The C-I plates are tapered to prevent wedging of lumps, and are loosely fitted on a square shaft to aid in freeing lumps. With sticky materials, stationary finger bars between disks will clear them. The spaces drop out the fines to a by-pass chute.

Roll-feeders, introduced by Hoover & Mason for ore and limestone, have a single hollow roller with C-I spiders and plate barrel (Fig 60); they also serve as gates. They feed too unsteadily for conveyers, but are good for filling cars. Variable-throw feeders are also used.

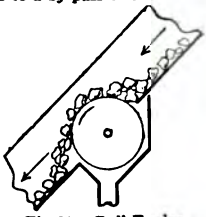


Fig 60. Roll Feeder

Gates of the undercut type are occasionally used driven by shafting and cranks or eccentrics. Cranks are preferable, giving and using less power.

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Gates of sliding type, though cheap, are often
fines, and freeze tight in winter, often requiring a
should be at least 1.5 × width.
isfactory (Fig 61). Duplex under
but for stone and ore, should hav
jamming. The Briggs (1873) fol
is easily moved. Clearance bet
must be small to avoid lodgment
may be external, as in Fig 63.
gate and chute for large stone,
freezing. Stationary grizzlies in
sing fines, are best made of taper
market. Clogging at lower end
accumulation of spalls is avoided by curving the end
slotted spacing bars are preferred by many to through
which offer more obst
to heavy falls of ore o
instead of plate.

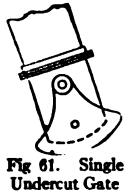


Fig 61. Single Undercut Gate

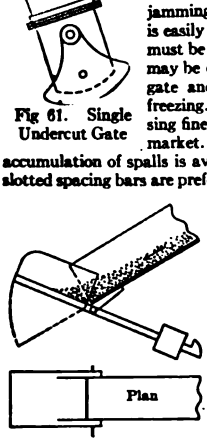


Fig 63. Briggs Gate

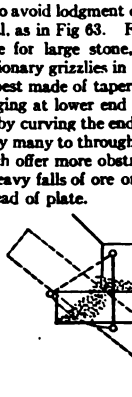


Fig 64. C

		tons per hr		
Reciprocating feeders require	1 hp	for	15–20	Roller
Plunger and push-plate "	1	"	20–30	Apron
Rotary plates "	1	"	10–15	Roll-f

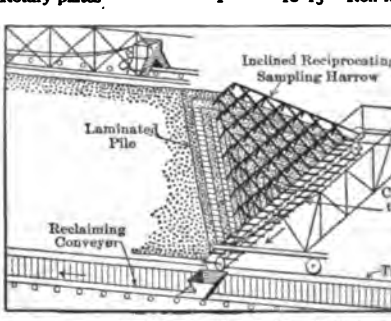


Fig 65. Messiter-Robins Reclaimer

A vibrating frame with harrow teeth, mounted
sample of all the alternate layers, to the reclai

9. Motors, Controls, Safeguard

Motors. For continuous conveyers, shunt cha
veyers which start loaded from rest perform better
slower acceleration in starting and permits use of a s
veyers frequently exceeds twice normal running load.

the short-time overload capacity of the motor, usually 0.25 to 0.5; in the best otors, with flame-proof insulation, it is 0.75 to twice normal load. Cold eases mechanical friction, but aids the motor. Waterproof casings seriously r capacity. When acceleration enters, as in intermittent service, compound tors are needed. Series motor should never be used if it can be cut loose from a clutch; it may run away and burst armature banding. For low total cost d drive reduction, medium-speed motors usually excel high and low-speed pur gears operate better. Forced ventilation by fan is sometimes imperative s. In very gritty atmosphere, perfectly enclosed motors are used, but allow- e made for their decreased rate of heat dissipation. • Open motors, surrounded g of steel angles covered with burlap, have been successful in very gritty at- A blower draws clean cool air from a distant point and maintains a small he motor room, the air slowly leaking through the burlap and excluding dust. ttent service, where accceleration enters, plot the entire cycle of a round trip, time-power curves and idle periods. If dynamic braking is used, charge it d, because the heating effect is the same. An idle motor does not dissipate dly as when the armature is in motion. During acceleration and retardation, eat dissipation is about midway between full speed and 0. Take about 0.25 l time idle instead of the actual time elapsed. For acceleration and retarda- bout 0.5 the total elapsed time, not the total time. The sq root of the mean he equivalent motor rating for continuous running. Starting torque must nvestigated; otherwise a motor with considerable amount of time idle may, hod, work out too small to start and accelerate the load. A good knowledge be done is essential when laying down a cycle of operation, especially with nachines. Retardation is important, otherwise at fairly high speeds the load e to rise after the motor has stopped, and surge back on the lines and fasten- angerous shocks.

and control. Rules of the National Board of Fire Underwriters carried out in the interest of safety and reliability. For movable ma- g contacts and flexible conductors may be required for safety from or naked conductors, copper wires are inferior to steel rails with soft which give much more durable installation. For a given current, res 10 to 15 times the cross-section of Lake copper, even if well bonded.

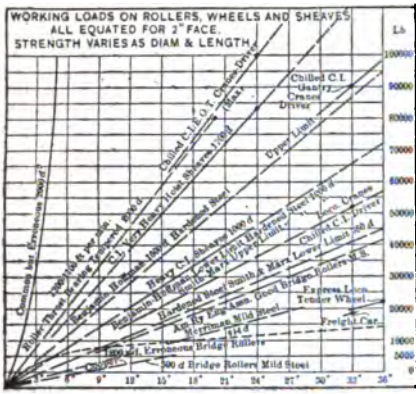
ver naked conductors are advisable to prevent short-circuits, fires, and acci- 90 000 machine has been imperiled because a tin pie plate blown by the wind 2 adjacent magnet control wires. For intermittent service, magnet control cost on motors of 25 hp and up; overloads can not be applied, acceleration rate mined and built into the control, and wasteful or careless manual control is

The use of 2 motors with series-parallel control, for lifting, is unsafe, because ad points in the control, permitting the load to drop unless very quick jumping roller contacts is made; inertia is insufficient to keep the load in motion verti- o similar motors operating as one, with magnet control, are sometimes prefer- parate motors, as 2 pinions can drive on the same gear, halving the tooth loads g amount of millwork.

Magnet brakes on motors should not be used as running brakes, but only to ature energy and to hold the load. The velocity of rubbing on shoe is too high an these duties. Running brakes, as near the load as possible, are always de- sbestos blocks give best service; maple or basswood, next best. V brakes will than flat, but the stop is rougher. (For use of springs, etc, see Sec 12, Art 5.) raking is desirable for machines in which the load is not fairly balanced; it is to mechanical braking, and safer. The heating effect is as severe as if the cur- used for lifting; treat it as a rising load. Close adjustment and stiffness are mportant for automatic mechanical brakes which set by gravity or helical spring, eased by a solenoid, with distant control. With this type of brake, a dash-pot, oil to by-pass through an adjustable cock from one side of piston to the other, to introduce a time lag when brake goes on, and to avoid a short quick stop.

sheaves, and rollers. The engineer must often pass on the numerous devices s field of work. Fig 66 shows current practice and its remarkably wide range, to speed, since shock varies as V^2. Flanges are the weakest parts of wheels.

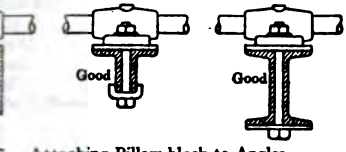
and should be extra thick where cramping on tracks oc
line contact, break on their length; when improperly su
fitted, their strength varies as diam and length, not as


WORKING LOADS ON ROLLERS, WHEELS AND SHEAVES
ALL EQUATED FOR 2" FACE.
STRENGTH VARIES AS DIAM & LENGTH
Lb
Chilled C.I. Gauge Crane Driver
Chilled C.I. 6 & 7 T. Crane Drive
Dyer Liner
Cast Steel Rolling Mill Pinion
Chilled C.I. Dyer
Cast Steel Rolling Mill Pinion
Forged Steel Rolling Mill Pinion
Cast Steel Rolling Mill Pinion
Forged Steel Rolling Mill Pinion
Forged Steel Rolling Mill Pinion
Cast Steel Rolling Mill Pinion
Forged Steel Rolling Mill Pinion
Forged Steel Rolling Mill Pinion
Forged

. Angle of teeth must not be too steep or the worm will overhaul; a gear 50 : 1 is good. Automatic solenoid grips are good, but expensive. In a e the utmost safety is desired, as over the roof of a power house, 2 stud-link laid along the tracks, anchored securely at each end, and with a bight carried ts or pocket sheaves. Links must be accurately pitched. Use a motor with gnet brake, and provide a screw post-brake for hand use in reserve. The thod is to use anchored wire ropes driven by geared windlasses. Tension tained in the ropes, as the drive is by friction. With a barom warning, there e to chain trucks to track and chock wheels.

10. Structures. Costs

upports receive the first shock, with small mass to absorb the blow; load and acceleration should be doubled when designing beams and connections. A descending empty carrier, suddenly arrested, may cause greater stress than the rising load plus acceleration. Large longitudinal stresses in conveyer supports, as from acceleration and retardation, often exist in addition to ction, requiring treatment as columns, and good cross-bracing. This in conveyers not driven at the head.



7. Attaching Pillow-block to Angles

ess often overlooked is due to the slewing of articulated spans. All spans uld be articulated, as it is impossible to drive the 2 ends alike; otherwise the will tear out. Spans supported on on the tower column caps require a lip them. This becomes a cross-bend- he horiz wind-truss system and upper s torsional stress on the supporting ng + and − stresses in columns, and esses in the column bracing. Mem- ten be calculated for deflection, not for cially for machinery supports. Pillow plates should not be bolted to legs of will deflect (Fig 67). On beams, use through-bolts to put stress into web, not g 68). Room for shims should be allowed on all structural work, to allow for inaccuracy in alinement. Engine bedplates and heavy pillow blocks on structural steel girders should have through-bolts to put tension directly into web, with ground-end stiffeners between chord angles to prevent rocking (Fig 69). Tension members should be bolted, not riveted, when connections are not in shear. Rivets in tension must be used in boiler bracing. However, many experiments show that rivets may cool in tension right up to the elastic limit; the longer the rivet, the greater the danger (see also Sec 43, Art 27). For machinery supports it is better to drill holes after erection to template, and use fitted bolts (tapered are better) for a driving fit, which will not work loose. Important bolts can rmly elastic by drilling a hole to reduce area of cross-sec in the body to that of on. Otherwise, bolt will fail in thread, as the rate of stretch is greater for a the lesser cross-sec. The pull on connection bolts is often twice as great as e member (Fig 70); there is a moment about the toe of each angle. Load

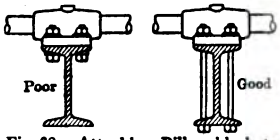
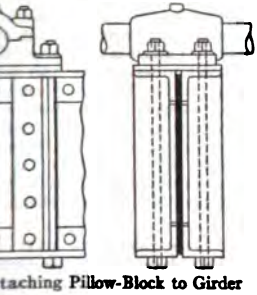


Fig 68. Attaching Pillow-block to I-beam



taching Pillow-Block to Girder

on each side, 0.5 P: W :: WF : PF. Net area at root pected, due to the difference in dies.

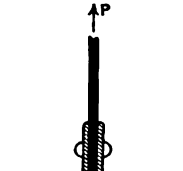


Fig 70. Stresses in Connection Rivets

Unit stresses and values. nance of Way Assn, for steel class of work (see also Sec 43). the equivalent static load may load + acceleration). An all the rails butt tightly and ex ing beams, which also furnish There is temptation to rivet tr section. It is rarely possible of structural steel work. A bolts being placed alternately rails will suffice for max wheel l Special crane rails with thick

Column feet or bases of s may have projecting rivet hea rivets and planing bases. La ting up columns on shims to grade (Fig 72). No founda diam; it will rust where it can not be seen, due to wate lugs for bolding down bolts should be stronger than

In general, use a few large members in preference to m members and bracing, to reduce maintenance, loss by ru to increase stiffness. Members which must stand with loose materials should be guarded against corrosion, lat of sliding material, and fire. Reinforced concrete is the tection yet devised (Sec 43, Art 14). It is difficult to bo rods in sliding material; they should not be used in su nor in ore pockets especially. Many short beams occ veyer structures. Note that standard connections (S '27) are too weak for short beams, I, L or channel, w loaded; also that very short beams used under columns by the wrinkling of their st on upper flanges of beams, used unless these flanges ar times breadth of flange.

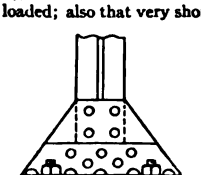


Fig 72. Setting Column Base

Timber, unfortunately, Art 20–24). Connections a a 12-in member; with large w Ends of plank braces should much wood ahead of bolt strongest when posts are but sills well drained and abov Notched timbers, holding ot of wood ahead of notch to skimped. If tension bars of they should lie on a through of material will pull them out. Large pockets require (Sec 12), seldom running below 60 in the most favorable to group posts occasionally into braced towers of 4 leg length of local timbers in the market. Otherwise, for ec bers." Manufacturer's Mutual Fire Ins Assn) the BM in deck of structure.

Concrete (Sec 43, Art 13). Floors of trenches and from water press at certain seasons. Reinforcing rods resist beam action. Small pits may float up under w hydrostatic press, weight them with concrete. Real tar and felt, behind a protecting layer of concrete not less

Costs. For bucket elevators and chain con from makers' catalogues. Discounts will about e

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ary considerably in price, due mainly to differences in grade of prod-
s per ft run of conveyer are misleading, owing to variations in length,
d cost of drive. Following list gives an idea of the range of costs
), including erection and wiring of motors, but no structures:

-in conveyer, no tripper, 100 ft long centers, $33 per lin ft.
" " " " 117 " " " 24 " " "
" " 1 " 190 " " " 18 " " "
" " no " 180 " " " 13 " " "
" " " " 50 " " " 19 " " "

l steel for trestles and runways, erected ..... 3.5–5¢ per lb.
" bridges and towers ..... 4–6¢ " "
gines for hoisting, slide-valve type (Sec 40).... 13–16¢ " "
er hp (Sec 42)..... $10–$25
ockets, per ton of capacity (Sec 12) ..... $60–$80
kets and bins, wt steel per ton capacity (Sec 12)..... 15–25%
el-concrete suspension bunker, steel 8–10%
linings of bunkers and bins, per sq ft ..... 30–40¢
ower, timber, 1-ton bucket..... $ 2 800–3 500
" steel " " " (40 tons steel)..... $ 3 500–4 000
" " 2 " " (100 " ")..... $30 000
" " 4 " " (200 " ")..... $40 000
rane, " 5 " " 250-ft span (400 tons)... $65 000
" " 5 " " 360-" " (600 ")..... $90 000

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SECTION

ORE DRESS

BY

ROBERT H. RICH

PROFESSOR OF MINING ENGINEERING AND METALLU
TECHNOLOGY, BOST

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37.	La	
38.	Mi	
	Bi	

Note. — Numbers in parentheses in text refer to B

1. Definitions

ng, or concentration, is the mechanical separation of gangue, or worth-rom valuable minerals of an ore. A MINERAL (Sec 1) is an element, or a ound, in which the valuable substance is present in constant or nearly rtion. An ORE (Sec 2, Art 13) is a mixture of minerals, the valuable sub-present in indefinite proportion. An ore may contain several minerals me valuable metal, or different metals; or a single mineral yielding several s; it may also contain one or more gangue minerals.

this section, valuable minerals, before concentration, will be termed ue or useless minerals, WASTE. Crude ore is termed FEED. (The word t used here, because not distinctive.) Separated values will be termed and waste minerals, TAILING. In a 2-mineral separation, as quartz and n the galena concentrate and the quartz tailing there will usually be an roduct, MIDDLING, composed of "included grains" of concentrate and waste, p and too rich to discard. Successful treatment of middling is often the nd costly problem in concentration. In a 3- or 4-mineral separation, for a (sp gr 7.5), pyrite (sp gr 5), blende (sp gr 4) and quartz (sp gr 2.64), the contain not only "included grains," but also medium-weight minerals, ing the separation.

2. Principles of Ore Dressing

s of minerals which assist in separating values from waste are: ty, hardness, tenacity, fracture, aggregation, color and luster, adhe-ness," electric conductivity, magnetic permeability, decrepitation. hese are defined below.. For further details see Richards' "Ore

vity (Sec 1, Art 3) is of prime importance, since it affects the settling ticles in water, air, or other media. Of two particles having same size and erent sp gr, the heavier will settle the faster; and of two particles having t different size, the larger will settle the faster. Thus, the sp gr of galena hat of calcite being 2.713, galena is 7.5 ÷ 2.713 or 2.762 times as heavy as merged in water, the ratio of effective sp gr of galena and calcite would be .713 — 1) = 3.794; hence the facility, in most cases, of wet, as compared ntration. See Table 1 on opposite page.

luster assist in distinguishing minerals in hand picking. ADHESION en-ght gold to be taken from pulp or sands by amalgamation. GREASINESS, ON, or aversion to being wetted with water, is the key to flotation methods. DUCTIVITY is the basis of electrostatic processes. Good conductors are d with electricity, and are then repelled by the electric pole, the reverse be poor conductors. By this means pyrite can generally be separated from NETIC PERMEABILITY is the basis of magnetic separation; for example, ble magnetite from non-permeable quartz.

BREAKING AND CRUSHING

3. Breaking by Hammers

dges, or 2-hand hammers, are used for breaking large pieces of ore able to enter the rock-breaker jaws. Where high-grade values cleave ow-grade, hand picking (Art 14) may accompany this work. Sledges striking faces, with beveled edges or sharp edges, or a sharp pean o aid in cleaving. The pean may be parallel or at right angles to ights of sledges are from 10 to 16 lb, 10-lb being preferred for at high altitudes, 16-lb for hard rock, and 12-lb for average service. ge from 28 to 36 in, 34-in being a common length. SPALLING e 2-hand hammers, weighing 2 to 4 lb, for breaking moderate-size

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Table 1. Specific Gravities of the Co
(For other minerals, see Sec

	Symbol	Sp gr	
Aluminum	Al	2.6	Hen
Amalgam	Ag_2Hg_3	13.75–14.1	Hes
Amphibole	(Sec 1, Table VII, 2, 3)	2.9–3.4	Iridi Irid
Anglesite	$PbSO_4$	6.1–6.4	Iron
Anthracite	.....	1.32–1.7	Lead
Antimony	Sb	6.5–6.7	Lim
Apatite	$Ca_5(Cl \cdot F)(PO_4)_3$	3.17–3.23	Mag
Argentite	Ag_2S	7.2–7.6	Mal
Arsenic	As	5.6–5.7	Man
Arsenopyrite	FeAsS	6.0–6.2	Mar
Azurite	$Cu_3(OH)_2(CO_3)_2$	3.8	Mer
Barite	$BaSO_4$	4.3–4.6	Mill
Biotite	$(H \cdot K)_2(Mg \cdot Fe)_2$ $Al_2(SiO_4)_3$	2.7	Moly Mus
Bismuth	Bi	9.7–9.8	Nick
Bitum coal	.....	1.14–1.4	Nick
Bornite	Cu_3FeS_4	4.9–5.4	Ozoc
Brown coal	.....	1.15–1.3	Plat
Calamine	$(ZnOH)_2SiO_3$	4.3–4.5	(n
Calcite	$CaCO_3$	2.7	Poly
Cassiterite	SnO_2	6.8–7.1	Prot
Cast iron	.....	7.3–7.73	Pyrr
Cerargyrite	AgCl	5.0–5.5	Pyri
Cerussite	$PbCO_3$	6.5–6.6	Pyro
Chalcocite	Cu_2S	5.5–5.8	Pyrr
Chalcopyrite	$CuFeS_2$	4.1–4.3	Quar
Chromite	$FeCr_2O_4$	4.3–4.6	Rho
Chromium	Cr	6.5	sit
Chrysocolla	$CuSiO_3 \cdot 2\,H_2O$	2.0–2.3	Serp
Cinnabar	HgS	8.0–8.2	Side
Cobalt	Co	8.6	Silv
Cobaltite	$CoAsS$	6.0–6.1	tiv
Copper (native)	Cu	8.8–8.9	Smit
Corundum	Al_2O_3	3.9–4.1	Spha
Cuprite	Cu_2O	5.8–6.1	Step
Dolomite	$CaMg(CO_3)_2$	2.8–2.9	Stib
Enargite	Cu_3AsS_4	4.4	Sulp
Epidote	$Ca_2(Al \cdot Fe)_3$ $(AlOH)(SiO_4)_3$	3.2–3.5	Sylv Talc
Fluorite	CaF_2	3.0–3.3	Tetr
Galenite	PbS	7.4–7.6	Tin
Garnet	$R_3R_2(SiO_4)_3$	3.1–4.3	Tita
Gold (native)	Au	15.6–19.3	Wolf
Graphite	C	2.1–2.2	Zinc
Gypsum	$CaSO_4 \cdot 2\,H_2O$	2.3	Zinc

lumps (4, 5, or 6 in) to uniform size of about
in long; about 1.5 in thick at hand end, and
¾ in for a distance of 10 in, beginning at 6
capacity of a man at 1 400 lb per hr of ordin
COBBING HAMMERS are 1-hand hammers, for
from waste, the cobber sitting with a pou
cobbing hammer has at one end a sharp wed

gles to handle. Weights range from 1.5 to 4 lb, average 2 to 3 lb; dly 12 in.

ammers (3) are used with success in Lake Superior district for all mass copper from adhering rock. Cylinder, 10 by 18 in, con- p at 60-lb pressure and 144 strokes per min. Weight of striking); of anvil, 3 000 lb. The shoe, of gray cast iron, lasts 90 days, durable than chilled iron. With this hammer 3 men can dress 1 pper per hr, yielding about 75% clean copper. Large steam ham- rly used at Lake Superior for breaking copper rock of 1 cu yd or been unnecessary since the introduction of dynamite in the mines. ERS (4) are for breaking large lumps for crusher, and cleaning mass smelter. In one size, the head, 7 ft long by 12 in diam, weighs drops 6 ft, running in guides like a pile-driver.

4. Coarse Breakers

eaker (5) (Fig 1) has a fixed jaw 3, and a pivoted moving jaw 4, eccentric 9, pitman 6, and toggles 14.

ed at top, the jaw makes ent at upper end, where e lumps, and a larger ow, where material is of The jaw plates, of chilled are replaceable when y wear, the space be- comes larger. Wedge 8, crew and lock nuts 16, opening and the size of Spring and rod 30 serve ws. During a half re- ving pulley, the jaws ned; during the other close upon the ore, the belt power being augmented by power stored in

Fig 1. Section of Blake Jaw Breaker

e of feed usually ranges from 6 to 24 in; crushed product, 1 to 4 in. The ushers built for the Lake Superior copper district take lumps 84 by 60 in, at 16 in.

e 2. Blake Jaw Breakers: Sizes, Capacities, Power, etc (6)

Capacities in tons per hr								Approx h p required	Weight complete, lb	Extreme dimensions, including shafts			Approx price
Size of product, in										Length	Breadth	Height	
1½	2	2½	3	4	5	6				ft in	ft in	ft in	
4	5	....	....	....	....	....		7	6 500	4- 8	4- 2¼	3-2⅛	$ 375
8	10	12	....	....	....	....		10	12 000	5- 5	5- 1⅝	3-7⅞	600
10	15	17½	20	....	....	....		15	15 000	6- 1	5-11¾	4-2	750
....	20	25	30	35	....	....		25	25 000	6-10½	8- 0	4-7	1 200
....	....	30	37½	45	50	....		40	47 000	8-11	9- 3	5-5½	2 100
....	....	....	60	72	85	100		60	80 000	10- 7	10- 8	9-8	2 700

apacities are approximate, varying according to character of material tance jaws are set apart, and to speed. Hard stone or ore that breaks will go through faster than sandstone. Average speed, 250–275 rev per

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Wear. Important items of cost of crushi
and time lost thereby. Table 3 contains dat

Table 3. Wear of Blake Jaw

ABBREVIATIONS. C I = cast iron; C P = chee
= chilled iron; Chr S = chrome steel; F J P
quality babbitt; J P = jaw plates; M S = manga
S = steel; S J P = swinging jaw plates; T = t
= tension rod; Wh I = white iron.

Wearing part	Material	Total weight, lb		Actual or estimated maximum life		Wear pe lb
		New	Old	Days	Tons	Gross
F J P.	M S..	1660	1385	135	56 600	0.02930
F J P.	Cd S.	323	274	19	10 400	0.03105
S J P.	M S ..	1340	1128	135	56 600	0.02340
S J P.	Cd S.	293	247	19	10 400	0.02820
J P...	M S..	441	140	60	1 440	0.30600
J P...	Chr S	441	140	45	980	0.45000
J P...	Wh I.	441	140	30	720	0.61200
J P...	M S..	468	140	104	12 000	0.03900
J P...	Chr S	468	140	65	7 500	0.06200
J P...	Ch I.	468	140	26	3 000	0.15600
J P...	Chr S	400	300	120	15 000	0.02600
J P...	Ch I.	400	300	80	10 000	0.04000
C P..	S.....	63	30	135	56 600	0.00111
C P..	M S..	90	72	130	15 000	0.00600
C P..	Chr S	90	72	78	9 000	0.01000
C P..	Ch I.	90	72	21	2 250	0.04000
C P..	Chr S	80	60	180	22 500	0.00400
C P..	Ch I.	80	60	120	15 000	0.00500
T....	C I...	70	70	....	.....	0.00000
T B..	C I...	26	0	180	96 000	0.00027
T R..	S.....	51	0	130	57 920	0.00088
P B..	G Q B	22	0	90	47 500	0.00046

(*a*) Soft sulphide-bearing quartz, Mex. (*b*) S
quartz, Calif. (*c*) Sulphide-bearing solid quartz a
quartz, barite, calcite, and rhodochrosite, Col.

Table 4. Wear of Jaw Plates o

Chrome steel	
Cast iron (probably chilled)	
Manganese steel	
White iron	
Chilled iron	

* This cost is unfavorably influenced by one v
reduces the average to 0.879¢.

† This contains one very high figure which, if c

Crushing efficiency. Table 5 gives resul
of a 15 by 9-in Blake breaker working on ru
and galena; also 2 tests on hard close-grained

Table 5. Sizing Tests on Blake Breaker Products

ite and galena		Hard lime-gravel gravel			
ening	Per cent	Screen opening			
.....	51 2	On 2 5 in.		7 0	6 5
875 in	16 6	2 5–2 0 "		11 0	23 0
625 "	7 3	2 0–1 5 "		23 0	20 1
375 "	7 9	1 5–1 0 "		25 2	17 8
25 "	6 2	1 0–0 75 "		6 4	4 8
00 mm	4 8	0 75–0 50 "		5 9	7 9
25 "	4 4	0 50–0 25 "		8 7	8 2
25 "	0 4	0 25–0 125 "		4 5	4 1
25 "	1 1	Through 0 125 "		7 3	6 6
	99 9			100 0	100 0

Table 6. Cost of Operating Blake Jaw Breakers

from large amount of mill data; labor rated at $2 per day of 12 hr

e of mouth, in	10×4	10×7	15×9	20×10	30×13
to 2 in per 24 hr	92	120	192	360	600
.....	5	8	12	20	40
er	$275	$500	$750	$1 050	$2 250
t, cents per ton:					
.....	0 020	0 020	0 020	0 020	0 020
prec.	0 097	0 135	0 127	0 095	0 122
.....	0 705	0 865	0 811	0 721	0 865
.....	4 348	3 333	2 083	1 111	0 667
.....	0 815	0 815	0 815	0 815	0 815
.....	0 462	0 462	0 462	0 462	0 462
.....	6 447	5 630	4 318	3 224	2 951

aker (Fig 2) has a fixed jaw 2, and a moving jaw 1, which, unlike ivoted below at 4. It is actuated by eccentric 8, connecting rod 7, The jaw movement st at the mouth, and he throat (7). Jaw placeable, as in the r; their adjustment ade by placing shims bearing. Owing to ce of action, the r has smaller capac- Blake for a given jaw makes a more uni-

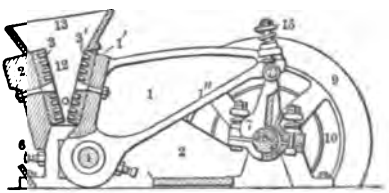

Fig 2. Section of Dodge Breaker

. For comparison of operation of Blake *vs* Dodge breaker, see g," Vol I, pp 33–36.

aker resembles Dodge in that motion at mouth is greater than at scillation is imparted by pitman and toggles, as in Blake breaker, g the heavy casting 1 (Fig 2). Test recorded in Table 9 was rom breaker.

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Table 7. Capacities, Power, and Si
(Data from catalog

Jaw opening, in	Pulley, in	H p required	Capaci tons p
6×4	4×20	2 to 4	18
6×6	5×12	3	24
8×7	6×16	5	72
10×7	6×24	4 to 8	72
10×8	8×24	6	96
12×8	8×24	7	96
Special			
15×9	10×30	9	192
15×11	10×36	8 to 12	192

Table 8. Wear of Dodg

ABBREVIATIONS. Ch I = chilled iron; C I = c = cast steel; F S = forged steel; H S = hamme = manganese steel; S = steel; S C = steel castin ings; W S = wrought steel.

Wear-ing part	Mate-rial	Total weight, lb		Cost per lb, new, cents	Sell per lb, old, cents	
		New	Old			Days
2 C P	C I	60	40	2½	0	(a)120
2 T	S C	100	(b)	8	0	180
4 T B	S C	80	(b)	8	0	60
2 J P	M S H S C S	470	82	10½ 9½ 6	0.65 0.65 0.65	75 65 35
2 C P	C S	114	86	6	0.65	150
2 J P	M S	250	100	(c)10	0	365
2 C P	S	60	40	(c)10	0	210
J P	F S	.....	.....	.....	0	.....
J P	Ch I	680	.....	5	.....	24

(a) Removed more often from breaking than fr the freight.

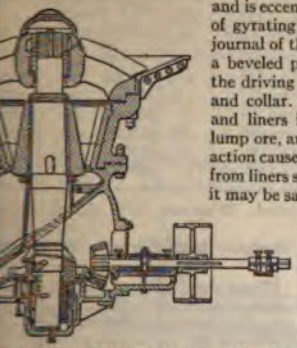
Table 9. Products of Dodge (Kro

Screen opening, square holes	I	
	Per cent	P
On 45 mm	.....	...
" 32 "	2.73	
" 26.7 "	11.43	
" 18.9 "	26.11	
" 13.3 "	20.31	
" 6.7 "	16.21	
" 3.3 "	8.19	
" 1.6 "	4.78	
" 0.8 "	2.73	
" 0.4 "	2.39	
Through 0.4 "	5.12	
	100.00	

Notes. — Tests I and III on granitic lead-zinc s vein quartz. Tests I and II, feed opening 11.5 t Tests III and IV, feed openng 12.6 to 15 cm; disc was fist size, without fines.

yratory breaker (Fig 3) crushes by gyrating a truncated conical end up) within a truncated conical ring die or ring of liners (small

ad spindle is pivoted at upper end, and is gyrated in a circle of small diam- end by a horizontal beveled gear. The spindle journal is in a long hub, and is eccentric to it by an amount equal to radius of gyrating circle. The outside of the hub is the journal of the bevel gear. This gear is driven by a beveled pinion on a horizontal shaft, to which the driving pulley is attached by a shearing pin and collar. The opening between crusher head and liners is large at the mouth, for receiving lump ore, and small at throat (9). The gyrating action causes breaker head to approach and recede from liners successively around the circle; whence it may be said that the gyratory breaker is crushing half the ore all the time, while the Blake and Dodge are crushing all the ore half the time. Moreover, the gyratory requires no flywheel.



n of Gates-McCully Gyratory Breaker

Adjustments. As the breaker head extends below the liners, wear is taken up by lifting crusher head. The Gates device for this was a vertical set-screw, and blocks or buttons aring of the shaft. The McCully improvement puts lock-nuts on upper end iminating considerable friction of spindle in the step-bearing, due to its liminishing the power required. The Gates style K breaker accomplishes a nut with split side and locking bolt.

l products. Like the Blake, the gyratory takes large sizes, 6 to ing them to 1 to 4 in. A large machine for coarse work may be 2 or 3 small machines for fine work. To receive lumps of the Gates gyratory needs about 3 times as much mill height as a jaw

e 10. Capacity, etc, Gates Gyratory Breaker, Type D (10)

℄ e- ıg-)	Wt complete, lb	Capacity per hr, tons. Machine set for 2½ to 5-in size	Smallest sized product at one break, in	Driving pulley, in		Speed of driving pulley, rev per min	H p required	Price
				Diam	Face			
6	650	.....	⅜	8	2⅜	700	1 to 1½	....
5	3 850	2 to 4	¾	16	6	500	3 to 4	$470
8	5 900	4 to 8	1	20	7	475	5 to 8	585
1	8 400	6 to 12	1⅛	24	8	450	7 to 12	700
2	14 480	10 to 20	1¾	28	10	425	10 to 16	925
7	21 700	15 to 30	1½	32	12	400	14 to 21	1 300
0	30 700	25 to 40	2	36	14	375	22 to 30	1 700
6	43 000	30 to 60	2¼	40	16	350	28 to 45	2 050
5	48 000	75 to 125	3	44	18	350	50 to 75	3 100

(a) Machine has 2 openings, each of size stated.

yratory crusher. Gyratory is more efficient, mechanically; hence here large capacity is required. Wear and repairs cost more for To take feed in large chunks, jaw crusher is lighter, cheaper, and ch less mill height. Best arrangement for such condition would be crushers delivering to 1 gyratory, with another gyratory in reserve.

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Table 11. Capacity, etc, Gates Gyr

	Size of each receiving opening, in (a)	Weight complete, lb	Capacity per hr, tons, to pass opening of							
			1½	1¾	2	2½	3	3½	4	5
4	8×30	22 000	15	20	25	30	40	...	...	...
5	10×38	32 800	...	30	40	50	60	70	...	...
6	12×44	48 000	...	...	50	70	80	90	...	...
7½	14×52	68 100	...	...	...	80	90	100	120	...
8	18×68	103 000	...	...	...	...	...	130	150	...
9	21×76	155 000	...	...	...	...	...	...	250	300

(a) There are 2 openings, each of size stated. Othe

Table 12. Sizing Tests of Products f

Size	Tests 3 a
	Per ce
Over 1 in	0.3
Through 1, on ¾ in	10.1
" ¾, " ½ "	43.0
" ½, " ¼ "	20.0
" ¼ in	24.5
Total	100.0

Table 13. Wear of Spind

ABBREVIATIONS. B P = breaking pin; B W iron; E = each; H = Head; L = Liners; M G

Wearing part	Material	Total wt, lb		Cost per lb, new, cents	Sell per lb, old, cents	Da
		New	Old			
H	Ch I	3 300	2 700	6	¾	3
L	"	1 440 1 760	1 340	6	¾	1 1
H	"	.....	.....	(a)	½	2
L	"	90 E	70 E	...	½	2
H	"	800	600	5⅜	...	1
12 L	"	600	400	5	...	2
B P	.....	(b)	.....	...	...	
H	"	880	500	7	1¾	1
L	"	763	500	7	1¾	
H	"	.....	.....	...	...	1
H	B W I	1 800	.....	7	...	1
L	"	1 375	.....	7	...	1
H	Ch I	1 000	700	4	¾	1
L	"	2 000	1 400	4	¾	1
M GB	.....	.....	.....	...	...	3

(a) $80 per set. (b)

Table 14. Cost of Crushing by Spindle Breakers (12)

outh, in	4×30	6×42	8×54	11×72	18×126
shed in 24 hr	72	216	540	1 080	3 000
wer	3	9	22	45	125
reaker (approx)	$375	$760	$1 800	$3 300	$7 000
g cost, cents per ton:					
	0.021	0.021	0.021	0.021	0.021
t and depreciation	0.169	0.114	0.108	0.099	0.076
	0.541	0.541	0.541	0.541	0.541
	5.556	1.852	0.741	0.370	0.133
	0.971	0.971	0.971	0.971	0.971
s	0.308	0.308	0.308	0.308	0.308
l	7.566	3.807	2.678	2.310	2.050

s disk crusher (Symons Bros Co, Milwaukee) (Fig 4) crushes be- early vertical, manganese-steel disks. Each disk is driven by its rotating direction, cing given y move- isks are hed, with nferences ther than s; hence, n when s analo- t of a gy- her. The ed to the crushed g carried h the disks, and is promptly thrown out by centrifugal force, thus giving ines. Disks are set up closer as they wear. As compared with other ushers, this has much the largest relative space for discharge of crushed

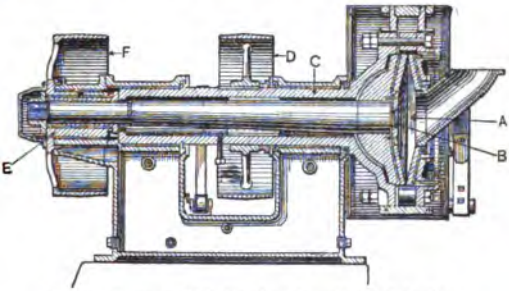

Fig 4. Symons Disk Crusher, Longitudinal Section

able 15. Weights and Capacities of Symons Disk Crushers

Size of crusher	48-in	36-in	24-in	18-in
ipping wt, lb	30 000	19 000	8 500	5 600
n elliptical feed spout, in	11¾×17	9¼×14½	7×10⅝	4¾×7
et disks at feed spout, in	8	5	3¼	2¼
pening for best results, in	1	¾	½	⅜
in, main pulley	100	133	200	200
in, eccentric pulley	250	300	400	450
lleys, in	54×16	44×14	34×10	28×8
ed	50–65	30–40	18–25	12–18
	Size Tons	Size Tons	Size Tons	Size Tons
tons per hr	1 = 45– 60 1½ = 60– 80 2 = 80–100 2½ = 100–120	¾ = 25–30 1 = 30–45 1½ = 45–60 2 = 50–65	½ = 12–15 ¾ = 18–20 1 = 20–25 1½ = 25–30	⅜ = 5– 8 ½ = 8–10 ¾ = 10–12 1 = 12–15

st on product from 48-in disk crushers, treating 2 to 4-in dolomitic galena ore **er, Mo: On** 1-in, 36%; on ¾-in, 35%; on 10-mm, 16%; on ¾-mm, 11%; **mm, 2%.**

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Symons "pillar" gyratory has crushing he
by a vertical collar eccentrically mounted on a
lateral movement of head is equal at top and
and more uniformly sized product than swingi
portant advantage is that the great length of
effective crushing leverage in the swinging typ
40% in height of machine. Weight of pillar g
than that of a Blake breaker of corresponding
than weight of swinging gyratory (111).

5. Crushing Rol

These consist of 2 parallel horizontal cylin
towards each other. Ore thrown between them

Construction and wearing parts (Fig 5). Bearing
in which case they are held towards bearings of the fi

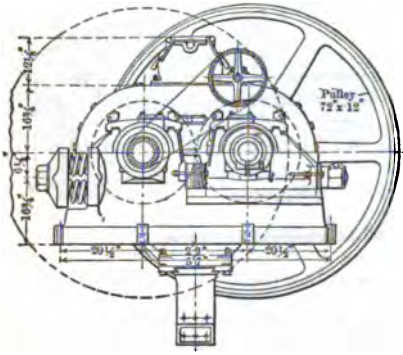

Fig 5. Side Elevation of Gates Rolls (13)

rather than chilled cast iron; manganese steel po
nary steel, in fact, is found more apt to become gr
iron is that it costs less; for the various steels, tha
may cost 2 to 4¢ per lb, cast steel 6¢, forged, ha
chrome or manganese steel 10¢ The question t
least cost per ton, taking into account labor of c
pages. ROLL FRAMES are best made in a comple
ing bearings, springs, etc, and 2 ends for stiffness
of mild steel or wrought iron, made extra heavy

Pillow blocks of fixed roll should be built true,
accurate alinement. This is done by springs, screws,
cheek bearing to help this, to which is sometimes add
ends of adjustment screws, causing the screws on both
amount. McFarlane has a sliding frame for moving
which guarantees parallel roll shafts. Krom has swin
shaft resting upon 2 parallel arms, keyed to a conn
bearing thus causing similar movement in the other.

Springs. The actual crushing force may no
to use springs giving much greater pressure, 15
size of rolls. Springs avoid damage when an

incur unequal crushing under ordinary operation. RIGID ROLLS Eng Wks) yield more uniform product, and can be protected by ed under a strong electro-magnet.

g mechanism. Slow-speed, geared rolls are still used to some ex- power being received by belt from a fast countershaft. Slow rota- nsmitted by gears to the fixed roll, from which long-tooth star gears e moving roll. Three pairs of star gears, graded in size, will serve a lls from maximum size when new to minimum size when worn out. m- and high-speed rolls a direct and a crossed belt make the simplest here it is desirable to have the power shaft near the rolls, an extra and large diam pulley are used on fixed roll, while a small belt and ve for the crossed belt for moving roll.

rolls. Width of face varies from 9 to 18 in, usually 14 or 16 in. ge from 12 to 42 in; sizes commonly used are 24, 26, 30, and 36 in. mill has rolls 72-in diam by 20-in face. The Utah Copper Co and mill both employ rolls of 54-in diameter.

Because of the great variation in diam of rolls, peripheral speed best measure for comparison. Speeds range from 30 to 40 ft per Cornish rolls, to 800 or 1 000 ft per min in modern high-speed rolls. believes average of modern coarse rolls is about 650 ft per min; eed fine rolls, 900 ft per min.

, ridges, corrugations, and pits. If rolls lap slightly over each unopposed edges wear less and bad flanges soon form. Circumfer- es and hollows are often formed on surface of rolls. Chilled shells, f of poor metal, may become pitted with cavities large enough to let ere drop through without being crushed. With close-fitting cheek s can do no crushing on their extreme ends, when crushing 1-in stuff, se both rolls become flanged at both ends.

ay be prevented, or cut away, by running with cheek plates offset from ends stance equal to the half-diameter of maximum grain in the feed. Flanges nd down by an emery block held against them by a heavy weight; or better, emery wheel may be used while roll is running slowly; or the roll may be the, with an emery wheel. If the rolls in a mill are uniform, new shells may fine crushing, shifting them to coarse rolls when surfaces are too rough for To secure uniform wear, rolls may be shifted every day; first lapping one he other. Side shifting may be done automatically; H. K. Burch has de- roll (made by Traylor & Co). J. S. Frazee (15) for dry fine crushing makes r than the other, so that 2 flanges form; by careful feeding this roll wears ghout its life. Badly pitted shells are best discarded.

. The following are all good: Gates stirrup feeder, Tulloch jerking- ndy Challenge, and the cylinder feeder. The feeder must prevent re, and distribute the ore evenly from side to side of rolls. Guide ometimes put into feed spout, to direct feed away from a low spot high spot on the shell.

feed. Angle of nip. Space between rolls. Size of feed rarely in (37 mm) diam, and a minimum of 1/12 in (2 mm) is rarely passed. e too large they cannot be nipped by the rolls. For quartz ores the one roll at point of contact with maximum grain of feed ore should gle not greater than 32° with the similar tangent of other roll. Fric- n roll and fragment will then be sufficient to draw in the fragment t. This angle is called ANGLE OF NIP. To secure it, diam of roll, een rolls, and diam of particle, must all be considered. Slippery e anthracite, graphite, and talc, need a smaller angle of nip.

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Table 16. Sizes of Feed which Give 32°

Diameter of rolls, in	Space between r			
	¾	⅝	½	
	Size of feed, in inches, m			
36	2.23	2.10	1.96	1
30	1.99	1.86	1.73	1
26	1.83	1.70	1.56	1
24	1.74	1.61	1.48	1
20	1.58	1.46	1.32	1
16	1.42	1.29	1.16	1
9	1.14	1.01	0.88	0

With rapid-running rolls it may be well to r
because rock fragments are batted from one r
time to settle and give friction a chance to nip

Graded crushing. While rolls rank as the prod
tage can be enhanced by crushing in stages. For e
to 12 mm, the second from 12 to 6 mm, and the thi
screened out between the stages, the total product
than if all the crushing had been done in one oper
fines at each step, the total quantity of fines is reduc
excess of fines, rolls should not be required to reduce
diam.

Choke crushing. In some circumstances it is de
of ore to a fine size without regard to production o
used small rolls with 90° pyramids all over their surf
spaced ½ in between points of pyramids. These w
customary for free crushing or minimum fines, ore p
ing in greatly increased capacity.

Free crushing. This is the normal way of runni
rolls be imagined to be rolling out a ribbon of dough,
Assume rolls 15 in wide (1.25 ft), with peripheral sp
(0.0417 ft), and that broken quartz weighs 110 lb p
min, the theoretical crushing capacity is:
(650 × 1.25 × 0.0417 × 1 440 × 110) ÷ 2 000 =
mills, the amount actually crushed is only from ½0

Ratio of roll opening to trommel hole.
quite down to size of their opening. When the
the screen holes should be 1.25 times the wid
trommel oversize to a small amount.

Journal friction. If rolls are set in contact, with
the whole pressure may be on the journals, requir
overcome friction even when no crushing is being do
by the thickness of a sheet of paper, then the sprin
and acts on the journals only when the crushing pr
space between the rolls. By observing this precaut
avoided.

Power required by rolls is difficult to stat
with speed, size of feed, size of product, and h
mates by a builder of rolls are given in Table 1

Table 17. Size, Weight, Power, etc, of Rolls (17)

Weight, lb	Approx cost	Approx h p
xtra heavy, style A		
197 000	$9 000	55–135
121 600	6 000	40–85
37 000	2 400	20
18 000	1 500	12
Style B		
39 000	$2 200	27
24 500	1 500	20
18 300	1 250	16
14 000	920	12

Size, in	Weight, lb	Approx cost	Approx h p
Economic type			
30×14	16 500	$1 100	16
24×14	10 300	725	12
18×10	4 700	425	10
Rigid type			
36×15	25 600	$1 500	20
36×10	21 600	1 200	13
Anaconda type			
54×24	98 000	$5 000	40–50
42×16	75 000	4 000	30–40
40×15	38 000	2 250	25–30

of work. Table 18 shows that rolls make much less fines than crushers; that, where values are in softer minerals and more easily n waste, richness increases in the finer sizes.

able 18. Sizing Test on Products of Montana Rolls (18)

Screen openings	Per cent weight	Silver, oz per ton
2.8 mm	9.03	12.7
2.0 "	34.70	14.3
1.4 "	16.21	13.9
1.0 "	11.70	14.1
0 6 "	7.42	15.3
0.16 "	11.44	17.5
0.16 "	9.50	23.6

crushing by rolls. Table 19 assumes labor at $3 per 12-hr shift, np costs $40 per year of 308 days of 24 hr.

Table 19. Cost of Crushing by Rolls (19)

	Cents per ton
er	1.30
ndance	1.50
r of roll shells	0.02 to 4.00
irs, oil, babbitt	0.37 to 0.60
tal (not including cost of truing rolls)	3.19 to 7.40

odale gives cost of crushing, including screening and elevating, at

6. Steam Stamps

amp is largest and most powerful of all finishing crushers. It has ed by and for the native copper rock of Lake Superior, in which mineral is tough and ductile; on ore containing rich brittle min- ion of slime would be excessive. It is operated by a vertical steam shing on a die in a mortar. The crushing is done in water, and the eave the mortar until fine enough to pass through the screen.

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Types. There are 3 chief makes: Nordl
Chalmers. The Nordberg is generally tandem
differential cylinde
a single simple cy
more powerful tha

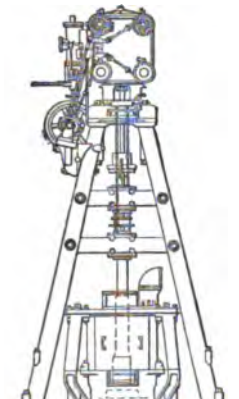


Fig 6. Nordberg Steam Stamp, Side Elevation

Foundation is of
deep. Above this fo
bers; now replaced b
built in sectional bl
capacity.

Mortar and scre
anvil, is a cup-shape
deep inside, with fl
down. Mortar hous
and are about 2.5 ft h
with shields to retain
water and sands. T
stem and for the feed
usually with 5/8-in r
3/16-in for conglomer
a frame and 2 fram
back; area of each s
4 screen frames looki
broken only by the

Die, die ring, and
8 in high, 20 and 22 i
its life it crushes 39
175 lb. A die ring c
820 lb, wearing to ab
tons. Mortar is lined with chilled iron "staves," re
237 lb, and wears to about 125 lb in crushing 39 00

Shoes, of hardest chilled iron, weigh 600 lb and
300 lb. They last 60 hr on conglomerate, 6 to 12
amygdaloid, and crush 1 500 to 6 000 tons.

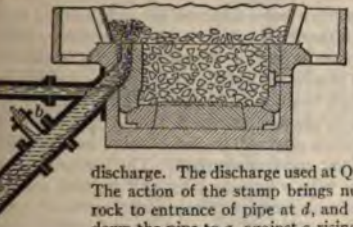
Stamp stem, piston rod, piston, and cylinder
Stem is of steel, 8 in diam, widened to 14 by 22 at h
with a double dovetail and shims for attaching the sh
is also 14 by 22 in. In ALLIS-CHALMERS stamp the
of shaft is connected with piston rod by flange and bo
ing rubber cushions to deaden shocks (20).

Piston, cylinder, and valves are of several desig
to obtain the most powerful blow, least waste of time
blows, greatest safety from striking top cylinder hea
too late admission of steam), or bottom cylinder hea
insufficient feed in mortar), and minimum loss of cle
both ends. LEAVITT differential stamp has consta
steam pressure (75 lb), acting upward on the smaller
higher pressure (100 lb), acting on the larger pistor
down stroke, which is made with an early cut off. Its
is 300 tons of conglomerate per 24 hr, through 3/4
NORDBERG stamp, usually tandem compound, has Co
gear. It will crush 88 tons of rock per ton of coal
single-cylinder stamp, 20 in diam by 24 in stroke, crus
63 tons per ton of coal. Capacity, 500 to 600 tons of
loid per 24 hr, through 5/8-in mesh. ALLIS-CHALMER
with 24 by 24-in cylinder, ranks with the highest
stamps at Lake Superior. Capacity, 600 to 700 to
diam, and it is claimed to be as economical as the

ady running, and ease of repairs, although its capacity is less per ton of coal berg. The usual single cylinders at Lake Superior are 20 by 24 in. Leavitt hp on up stroke, 103 hp on down, total 146 hp. Allis-Chalmers stamp, nd 109 hp on down stroke, total, 171 hp.

ta. Rotation is by belt to pulley on stamp shaft between its stamp revolves once in about 6 strokes. It makes 90 to 100 24-in in; speed at instant of blow, 20 to 26 ft per sec. Approx weight of : shaft, 3 500 lb; shoe, 600 lb; piston rod and piston, 700 lb; total, b. Water used in stamping, about 55 tons per ton of rock.

The rock, which has passed through a 3-in grizzly, is fed from a chute. picks out nugget copper, wood, and sometimes waste rock, and regulates the feed. A tell-tale bell, struck by a tappet on stamp shaft, rings if stamp drops too far, owing to mortar being too empty. The crushed material passes through screens, much of the nugget copper going through a special mortar discharge. The discharge used at Quincy mill is shown in Fig 8. The action of the stamp brings nugget copper and pieces of rock to entrance of pipe at *d*, and the nuggets find their way down the pipe to *a*, against a rising current of water too swift to allow waste to settle. Valve at *b* must be closed when nuggets are discharged from pipe at *a* (21).



y Mortar (21)

Clean-up. Stamp shoe wears out in 2½ days on conglomerate, and 6 to 12 days on amygdaloid. Stamp is then stopped, front screen removed, shims driven out, old shoe replaced by a new one, shims replaced, screen, staves, die, and other parts frame replaced, and the stamp restarted. The residue is cleaned up and n a "bull jig."

Table 20. Cost of Crushing by Steam Stamps (22)

	Amygdaloid rock, cents per ton	Conglomerate rock, cents per ton
.....	1.600	1.846
.....	12.152	12.152
.....	0.139	0.258
.....	0.475	2.350
.....	0.054	0.112
.....	0.041	0.028
.....	0.071	0.102
.....	0.385	0.385
.....	1.639	1.639
	16.556	18.872

g Test on Amygdaloid Crushed by Steam Stamp (23)

	Per cent
5 mesh.....	44.5
o "	11.1
o "	7.0
o "	6.2
o "	4.8
o "	3.8
o "	22.2
.....	1.2
	100.0

Table 22. Sizing Test on Butte Sulphide Ore Crushed by Steam Stamp (22)

				Per cent
4	mm to	2.8	mm.....	2.40
2.8	" "	2.0	"	8.22
2.0	" "	1.4	"	8.22
1.4	" "	1.0	"	9.12
1.0	" "	0.6	"	8.85
0.6	" "	0.42	"	7.77
0.42	" "	0.25	"	9.66
0.25	" "	0.12	"	6.24
Through 0.12			"	39.52
				1000.0

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Table 23. Efficiency of Stamps Co

	Wt of stamp, lb per sq in of shoe area	Velocity when striking blow, ft per sec
California stamp.....	15.38	6.62
Colorado stamp.....	13.95	8.97
Leavitt steam stamp...	18.55	20.00

Small steam stamps have been designed for g
TREMAIN stamp, of which there are two in a morta
to 8 in. Mortar is 12 by 20 in inside; shoe and di
9 in high weighs 112 lb. A new die weighs 62 lb an
lip 2 in above it. A chuck block may be used to r
iron-wire screen, sloping 11° to vertical, lasts 150
125 tons. The stamps drop alternately and revol
Shoes and dies lose 10 lb per 100 tons crushed. Th
contained, 7 ft 6 in high, with a base 21.5 by 23.5
to 8 days. Cylinders have an annular piston, unde
stroke, upper surface 23.78 sq in. When near top

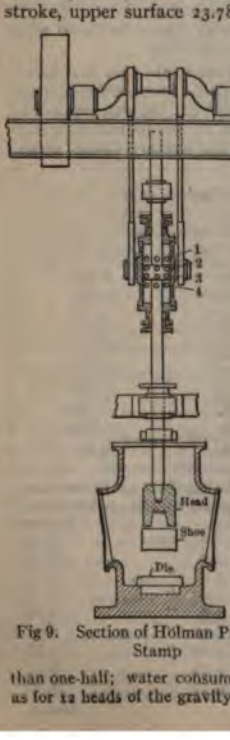

Fig 9. Section of Holman Pneumatic Stamp

a port which sl
from under to
crushing stroke
sure, 140 strok
strokes per mi
8 to 20 tons per
10 hp. Boiler
wood runs sta
12.5 tons water
amalgamating
and only 4 ton
sloping 2 in per
gamator and re

Holman pneu
man Bros, Cambo
The stem passes t
by 2 connecting ro
being attached t
is thus interpos
stamp, improving
high speed (up to
jury. Above and
in cylinder walls.
trapping air abov
and 4 are plugge
wear, these holes
are plugged. Th
is thus regulated
the pneumatic, as
that: first cost, w
foundation are s
nance is consider
than one-half; water consumption is two-thirds; power
as for 12 heads of the gravity type, as stated in Holman

7. Gravity Stamps

ons. A trench is dug to bed rock and 30 in or more of concrete the concrete are placed 2 planks spiked together, on which the ck is built of timber or planks bolted together and standing on end. ic concrete foundation and mortar block, Fig 11, has been adopted

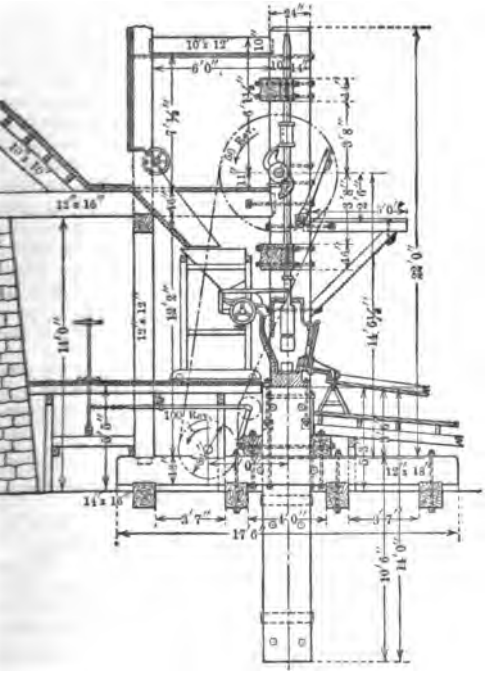

Fig 10. Stamp Mill, Side Elevation (25)

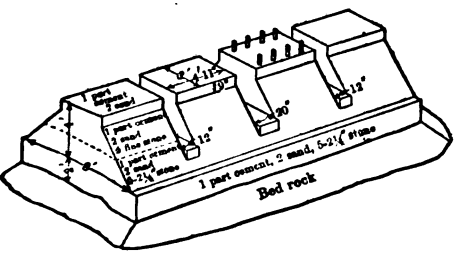

Fig 11. Concrete Mortar Block

Mortar block is trued up, extending somewhat beyond flange of knesses of blanket soaked in tar are laid upon it, and mortar

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bolted down. MUD SILLS, for battery frame, are
are cross sills, on which stand 2 posts for carrying c

Battery frames are of many designs. Fig 1
with plenty of light and space. GUIDES are also
are best (
14) wear i
durable.

Mortar
issue, with
inside, wit
Some hav
ing; othe
ing. Dou
screen, ar
large caps
mortar (
nage. M
are often
ings of 2-i
in the sp
at rear of
crushed r

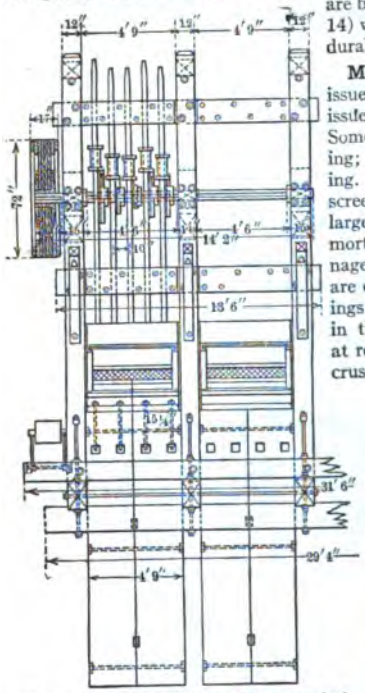

Fig 12. Stamp Mill, Front Elevation (25)

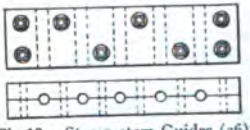

Fig 13. Stamp-stem Guides (26)

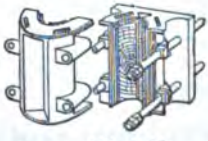

Fig 14. Allis Guide

are mounted on wooden frames, wedged into place with gunny
; to prevent leaks. They are commonly of punched steel plate,
nward. Coarser size, 1 mm; finer, 0.35 mm. Round holes are
eferred, as producing less variation in size of grain; but, where
hr-punched, the slot must be used for fine sizes. For coarse crush-
nan 1 mm) a slot may be preferred. Steel-wire cloth screens have
centage of opening than punched plate; they cost less, but are
. Brass cloth is best where water is acid. Percentage of opening
same for wire screens of all meshes, if the
equals the width of hole, but in plate screens
e of opening diminishes rapidly on finer sizes.
eld is placed outside of screens to prevent

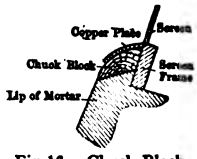

Fig 16. Chuck Block

amonest SLOPE OF SCREENS is 10° from ver-
K BLOCK (28) is of wood, for adjusting height
of screen (Fig 16). Having several screen
different heights of chuck block, the HEIGHT
E (vert distance from top of die to bottom
n be maintained fairly constant. This is one of the most important
f a battery. Inside amalgamated plates, if used, are placed on chuck
imes also in rear of mortar.

, and boss (Fig 17, 18, 19). Dies are cylindrical blocks with a
set in bottom of mortar for taking wear of the stamps (28). False
sometimes placed under dies, to maintain height of discharge, but
are more easily adjusted. The SHOE is the wearing part at lower
end of stamp. It has a coned shank, which
enters a socket in the boss (Fig 19), a firm
joint being made by placing wooden wedges
around the shank and dropping boss onto
them. A socket in top of boss receives the
stamp stem, the joint being made by wrap-
ping cotton cloth around coned end of stem
and dropping it into boss. Boss has 2
key-ways for wedging off stem and shoe;
of cast iron, with wrought-iron bands shrunk on to prevent split-
nes of low-carbon or manganese steel. STEMS are wrought iron
12 to 15 ft long by 2⅞ to 3 7/16 in diam, coned at both ends to fit
may be reversed. Keyed on stamp stem is the TAPPET, on which
below) acts for lifting the stamp. Tappet is reinforced by ribs
om; also, on one side, against pressure of gib keys. GIB is set
stem by 2 keys, to prevent tappet from slipping. The Wheeler
are of cast, chrome, or manganese steel.

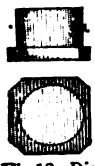
Fig 18. Die

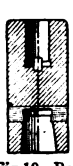
Fig 19. Boss

es and dies. They are of chilled iron and different steels, cost-
per lb. Shoes wear 1.2 to 1.8 times as fast as dies. Rotation of
ends to keep the wear even. Life of shoe varies from 84 to 900
of die, 80 to 600 tons (Richards' "Ore Dressing," p 182).

is of iron or steel, 4.5 to 6 in diam by 12 to 14 ft long, for 2 bat-
mps each; it has a short key-way for driving-pulley and a long
there is a collar at each end. Bearings are generally babbitted,
tted soft cast-iron boxes have some advantages.

have a spiral face, developed as shown by Fig 20. The radius of
circle is the distance from center of cam shaft to center of stem.
f the curve at cam tip is changed to a circle, to prevent cam from

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FINE CRUSH

8. Chili Mill

Construction (Fig 21). There is an annular pan-sh
heavy wheel-like rollers move in a circular path, crushi
of roller does true roller work, while outer part of tread s

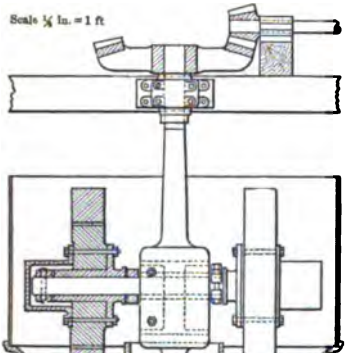

Fig 21. Section of Chili Mill

It is t
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and
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upon
socket
nals a
from
bevele

Qu
mill i
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stamp
which
to pas
Trent
throu

Table 28. Sizing Test of 5-ft Tre

On 4 mesh (over 3.99 mm)	
Through 4 on 6 mesh (3.99 to 2.79 mm)	
" 6 " 10 " (2.79 " 2.01 ")	
" 10 " 14 " (2.01 " 1.40 ")	
" 14 " 20 " (1.40 " 0.99 ")	
" 20 " 30 " (0.99 " 0.61 ")	
" 30 " 60 " (0.61 " 0.25 ")	
" 60 " 120 " (0.25 " 0.119 ")	
" 120 " (0.119 mm)	

Table 29 gives results of test on products
running 40 rev per min, and crushing 4 tons pe

Table 29. Comparison of Rolls

Through 1 on 4 mesh (25.4 to 3.99 mm)	
" 4 " 6 " (3.99 " 2.79 ")	
" 6 " 10 " (2.79 " 2.01 ")	
" 10 " 14 " (2.01 " 1.40 ")	
" 14 " 20 " (1.40 " 0.99 ")	
" 20 " 30 " (0.99 " 0.61 ")	
" 30 " (0.61 mm)	

. A 5-ft Bryan Chili mill, crushing 25 to 35 tons in 24 hr, had rollers 44 in ace, and made 37 rev per min. Each roller weighed 3 650 lb; tire, 1 100 lb; 50 lb. The mill used 400 to 1 000 gal water per hr when crushing jig mid- required 10 hp.

Table 30. Capacities, etc, of Chili Mills (33)

Maker	Type	Tons per 24 hr	Rev per min	Total weight, lb	H p required	Price
Seaver-Morgan Co	5-ft Akron.....	25	35	32 000	6–8	....
	6-ft "	50	30	48 000	6–8	....
Iron Works Co....	10-ft Lane.....	40	6	.....	12	....
Seaver-Morgan Co.	6-ft Garfield....	100(*a*)	....	.....	6–8	....
lmers.....	6-ft Cole.....	...	....	55 000	30–40	$35 00
g Co.....	6-ft Monadnock	75	33	.....	55	....

(*a*) Soft porphyry ore, from ¼ in to 40 mesh.

9. Huntington Mill

a fine crusher of the roller type, the rollers having vertical axles sus- flexible joints from a central spider, thus allowing them to swing out gal force against a ring die in rim of pan. The ore, fed with water, is ainst the die, crushed by the rollers, and can not leave the mill until gh to pass through the screen. Like the Chili mill, the Huntington y adapted to recrushing middlings. It has to some extent replaced mill for gold ores, but in preparing for amalgamation it is inferior to it does not burnish the gold particles so well.

tion. The pan has a central hollow post, through which a vertical shaft wer through bevel gears to rollers. Above the ring die the rim carries screens ree-quarters of the circumference, and above the screen the sides rise to height . The feed spout, which is level with the screens, occupies about one-quarter Around the screens is a ring launder and spout for discharging pulp. The have T-shaped tops and bearings, which allow the rollers to swing freely nst the die. The centrifugal crushing force increases with speed of rotation. nd die have replaceable shells. Roller shells wear from 1.5 in to 0.5 in; die 2 in to 0.75 in. Roller shells of Latrobe and Midvale steel last an average die rings, 75 days.

screens. Though the mill can crush coarser sizes, it does best work on feed 6.3 mm) maximum grains. Horizontal-slot screens are better than those or square holes, because of the tangential path of particles passing through. screen has 1.5 times the percentage area of opening of the punched slot; orded 14-day test, the woven slot did poorer work than the punched screen, .43 tons per 24 hr, against 73.57 tons from the punched slot. Difference was ding" of the woven screen.

Table 31. Cost of Crushing in Huntington Mills (34)

(In cents per ton.)

	3.5-ft mill(*a*)		5-ft mill	6-ft mill
.....	2.30	Operating labor.....	1.56	1.56
.....	3.94	Repair labor.....	0.78	0.52
.....	0.13	Repair supplies.....	3.74	3.07
.....	0.94	Power.....	2.01	1.34
e.....	0.30		8.09	6.49
il, etc....	2.17			
	9.78			

(*a*) Crushing 15 tons per hr through 1/40 in (0.63 mm).

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Capacity, power, and water depend chiefl
feed, and of crushed product. Table 32 shows
passed through 0.75 in, and was crushed to pass

Table 32. Capacity, etc, of Hur

Size of mill	Tons per 24 hr	Rev per min	Water p hr, gal
3.5-ft.....	8–12	90	750
5 ".....	20–25	70	1 000–1 2
6 ".....	40–50	55	1 400–1 7
6 " (Anaconda)..	60–75	55	1 500–2 0

(*a*) Mills also made sectional, with no pi

Table 33. Screen Test of Feed and Product

Screen, size	Feed	
	% of total material	%
On 8-mesh.....	1.41	
" 10 "	2.73	
" 20 "	61.00	
" 40 "	28.06	
" 60 "	3.54	
" 80 "	1.79	
" 100 "	0.74	
" 150 "	0.26	
" 200 "	0.26	
Through 200 "	0.21	

10. Tube-mill

Uses. Regrinding stamp-mill sands, after
preparatory to cyaniding; regrinding smaller
for retreatment; grinding cement clinker.

Construction. There are two types, cylindrical
mill is a steel cylinder with C I ends, supported by h
one trunnion and pulp discharges through other

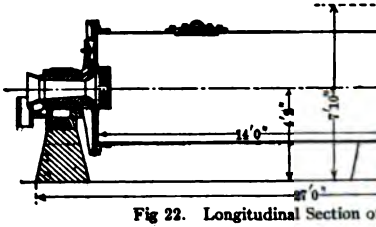

Fig 22. Longitudinal Section o

larger, to cause pulp to flow in that direction (somet
ing spiral, for removing the pulp; in which case a s
is operated by a driving pinion, engaging with a t
Cylinder is lined with silex bricks, or with flint pebb

al iron ribs, or with steel or chilled-iron blocks. It is charged with loose flint ebbles (lumps of hard, tough ore are sometimes used), the cylinder being kept pebbles as possible, without their discharging with the pulp. New pebbles are day to make up for wear. Grinding is effected by: falling of the pebbles as es, crushing between pebbles, and crushing between pebbles and lining.

es and lining. Greenland flints have proved best; cost $20 per ton; 1 ebbles will crush 1 ton of sand. Cheaper pebbles are also used, of which will crush 1 ton of sand. From ½ to 1 lb of liners will do the same. charge for a mill 5 ft diam by 14 ft long is 4 tons of pebbles, costing 4 per ton, according to quality. Cost of silex lining varies, with quality, or $1 per sq ft, when 2.5 in thick, to $1.60 per sq ft, when 4 in thick. practice shows a service of 8 months on 2.5-in lining, 1 year on 4-in Price of chrome-steel lining for ball mills is 6¾¢ per lb.

mill feed is usually 8-mesh and finer, but a size of not over 0.02 in t results. For cyanide process, Argall gives this rule: Breaker to 2 in; .75 in; rolls to 0.24 in; rolls to 0.06 in; grinding pan to 0.02 in; tube- 0.0025 in (200 mesh). Fineness of tube-mill crushing is largely con- y rate of feed; the faster the feed, the coarser the product. Highest is attained on pulp containing about 39% water.

and power required. A 36-in tube-mill makes 36 rev per min; rev; 60-in, 28 rev; 72-in, 25 rev. Power varies with size of mill and f pebbles used. Davidson gives the rule: Hp = 0.15 × (volume of in cu ft); and suitable weight of pebbles in lb = 44 × (inside volume ill in cu ft). Experience with the Hardinge mill in Lake Superior mills a saving of about 10% in cost of installing power transmission by use lual motors, as against line-shaft method (107).

y of pulp. The ordinary overflow discharge has the disadvantage, with the spiral lift discharge, that the product delivered is classified; ts maximum grains of valuable heavy mineral are smaller than the grains of light waste, because they are of equal settling velocity. shows results at Kalgoorlie, where a classifier returned coarse ma- ube-mill for very fine grinding.

Table 34. Screen Test of Tube-mill Product, Kalgoorlie (35)

Size	Feed	Pulp from tube-mills	Finished product
40 mesh.....	4.5%	0.1%	.....
60 "	9.5	2.0	.....
80 "	6.8	4.2	0.2%
100 "	13.1	12.3	0.5
150 "	19.0	20.3	4.0
150 "	47.1	62.8	95.3

ty varies with hardness of material, size of feed and of product, and pebbles (Table 35).

Table 35. Capacity of Tube-mills (35)

	Diam	Length	Tons, 24 hr	From	To
, Col....	5 ft	22 ft	125	10 mesh	100 mesh
ato, Mex.	4 ft 6 in	20 "	60	26 "	70% thro' 120 mesh

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Marathon mill (112) is 7 ft long by 3 ft diam, line with 4 480 lb of c-i plates, and loaded with 7 000 lb c steel rods, ½ to 2 in diam, extending full length tube. Advantages claimed are: high mechanical eff ciency and uniformity of product.

Efficiency of tube-mill. Discussion is to lengthy to insert here. They have nearly dis placed all other machines for fine grinding. (Se Richards' "Ore Dressing," p 1292.)

Table 37. Wet-grinding Tu

Size, ft	Weight, lb	Approx cost
5 ×22	57 000	$2 600
5 ×16	48 000	2 350
5 ×12	43 000	2 200
4.5×20	42 500	2 300
4.5×12	33 000	2 000
3.5×20	26 500	1 700
3.5×14	20 000	1 450

Above mills equipped with standard spur-gear d lining and spiral feeder.

Hardinge tube-mill (Fig 23) has the form o about 2 ft of cylindrical surface between them.

At feed end the slope is about 60° to the vertical, at with mixture of large and small pebbles; the larger seg climb up nearer the axis of rotation. Hence the co

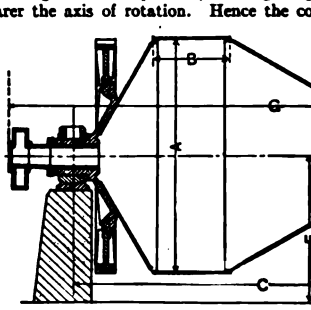

Fig 23. Hardinge Tube-

ore, the smaller on the finer ore. Steel balls may claimed for this design that when the ore is ground suf and that a particle must be of the desired fineness be 2-in size can be reduced to 16 or 20 mesh in one oper ing, feed end may be raised 1 to 2 in above discharge cast chrome steel, weighing 4 100 lb ($275–300) for a 6-ft mill. Linings for pebble mills are of 3 kinds: (a) set in neat cement mortar. (b) Cole pebble lining; fli in neat cement mortar. (c) Britannia lining, of short 1¾ in long, set on end in bed of mortar (1 cement: 2 s ing. Operate for several hours with half-charge of lump

Table 38. Dimensions of Hardinge Mills (Fig. 23) (107)

	C	D	E	F (a)	G	Pulley	Weight lb (e)	Price (d)
7"	6'- 9"	13"	4'-0 1/2"	6'-1 1/2"	9'- 8 1/4"	30" × 10"	10 600	$1 550
5	6 - 9	10	3 -1 1/8	6 -1 1/2	7 -11 3/8	28 × 8	4 000	1 050
2	11 - 0	13	...	...	...	...	8 000	1 215
5	8 - 5	(b)15	4 -6	8 -2 1/8	11 - 9 1/2	42 × 14	17 700	2 480
2	8 -11	13	4 -0 1/2	8-1 3/8	11 -10 1/4	40 × 10	8 000	1 500
3	11 - 1	13	4 -0 1/2	8 -2 3/8	14 -0 1/4	42 × 12	8 750	1 550
2	13 - 1	15	5 -0	10 -2 1/4	16 - 6 1/4	42 × 14	11 000	2 250
2	11-11 3/4	(b)15	5 -0	10 -2 1/4	15 - 4 1/4	42 × 14	11 500	2 250
1	13 - 5 3/4	15	5 -0	...	16 -10 1/4	42 × 14	...	...
8	13 - 1 1/8	20	6 -6	...	17 - 1 3/8	...	...	...

mill. P = Pebble mill. (a) Over-all horizontal width to outer side of ur shaft; length of pedestals is about the same. (b) Cast steel; others are (c) Special. (d) f o b Braddock, Pa. Price of ball-mill includes liners at- e of pebble-mills does not include liners nor pebbles. (e) Weight of ball- s lining; weight of pebble-mills does not include lining nor pebbles.

Table 39. Screen Tests of Hardinge Mill Products (107)

	I		II		III		IV		V		VI	
	Feed	Prod	Feed	Prod	Feed	Prod	Feed	Prod	Feed	Prod	Feed	Prod
10-mesh	94 1	2 1	60 2	0 4	71 4	0 0	65 7	...	76 1	10 3	40 1	...
20 "	2 4	8 0	28 7	5 3	13 [illegible]	3 20	2	...	7 3	20 3	17 7	...
40 "	3 5	22 8	6 9	17 0	7 1	23 9	4 3	3 7	3 0	11 9	17 7	5 5
60 "	...	11 0	1 9	18 3	2 0	11 7	0 5	8 5	2 7	11 7	2 5	12 5
80 "	...	12 0	...	...	0 7	5 0	0 1	3 3	0 8	3 4	2 5	12 5
100 "	...	4 0	1 2	24 8	2 4	4 4	...	9 4	0 8	2 8	...	14 5
150 "	...	10 1	1 1	35 0	0 2	3 5	...	17 2	0 9	4 1	...	5 5
200 "	...	...	...	...	0 5	5 1	...	10 0	1 0	4 3	...	12 ...
200 "	...	...	...	...	5 8	31 7	...	31 0	...	...	...	...
...	...	...	...	...	...	...	...	...	1 1	3 8	...	...
d ...	...	...	...	...	...	...	...	...	6 1	18 2	...	...
, tons per hr	...	2	...	10 5	...	8 3	...	2 7	...	14 2	...	...
ons	...	B 2	...	B 1 1	...	P 5 25	...	P 5	...	B 4	...	...
tion, lb per ton	...	...	...	0 72	...	2 4	...	2 5	...	...	...	...
o m ...	...	33	...	28	...	24	...	28	...	28	...	...
p ...	...	15-17	...	38-40	...	55	...	...	...	...	...	...
wt of pulp	...	50	...	40	...	60	...	64	...	...	...	...

P, Pebbles. I. Ball-mill, [illegible] ft, Nipissing Porcupine Mines, 1912. Quartz and basalt ore from coarse crusher at [illegible] rtzose jig tailings, through 1-in screen. Total cost [illegible] Pebble-mill, 8 ft, Arizona Copper Co, Morenci, [illegible] ened product from rolls at 1/2 in. Total cost [illegible] -mill, 8 ft, Copper Range Consol, Freda, Mich. Co [illegible] through 14-in screen. V. Roll-mill, 6 ft, Mines [illegible] z. Porphyry, from coarse crusher at [illegible] 8 ft, Federal Min & Sm Co, Morning, Idaho. Fe [illegible] through 4-mesh.

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11. Ball-mill

Construction. Ore is crushed as in the tu balls. Mill consists of a short cylinder with lined with C I or steel slotted dies, or wearing the slots and openings between the plates (F

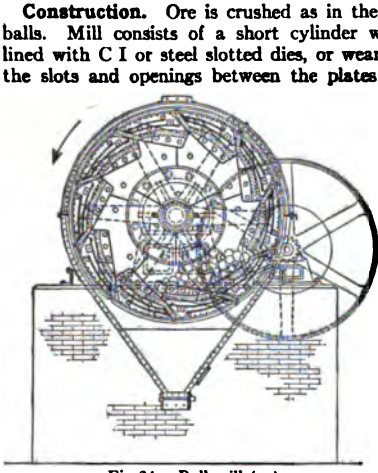

Fig 24. Ball-mill (37)

with calcareous and silicious material, of which 3% w than 2 in, yielded 33 tons of 20-mesh product per 24 24 hp. One steel ball, weighing 18 lb, was added daily. Grinding plates and side liners were replaced every 7 or 8 months, at cost of about $900. Table 40 gives a sizing test of product.

Marcy ball-mill (113) is a cylindrical tube-mill 8 ft diam by 6 ft long, with iron or steel lining, and loaded with 5-in steel balls. At Inspiration mine, capacity per mill was 400 tons per day, crushing from 2-in max to 60% through 200-mesh, consuming 10 kw-hr per ton.

Table 41. Capacity, etc, of
Crushing Portland Cemen

Wt without balls	Wt charge of balls	Capacity, bbl per hr, to 20 mesh
29 500 lb	3 000 lb	12–16
41 100 lb	4 500 lb	18–24

Note. — From 100 to 120% added power is req When pulverizing to pass all through 20-mesh, from

12. Grinding Pan; Clea

Uses. Pans were developed and formerly amalgamating silver ores; today their special grinding for cyanidation. Argall states that th in down to 0.02 in, or will do the whole work f

on. Fig 25 and 26 show a 5-ft pan. The cast-iron bottom has a flange around which are riveted the sheet-steel sides, 24 in high. In a hollow cast-iron bolted to bottom of pan, revolves a vertical shaft. On the bottom, around the grinding dies, set radially like sectors of a circle, with narrow spaces be-

Lugs on under side of into recesses in pan bot- ted with a spline on the is the muller, the lower is a horizontal disk, to es, shaped like the dies, d. The muller can be ered by a hand wheel, pace between shoes and mixing only is needed, ised; for grinding, they o touch dies. A lock nut nd wheel holds the ad- The muller shaft is driven eled gears under the pan ve.

s driven radially outward spaces between shoes and es on sides of pan. To ough mixing, 3 curved tached to sides of pan ak the flow of pulp and ards the center. Large he muller admit pulp to f shoes and dies. This is of prime importance.

made 3 to 8 ft diam, a common size. The arged at top through d discharged through gots at different heights m as needed. Run- er charge, quantity of density of the pulp, termined by trial. If thick, consumption of xcessive; if too thin. the dies is imperfect, at 45 rev per min, using 7 hp, ground 30 tons in 24 hr. Shoes and to 6 months. Pans can be run intermittently, or continuously in discharging into another.

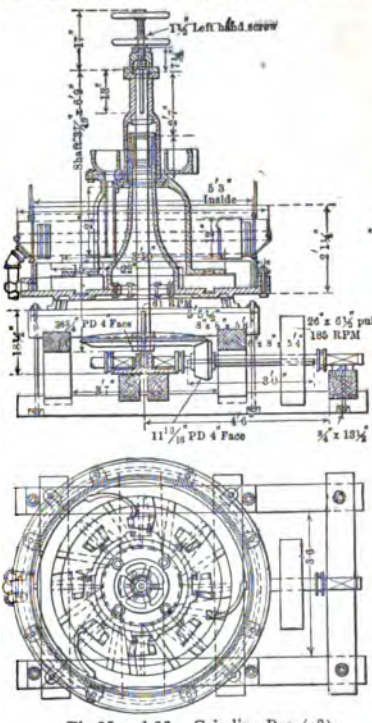

Fig 25 and 26. Grinding Pan (38)

le 42. Sizing Tests of Grinding Pans at Ivanhoe Mill (38)

Size mesh		Feed pulp	Discharge pulp
Through	On	%	%
.....	40	31.4	0.6
40	60	21.8	6.2
60	100	17.6	28.7
100	150	6.1	8.0
150	.....	23.1	56.5

barrels (Fig 27) are cast-iron cylinders, usually 4 ft long by 2 ft shell 2 in thick, revolving on trunnions.

Tight and loose driving pulleys are mounted on a
There are 2 hand-holes for charging and dischargin

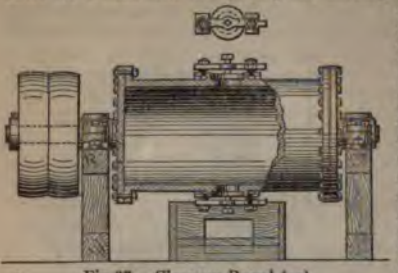

Fig 27. Clean-up Barrel (40)

gam and pyrite tailings, which are taken to clean-up
balls and scrap iron are removed, the pyrite residue
another charge.

PRELIMINARY TR

13. Washers

Log washers are for washing and separating
from lump ore. They consist of an iron or wo
2°45′ to 6°, in which revolves a heavy wooden, i
ing iron blades set obliquely to the axis, form
Ore is fed near lower end, wash water at upp
carried slowly to upper end of trough, where
and wash water cutting and cleaning from the
clay material, which runs to overflow at lower

Length varies from 12 to 31 ft; width 16 to 36 in
machines, and 36 to 88 in for double washers; depth
They are usually run in pairs, at 12 to 18 rev per min
capacity being 220 to 440 tons per 24 hr. Feed wate
gal per min. Operating cost (labor, power, and water
per ton of product, averaging say 10¢. A cross-sec
Longdale log, used in washing limonite at Longdale, V
in Fig 28. The blades, of chilled iron, are put on i
180° apart, with a pitch of 5 ft. Cost of one complete
ing 2 566 lb, 1.25¢ per lb. Range of cost, say 1 to
6.25 hp was required for a log 17 ft long, 11.5 in di
a circle 28.5 in diam. Cleaned lump ore, from washin
to remove quartz pebbles. Jigs have been used for

Turbo log washer has 2 logs, set in a troug
of steel plate, up through which water is force
material and assists in washing. This is use
finer sizes of iron ore. A turbo washer, 18 ft l
screen, requires 3 hp and 150 gal water per mi

14. Hand Pickin

Hand picking underground saves cost of tram
nishes material for filling, but may cause loss of
Hand picking in ore-house may save lump ore fo
sliming in mill), and by rejecting waste, the cost
centrating is reduced; mill capacity is also increas

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st of crushing, by removing clean mineral, and is sometimes adopted into 2 or 3 classes, to receive separate treatment. Sledges, and spalling hammers (Art 3) may be used in breaking ore and hand picking.

lts and tables. Picking belt receives ore from breaker, and conveys it ys or men, stationed on one or both sides, pick out certain materials, for e rock, rich mineral, or blende middling. Annular revolving tables, about d at one point and discharged at another, are sometimes used in Europe.

and picking. On the Rand (42), in 1895, hand picking cost about yielding 10 to 30% waste. Cost of gold milling was then $1.10 yaniding, $1.24 per ton. Thus, saving was about $2.20 per ton of l out, beside saving transport of waste. The oversize of a grizzly hand picking than run-of-mine, as fines do not obscure lumps. of size for hand picking may be 1 or 1.5 in; rarely less than 1 in, ntity picked drops, and cost per ton increases rapidly.

purely theoretical, supposing a boy to pick as fast as he can all day. This, brief mill test, will assist in determining cost of, and saving by, hand picking.

3. **Estimated (Theoretical) Cost of Hand Picking Galena** (43)

	Weight of a single lump of galena, lb	Time to pick one lump, sec	Wt of galena picked in 10 hr, lb	Cost per ton picked out. Wages $1.00 per 10 hr
es.....	58.46	42	50 100	$0.040
.....	33.83	24	50 750	0.039
.....	17.321	12	51 970	0.038
.....	7.308	5	52 620	0.038
.....	2.165	3	25 980	0.077
.....	0.9134	2	16 442	0.122
.....	0.27061	1	9 743	0.205
.....	0.11421	1	4 112	0.486
.....	0.03383	1	1 218	1 642

SIZING AND CLASSIFYING

15. Screens

Stationary, vibrating, and revolving screens; made of bars (grizron perforated plates, punched plates, or woven-wire cloth. The naining on the screen is the that which passes through, ize."

ars (Fig 29) are best of taper revent choking; sometimes of -section, such that when worn ll be minimum waste. (See .) Bars are threaded on rods d by thimbles. Their slope the angle of friction (usually 32°); when less than 40°, the netimes be helped forward by y are generally used for separom breaker feed. Table 44 of grizzlies from practice (45). Grizzly dimensions are determined de of feeding and using, than by tonnage treated; capacity should ed demands.

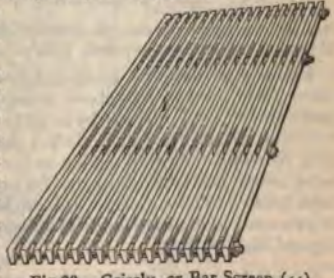

Fig 29. Grizzly, or Bar Screen (44)

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Table 44. Details of Grizzlies

Slope	Length	Width	Bar		Space	S
			Width	Depth		
	ft in	ft in	in	in	in	
35°	16	4 6	2	3	2	...
(*k*) 4°	5 1	1 7	....	....	1¼	{ 8 { f
Horiz	2	2	½	¾	3	Re
45°	12	4	½	3½	1⅜	
40°	9 6	3 6	⅝	2	1½	

(*k*) Vibrating gri

Edison slotted screen is applicable to fine wear is small (45). For coarser screen, slots ar

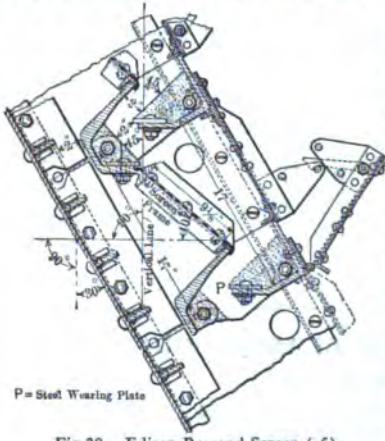

Fig 30. Edison-Rowand Screen (46)

ugal force against upper part, the oversize being colle thrown against screen cone below. It has screened 1

Ferraris screen is a horizontal box screen, suspended by rods above, or supported below. It is vibrated by an eccentric, making 350 1 to 1.25-in strokes per min, resulting in upward motion on forward stroke and downward motion on back stroke. This propels oversize rapidly forward, and makes for clean screening and high capacity. Over

At Monteponi, screen is 13 ft long by 27.5 in wide, having 14-mm holes on first half, with 20 and 30 mm on remaining quarters. Fines are rem the 14-mm screen. A second screen, of 5-, 7-, and 14-mm. For sizes smaller than 5 mm, the screen

31.) In this case, at Sulitelma, Norway, screen slopes up at discharge end to tank, followed again by short length of horizontal screen. This does excel- sizes down to 0.5 mm. Undersize discharges from tank through a spigot.

ni, these screens replaced ved 9 ft 10 in of mill height, city and required less power. slight.

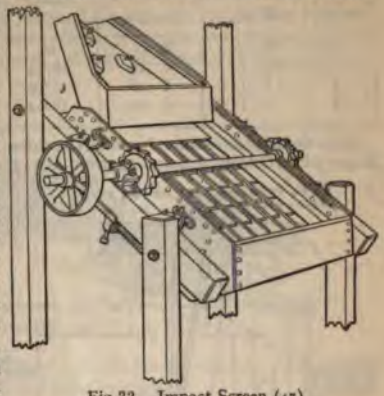

Fig 32. Impact Screen (47)

screen (Fig 32) is rec- ith about 30° slope, flex- ted and free to move up t right angles to plane of is vibrated by 2 ratchet These give the downward opposing springs giving , which is stopped by fers. Can be run dry or ct screens are made in 2 ft and 3 by 4 ft, prices being respectively $200 and $250 and 800 lb. re 0.25 hp. At Utah 's mill, this screen re- mels, greatly increasing y, at operating cost of ½¢ per ton, for labor, wire cloth and repairs, .5¢ for trommels.

. Approximate Capacity of Impact Screen, Tons per 24 hr (48)

	2.5	4	8	14	30	60	90
rsize per 24 hr.....	250	200	150	100	60	25	15
%.....	97	95	92	90	85	75	70
rticles in feed.....	1.5 in	1 in	0.5 in	4 mesh	8 mesh	12 mesh	30 mesh

Fig 33. Newaygo Separator

Newaygo separator (Fig 33) is set at steep pitch (say 45°), and is vibrated by very light hammers 12, hinged in pairs to rapidly revolving shafts 21. Hammers strike on small posts 13 on screen cover, vibration being transmitted to metal strips 5 on screen, which is tightly stretched (47). Field of work covers moderately dry pulverized materials, even as fine as clay. Capacity said to be 5 000 to 40 000

Screen is fed through hopper 15, in a continuous thin stream, by screw . Price, $443 to $608. When extracting fine ore from a coarsely d, it is advisible to protect the fine screen by adding a coarser one Special designs of this machine will deliver 2, 3, or 4 sizes at once.

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Table 46. Specifications of Newayg

Approx dimensions			Pulley	Sp re r
Depth	Width	Height		
ft	ft in	ft in	in	
5	5 1	6 10	18×4	1
5	7 1	6 10	18×4	1
5	9 1	6 10	18×4	1

Other vibrating screens are: Imperial, Vibracone, Tray Mayer riddle, Columbian pulsating screen, Coxe gyratin

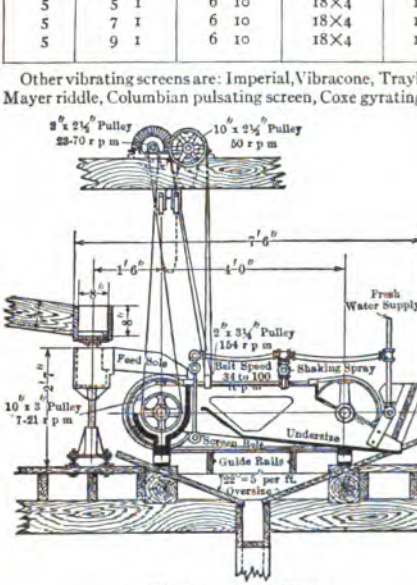

Fig 34. Callow Screen

capacity, when feed contains 50% oversize, is given i vibration, the only repairs necessary are screen renewa

Table 47. Capacities of Callow

20 mesh	250 tons feed per 24 hr		80 mesh		
30 "	200 " " " " "		100 "		
40 "	150 " " " " "		150 "		
60 "	125 " " " " "				

5 000 tons, and costs $125 for meshes between 22 and 1¢ per ton. Feed water varies from 3 parts water to 1 should not exceed 120 gal per min, for treating 20 or 30 Machine does good work, considering the fine sizes for ing from 5 to 13% fines. Table 48 gives details of dim

Table 48. Callow Traveling-

Type	Extreme length	Dimensions, in	
		Width	Height
Simplex	96	48	32
Duplex	96	76	32

16. Trommels

ing screens or trommels (Fig 35) are today more used than any ough Ferraris and impact screens may take their place. A sloping, cylinder, of screen cloth or plates, if fed at upper end, will convey the y a helical path to lower end. Undersize drops into the housing be- ize being delivered at lower end. Results aimed at are: large capacity,

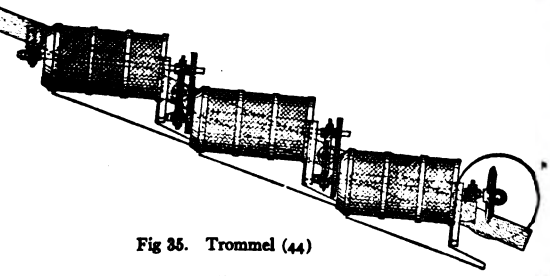

Fig 35. Trommel (44)

rfect a separation as possible of oversize from undersize. Factors in- e: length, diameter, slope, and rotative speed.

The longer the trommel, the greater its capacity, and the cleaner the sepa- it, as increased length does not decrease wear of screens, nor increase their ger the trommel, the greater the cost of replacing screens. Moreover, a very el has a thicker bank of oversize passing through it, not only increasing wear hat hindering the screening, but also requiring more driving power. Lengths from about 3 to 12 ft. The writer believes 9 ft to be about the best length.

r. A large diameter gives a thinner bank, therefore screens more freely; ar is less, because a smaller proportion of the screen takes the scour at any But, large diameter increases first cost, and cost of replacing worn screens The question in each case is whether saving from increased length and di- re than offsets the extra cost. Diameters are usually from 24 to 50 in, and believes about 42 in is best.

most important. Flat slope makes for long helical path, increasing time for ough screen; the bank is therefore thicker, scour of sand on metal greater,

Capacity in 24 hr of a Trommel 36 in Diam, 72 in Long, at 20 Rev in, for Given Depths of Bank, and at Different Slopes; Assuming 1 cu ft Broken Ore Weighs 94 lb (50)

f trommel	Ore bank, ¼ in deep. Trommel contains 3.91 lb ore (*a*)	Ore bank, ½ in deep. Trommel contains 11.03 lb ore (*a*)	Ore bank, 1 in deep. Trommel contains 31.06 lb ore (*a*)	Ore bank, 2 in deep. Trommel contains 87.14 lb ore (*a*)
In per ft	tons	tons	tons	tons
⅜+	4.6	13.1	36.8	103.3
½+	5.8	16.4	46.1	129.4
¾−	8.1	23.0	64.7	181.6
1+	11.7	33.0	92.8	260.3
1½−	16.5	46.4	130.8	366.9
2	22.6	63.8	179.6	503.9
3	34.5	97.4	274.4	769.8

(*a*) At any one time, and including the undersize.

and more power is required. Steep slope shortens
lessens the scouring weight on screen plate and the po
of undersize, and lengthens life of screens. Excessiv
trommel too rapidly for efficient screening, unless le
mel has great capacity. Usual slopes are: 1°50′ (o.
Writer recommends, as best slope, from 9°30′ (2 in

Speed. If too slow, capacity is reduced. If too
rising side, and screening is interfered with; at very h
is carried over, and screening ceases. About 18 to 2

Construction. A 9-ft trommel, with 3-ft le
4 hubs on central shaft, 4 spokes screwed to eac
spokes, and 4 outside hoops with bolt tighteners
should be a short length of unpunched plate at l
cone at feed end. HOUSING, of metal or wood,
is operated. Its sides extend above center of
enough to convey undersize to discharge spout
collected and delivered. MATERIAL FOR SCR
chilled-iron sections cast with holes are advan
mm (1 in) and less, punched-steel plate or
Wire cloth should be double crimp, as the holes
cloth has larger percentage area of opening, is c
holes better, but is shorter-lived and blinds mo

Screens are subject not only to wear but also to c
sulphate, and carbonic acid. In Arizona corrosion is
is fed with the ore in sufficient quantity to neutralize

Driving mechanism. Approved mode is to
veled gears, and from the shaft of this to drive th

Power, feed, and capacity. POWER per tro
it is less for steep than for nearly horizontal ton
bank. FEED must be regular and constant. E
putting much undersize into the oversize. C
diam by 9 ft long, sloping 2 in per ft, safe fe
25-mm hole, 400 tons; 20 mm, 325 tons; 15 m
5 mm, 100 tons (deduced from Richards "Ore
is used as a spray on up-coming side of
perfectly, and helping to prevent blinding of sc

Cost of screening is approximately as fo
0.243¢ per ton; labor of changing screens, 0.1
0.130; lubrication, 0.020; attendance, 0.500;
series of trommels, the cost is greater, but not
(the largest item) may be the same as for a sin

Variations of design. HEXAGONAL or O
sometimes adopted, instead of cylinders; they si
and PYRAMIDS simplify driving connections, as
no beveled gears; but, the screen plate or cloth
cost and cost of repairs. Saving in gearing is
extra cost of screens. TANDEM SCREENS are so
of screen has a smaller hole and the second a la
of mechanism, but the finer screen wears out fa
than if screens were individual. Hence, tander
CONCENTRIC SCREENS. The coarser screen is i
not economical, as area of coarse screen should
the fine one, and the screens will not wear out sim

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arises whether to put back a partly worn screen with a newly replaced h does not lead to economy of repair time. For anthracite screening, trommels is frequently arranged thus: feed delivered to medium screen, andem with coarse screen; fine screen is concentric with medium.

al; single screen on a single trommel is preferable. SPIRAL SCREENS. The h of each screen cloth may be proportioned as required, but objection is that, le screen of a number wears out first, the trommel must be dismantled for re- question is then whether to use new and partly worn screens on same trommel.

creen (Fig 36), invented in Colo, has had its greatest success at Sulit- way. Screen is 5 ft diam by 3.5 ft long, of octagonal section, with inward eed, 12 rev per min. It is fed on the outside with wet ore, and through is washed . This de- tly reduces onsumption, repairs, and d separation

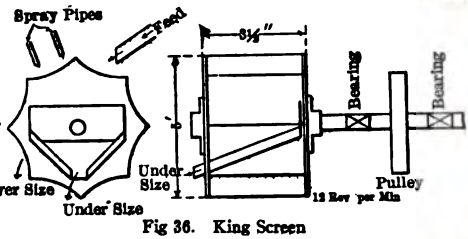

Fig 36. King Screen

l arrange- plest and best with coarsest d follow with rder of the size of hole. Each feeds its undersize to the next finer screen below. Dividing 2 groups, the coarser and the finer, and delivering the entire feed to the coarsest roup, sending its oversize to the coarser group, saves somewhat the diluting of he plan overworks the coarsest of the fine screens and complicates the mill. everal other schemes of screening, but all complicate the mill more than they work.

ale, screen sizes, screen analysis. Sizes of holes in screens, whether rcles, are measured in fractions of an inch or in millimeters (mm). The sizes of screens in a given mill form what is called the "sieve scale" of that mill. ld that there should be a definite ratio between these sizes, and suggested $\sqrt{2} =$ ratio for diameters of round holes or edges of square holes, giving a scale of: mm, 2 mm, and so on. The largest screen is first, the smallest last. For s this scale becomes 1 sq mm, 2 sq mm, 4, 8, 16, 32, and so on. An ore is d" when it has been screened first on the coarsest sieve, yielding oversize ze; and the undersize has been screened on next sieve, yielding oversize and nd so on down. The successive oversize products, and final undersize, ed, give the percentages of each size; and, if analyzed for the valuable metal, e "screen analysis" of the ore. Such a set of products is designated as fol- ct 1, through 32 mm on 22.6 mm; product 2, through 22.6 on 16, and so on. argument for a constant ratio is this: the difficulty of separating, in sub- centration, the smallest (22.6-mm) grain of galena, for example, from the mm) grain of quartz in the product "through 32 mm on 22.6 mm," is precisely the difficulty of separating the smallest (2.8-mm) grain of galena from the m) grain of quartz in the product "through 4 mm on 2.8 mm," and so on. hich seems wise and right, is not adopted in mills, because of the real or sup- ability of concentrating machines to treat almost any product brought to men therefore are inclined to adapt the sieve scale to the work of the mill; if, of two jigs, the one treating the coarser product is underworked, and er jig is overworked, on purchasing the next lot of screens they will get a r mesh, which will give more feed to the coarser and less to the finer jig. ther arguments also for changing the sieve ratio.

ests of trommels show how far from perfect is the screening done in e rush and over-crowding for maximum product is practiced, even if of screening falls off.

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Table 50. Sizing Tests of Trommel Products, I

Test sizes	Through 25 on 15 mm		Through 15 on 10 mm	
	Per cent	Cumulative per cent	Per cent	Cumulative per cent
31.9 –22.3 mm.	2.3	2.3	.....	.....
22.3 –16.0 "	23.0	25.3	.....	.....
16.0 –11.2 "	53.1	78.4	18.2	18.
11.2 – 8.02 "	17.0	95.4	51.6	69.
8.02 – 5.61 "	2.9	98.3	23.3	93.
5.61 – 3.94 "	0.5	98.8	3.7	96.
3.94 – 2.69 "	0.2	99.0	1.2	98.
2.69 – 1.89 "	0.1	99.1	0.7	98.
1.89 – 1.49 "	0.1	99.2	0.2	98.
1.49 – 0.945 "	.....	.....	0.2	99.
0.945– 0.667 "	.....	.....	.....	.....
0.667– 0.493 "	.....	.....	.....	.....
0.493– 0.371 "	.....	.....	.....	.....
0.371– 0.270 "	.....	.....	.....	.....
0.270– 0.158 "	.....	.....	.....	.....
0.158– 0.119 "	.....	.....	.....	.....
0.119– 0.073 "	.....	.....	.....	.....
Fines	0.9	.....	0.9	.....
Total	100.1	.....	100.0	.....

In Table 50, column marked "per cent" i
final undersize. "Cumulative per cent" is the
ponding screen in first column. Cumulative pe
result if each screen were the only one in the se

Shape and arrangement of holes. Square
larger products than round holes of same diam.
than square holes; they may even pass quite lar
holes may be laid out in rows at 90° or 60°, lat
than former. The larger the % of opening, th
The % of opening is the sum of areas of holes in
by that area and multiplied by 100. The %
high on coarse screens, but on fine it becomes
Screen plate should not be thicker than width
space between holes is large and the % of area
wire-cloth screens can hardly be less than 25%
width of hole, and it runs up to 40% or even hig

17. Classifiers

Classifiers are used to sort grains of different
a series of products of diminishing falling velo
subsequent treatment; also to separate sand fr
or slime. They deal with fine sizes better tha
screens in a mill, generally beginning at about

Principles. A classifier conveys sand and water i
pockets or depressions, from feed end to overflow

gots in the bottom, ranging from coarse in first spigot to fine in the last, the ing finest of all. Classification is improved by rising currents of water in op- the falling sands, graded from strongest current at first spigot to weakest in These currents prevent finer grains from being entangled with the coarser. An sing rising currents of clear water is a HYDRAULIC CLASSIFIER. The vertical 7) in which the rising current mainly acts is called the SORTING COLUMN. The t above the sorting column, which brings the feed, makes a rough separation, ws the light, rejected part, is called the POCKET. The enlargement below, which e clear hydraulic water, delivers the rising current to sorting column, and nd water to the spigot below, is called the PRESSURE BOX.

of hydraulic classifiers. They are capable of treating material om 8 mm (¾ in) to dust, but best work is done between 6.3 mm (¼ in) m (0.004 in.). For grains above 8 mm, the quantity of water in the xcessive; for grains finer than 0.1 mm, box classifiers (spitzkasten), nes and drag thickeners are best.

ydraulic classifier consists of a series of cones, one overflowing into used on products below 1 mm. Though often found in concentrating classifier violates a fundamental law of good classification; namely, elocity of rising current in any pocket, due to feed water and sorting- ater combined, must be equal to velocity in sorting column. If less, ns will be too heavy to rise with the overflow, but too light to settle ng column. These grains will form a troublesome bank, for which emedy is to slacken the rising current, and let down all grains which e into the overflow. This spoils the spigot product, because much also come down. To overcome this the cone may be fed by a plung- , which breaks up the bank and lifts it out; but some coarser grains in a previous spigot are carried into a following one by this device.

-cone hydraulic classifier, for products of 1 mm or less, attempts to the fault of the single cone by an inner conical baffle-plate, of slightly le than the outer, to equalize the rising velocity throughout. There extra adjustment for varying area of cross section of pocket; but ly one position giving a velocity uniform to the top.

ls vortex hydraulic classifier provides for all velocities by having ectional area of sorting columns computed to discharge the required

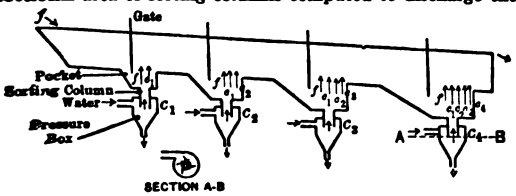

Fig 37. Richards Vortex Hydraulic Classifier

required velocity through each sorting column, and to lift the re- ume with required velocity in the pockets. Hence, no troublesome med. It can be used for sizes up to 6.3 mm (0.25 in), or even larger. od of computation for design of this classifier is shown in Table 51.

putations (54) involved in Table 51 are as follows: Line 1. Tons per 24 hr are obtained by a sizing test. Line 2. Feed and overflow are assumed. To get s of grains in the other products, we have 10 + 15 = 25 tons; 100 : 25 = 2.5 mm; 10 + 15 + 20 = 45; 100 : 45 = 2.5 mm: 1.15 mm; 10 + 15 + 20 + 25 = = 2.5 mm: 1.75 mm. Line 3. Kg per min are computed from tons per ultiplying by 0.631; thus 30 tons × 0.631 = 18.93 kg per min. Line 4. Ve-

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Table 51. Computation for Richards' Poc

	Feed	Spigot 1	Spig
1. Sand, tons per 24 hr...	100	30	
2. Sand sizes, mm.....	2.5–0	2.50–1.75	1.75-
3. Sand, kg per min.....	63.1	18.9	1
4. Rising velocities, mm per sec.....	....	139	1
5. Rising velocities, mm per min.....	....	8 340	6 1
6. Sand, liters per min...	23.9	7.15	
7. Water, liters per min...	252.40	21.45	1
8. Total pulp, liters per min.....	276.3	28.60	2
9. Sectional area, sorting column, sq mm.....	....	3 410	3 8
10. Diam of pipe, theoretical, in.....	....	2⅜	
11. Diam of pipe, practical, in.....	....	3	
12. Carrying current, sand and water, liters per min.....	....	290.60	30
13. Pocket area, sq mm...	....	34 600	48 9
14. " " sq in.....	....	53.6	7
15. " diameter, in...	....	7.32	
16. Head of water on spigot, ft.....	....	2.2	
17. Velocity due to head, mm per sec.....	....	3 627	3 7
18. Velocity due to head, mm per min.....	....	217 620	226 7
19. Spigot diameter, in....	....	½	
20. " area, sq mm....	....	127	1
21. " discharge, liters per min.....	....	27.638	2
22. Spigot water, liters per min.....	....	20.49	2
23. Spigot water, gal per min.....	....	5.413	

locities in mm per sec are taken from Table 52. W
or gangue; heavy mineral takes care of itself. Line
by 60 to convert seconds into minutes. Line 6. Li
ing ore to be wholly quartz, and then dividing the kg
This gives liters of sand. The error due to assumpti
liters per min. Assume that feed will have 4 time
and that the rising water for any given sorting colun
sand. Hence, multiply liters sand per min by 3 an
min for each sorting column. Line 8. Total liters
7. Line 9. Area in sq mm needed in section of sor
line 8 by line 5 and multiplying by 1 000 000. Lin
area given in line 9 is found by computation. Thes
for the treatment. Line 11 gives commercial size of
of line 10, allowing a little increase in diam for fricti
rising in pockets; computed from lines 6, 7, 8. Fo
subtract 7.15, giving 290.60; for the second, add
giving 302.56; and so on for the third, fourth, and o
13. To get area of pockets, in which line = 12 liters
line 12 by line 5, getting sq mm area. Line 14. T

dividing the former by 645. Line 15. Extract the square roots of line 14, and width of pockets. The numbers adopted, 7½, 8¾, 11, and 16¾, are tions, to aid carpenter work. The cubical form of pocket is satisfactory. refore, be 7.5 in cube, 8.75 in cube, 11 in cube, 16.75 in cube. These pockets desired rising currents. Line 16 gives approx heights or columns of water ig to force water out of the spigots. These values are dependent upon er on spigot orifice. Line 17 gives velocity in mm per sec of body falling um due to stated heights. Line 18. Velocity in mm per min is obtained by he 17 by 60. Line 19 gives sizes of spigots assumed. Line 20 gives areas spigot orifices, by computation. Line 21 gives liters of water and sand per gots, obtained by multiplying line 20 by line 18, and dividing by 1 000 000. liters of water per min, obtained by subtracting line 6 from line 21. Line ter per min from the spigots, obtained by multiplying line 22 by 0.26419. f overflow, it is line 7 that is multiplied. These final values of the gallons are undoubtedly too high, as no figures exist which allow for friction. The ay be ¾ or ⅔ of the figures arrived at above.

able 52. Velocities of Quartz Grains Falling in Water
omputing spigots, sorting columns, pockets, cones, and tanks (54)

ers., mm		Velocities, mm per sec			Diameters, mm (a)			Velocities, av, mm per sec
On reen	Average	Average	Maximum	Minimum	Maximum	Minimum	Average	
.01	11.93	393	489	271	0.509	0.243	0.369	39.0
.51	10.26	361	489	252	0.413	0.163	0.305	34.48
.79	8.65	340	489	231	0.339	0.114	0.234	28.57
.85	7.32	303	489	227	0.321	0.130	0.199	24.39
.97	6.41	289	416	185	0.298	0.103	0.182	20.41
.95	5.46	260	362	172	0.254	0.0827	0.156	17.24
.21	4.58	240	345	155	0.252	0.0655	0.135	14.49
.12	4.17	225	345	154	0.249	0.0803	0.126	12.05
.83	3.48	209	307	154	0.237	0.0679	0.121	10.20
.82	2.83	191	256	127	0.219	0.0655	0.116	8.55
.06	2.44	168	252	119	0.235	0.0569	0.112	7.14
.05	2.28	166	238	89	0.168	0.0608	0.0912	6.02
.63	1.85	146	227	71	0.158	0.0429	0.0846	5.05
.46	1.60	126	192	68	0.147	0.0414	0.0800	4.26
.27	1.37	118	173	82	0.120	0.0364	0.0747	3.57
.10	1.19	105	156	69	0.0972	0.0387	0.0689	3.00
.97	1.04	94	147	64	0.0970	0.0315	0.0629	2.52
.84	0.91	84	119	43	0.0918	0.0317	0.0555	2.12
.68	0.76	77	114	40	0.0842	0.0200	0.0503	1.78
.57	0.63	67	89	36	0.0718	0.0164	0.0478	1.50
.45	0.51	53	78	32	0.0715	0.0138	0.0425	1.26
.36	0.41	42	61	27	0.0660	0.0109	0.0377	1.06
					0.0629	0.0109	0.0344	0.887
					0.0611	0.0081	0.0319	0.746
					0.0539	0.0096	0.0282	0.627
					0.0543	0.0078	0.0267	0.526
					0.0462	0.0087	0.0253	0.442
					0.0462	0.0067	0.0232	0.372
					0.0394	0.0042	0.0209	0.313
					0.0408	0.0033	0.0188	0.262
					0.0323	.....	0.0182	0.220
					0.0301	.....	0.0161	0.185
					0.0412	.....	0.0144	0.156
					0.0403	.....	0.0126	0.131
					0.0318	.....	0.0088	0.110
					0.0264	.....	0.0059	0.092

Microscopically determined.

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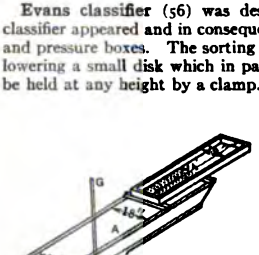
Table 53. Sizing Test of Feed and Products f
settling Classifier

Size, mm	Feed	Spigot 1	S
	%	%	
Above 0.907	0.41	0.0	
0.907–0.566	20.55	52.05	
0.566–0.427	14.84	24.62	
0.427–0.351	12.39	14.30	
0.351–0.277	9.87	5.22	
0.277–0.206	9.05	2.61	
0.206–0.137	10.03	1.00	
0.137–0.130	3.59	.....	
0.130–0.107	4.08	0.38	
Under 0.107	15.25	.....	..
Total.....	100.06	100.18	

Calumet (Richards-Coggin) classifier (55) is th
writer, and when introduced delivered feed free fr
have replaced it at Calumet, but it is still used in s
stood by the user it does good work. It may at one
time a hindered-settling classifier. It was designed b

Evans classifier (56) was designed by W. J.
classifier appeared and in consequence thereof. It h
and pressure boxes. The sorting column can have
lowering a small disk which in part closes it. This
be held at any height by a clamp. It is ingenious,



(a) Perspective (c) Sec K–L (b) Sec of Sorting Column

Fig 38. Gates Hydraulic Classifier

Richards annular-vortex hydraulic clas
coarser and for finer sands (58). It is sometim
as in the fine iron sand, 1/16 in to 0, from log
sometimes for separating coarser material (8 m
jig, from finer (2.5 to 0 mm) for Wilfley tables

One size has an annular sorting column, 13-in
about 10 in high. This will treat 5 tons per hr of
size, yielding 2.5 tons per hr of furnace concentra
tangential jets around bottom of sorting column.

Free vs hindered settling. In a FREE-SETTLING
to fall or rise in an upward current, without hind
shows an ideal classifier of this type, the operation
quartz and galena grains, below 2.75 mm diam, we
mm per sec, thus separating grains which rose wit
against it. The "spigot product" was dried, screen

naterial was then run with a 279-mm current, yielding a second spigot
h was also sized, weighed, and set aside. In like manner runs were suc-
le on each overflow, with currents of 200, 144, 103, 73.7, 52.8, 37.9, 27.2,
d 10 mm per sec, until the entire quantity of feed
rated into a series of spigot products and a final
f which had been screen-sized and weighed. . Such
means of computing the ratio of average diameters
average diameters of galena particles, when they
ling under free-settling conditions. Thus, for the
settled against the 73.7-mm current, the ratio was
71; that is, the av diam of the quartz grains is 3.71
the galena. Discarding the first 3 spigot products,
tion that the classification had not reached its pace;
products, as being beyond the range of good classi-
taking the average of the ratios from the middle 6
final average becomes 3.501. In HINDERED SET-
les of various sizes, shapes, and gravities, in a
, yet free to move amongst themselves, are sorted
rrent, the velocity of which is less than the free-
y of the particles, but sufficient to keep them in
Referring to Fig 40, a mixture of quartz and galena
s tube (7/8-in diam, narrowed to 1/4 in at lower end,
ere wash water was admitted). The rising current
s in "full teeter." They were in active motion, and
st one another, being able neither to settle into the
or to rise to the overflow. When equilibrium was
all grains had found their places, the current was
the lower grains fell into the bulb; then the cur-
n quickened, the bulb removed, by pinching the rubber tube, and another
ce. This was repeated until all the material except some extremely fine
wn out into 45 bulbs. These products were dried, screen-sized, and weighed.
Dividing the av diam of the quartz grains in any bulb by that of the galena, the ratio of diameters under hindered-settling conditions was obtained. Discarding, as before, the heavier and the lighter spigot products, and taking only the intermediate (bulbs 19 to 34 inclusive), the final av ratio under hindered-settling conditions was found to be 6.92; that is, the av diam of the quartz grains was 6.92 times that of the galena, or nearly twice as great as the ratio under free-settling conditions.

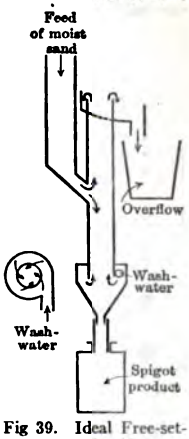


Fig 39. Ideal Free-settling Classifier

Thus, a FREE-SETTLING classifier has a launder or tank, a pocket for rough separation, a sorting column to finish separation, and a pressure box to deliver wash water up and spigot product down. A HINDERED-SETTLING classifier has all the above parts, together with a "teeter chamber" just above the sorting column or constriction. This chamber must in general hold some definite ratio of area to that of sorting column.

Hindered-settling classifier has the following advantages: Larger "added increment;" wider separation between the minerals when concentrated; more perfect separation of slime from spigot products; smaller water consumption than the free-settling classifier.

red-
(53)

42 explain the meaning of "added increment." In each diagram, at the
cal tube, in which grains of quartz and galena are supposed to have ar-
elves according to their settling ability; in Fig 41, under free-settling con-
g 42, under hindered-settling conditions. In each case the adjacent grains
spective ratios of diameter (see above). Therefore there is much more
the largest grain of quartz in the hindered-settling, than in the free-settling
41 a free-settling classifier is shown beside the vertical tube, and in Fig 42 a
ing classifier. In each case the galena that settles faster than the fastest qu

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appears in first spigot, in addition to the quartz and
(that is, the truly classified grains). This added ga
first spigot; and, because of the greater diam ratio of
more galena in first spigot of hindered-settling classifie

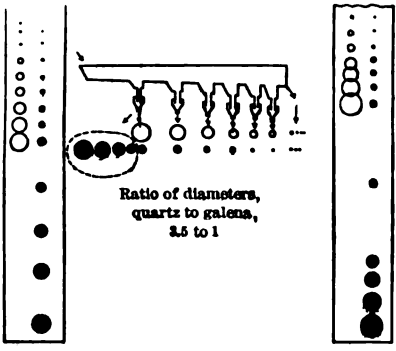
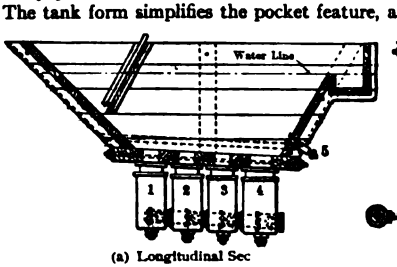


Fig 41. Diagram of Free Settling (53) Fig 42

Richards hindered-settling classifier (Fig
with water admitted by rings of jets, 4 jets fo
6 in the lower. The jets are alternately radia
tirely preventing the descent into spigot box
The tank form simplifies the pocket feature, a



(a) Longitudinal Sec

Fig 43. Richards Hindered-se

Spigot No 5, "safety spigot," is used in case
fine grains, too light to settle in No 4 and to
This largely-used classifier is made in differe
tons per 24 hr. Cost depends on number and

Table 54. Dimensions of Richards Pulsator

Dimensions, in inches (approx onl

Size	A	B	C	D	E	F	G
3-in. 6-compartment..	43	51	27	78	26	13	5
4-in. 6- "	48	67	31	85	34	15	6

Both sizes have 12 by 2¾-in pulley (160 rev per m

inverted pulsator classifier (Fig 44) is a hindered-settling classi- "inverted" because it lifts the light stuff out first, and the products weight to the last (60). It has a screen below, which serves as con- A heavy bed of shot is placed on screen, and little dams prevent the moving forward. The wash water is admitted through a revolving produces an intermittent current. This classifier gives a graded set

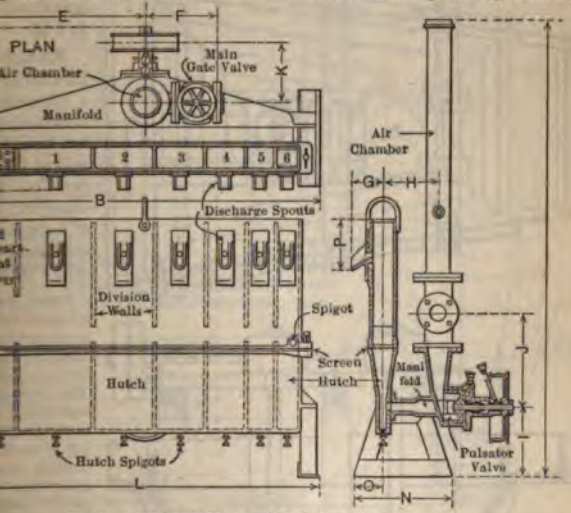


Fig 44. Richards Inverted Pulsator Classifier

ranging from No 1, the finest, to No 7, the coarsest. Spigot No 6 is or down, to hasten or retard the forward movement of heavier part. afety spigot," opened only when ore contains a product so heavy that to lift it to overflow, No 6. It is doing good work. In one mill it e an amalgamator, also, and in one year saved on its screen $30 060 that the traps did not catch. Capacity of the 3-in classifier is 90 or less, according to size of maximum grains; that of the 4-in is ons per day. Cost of the 3-in is $325; of the 4-in, $430.

Fig 45. Richards Launder Pulsator Classifier

pulsator classifier, launder type (Fig 45), is a hindered-settling th a separate revolving cock and driving pulley for each pocket and ber (60). Pockets may be placed at such distance apart as mill

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design requires; any number may be used,
Capacity of classifier depends on size and num
Capacities of 25 to 300 tons, or more, per
pockets ranges from $80, for 2.5-in, to $116, for

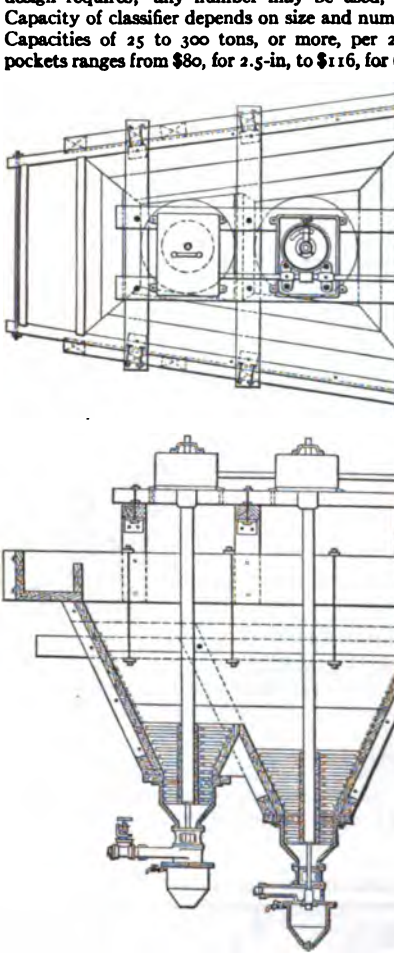

Fig 46. Richards-Janney Classifier (59)

the hollow shaft; to this rod is attached a valve
When the valve is opened, the sorted product falls in
discharge through the valve is intermittent, while t

Capacity of a 5-spigot classifier varies from
hr, on material up to 7 mm. Water consumpti
of ore, on material below 6 mesh. Table 55 giv
Magna plant of Utah Copper Co.

Sizing Test on Products of Richards-Janney Classifier, at Magna Plant of Utah Copper Co (59)

	Feed	Spigot 1	Spigot 2	Spigot 3	Spigot 4	Overflow
	%	%	%	%	%	%
10	6.9	12.5	.....	.....	.....	.....
20	32.1	62.8	46.3	.....	.....	.....
30	11.9	16.4	35.5	.....	.....	.....
40	6.0	4.0	7.4	19.9	.....	.....
50	5.7	1.4	4.1	41.3	9.1	.....
60	1.7	0.3	1.3	10.5	6.6	.....
80	3.8	0.6	1.5	10.7	20.5	.....
00	3.0	0.4	1.1	5.6	25.5	.....
20	0.6	0.0	0.0	0.4	7.7	.....
50	7.5	0.3	0.5	3.3	17.4	5.4
00	0.9	0.1	0.5	0.8	2.8	12.0
00	19.9	1.2	1.8	7.5	10.4	83.6
	100.0	100.0	100.0	100.0	100.0	100.0

(Wheeler-Wiggin-Richards) (61) **classifier** direct outcome of the first hindered-settling ig 40), having the cone, teeter-chamber, con-

Sizes	Dimensions A	B
No 1	7'-0"	4'-3"
" 2	2-4	3-6
" 3	Same as No 1	
" 4	2-1	3-6
" 5	1-11	2-6
" 6	5-0	3-6

current, pressure box, and spigot (61). It oted by Wheeler and Wiggin, of the Ana- Mining Co, for removing slime from sand, ncentration, and for grading very fine sand l has superseded other classifiers at that

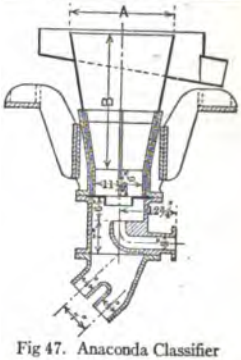

Fig 47. Anaconda Classifier

and-slime separator (Fig 48) is a pulsating, cone classifier, for separat- slime. It can divide on any point decided upon; that is, to settle against a 10, or 5, or 1 mm per sec current, the slower grains in each case going to overflow. Having a classifier capable of handling the quantity of feed water, it is then adjusted for the size desired. It must have a pulsating current, otherwise there will be a down current in one place and an up current in another. It has a screen plate on which is a bed of rounded pebbles, about 1 in diam. Thus far it has been proved only in the laboratory, doing good work there, and showing that by this principle, fine lines can be drawn between size of grain in the sand and in the slime.

ards Sand- rator (53)

Deister classifier (Fig 49) consists of a vertical series of conical, slotted diaphragms, enclosed in a slightly tapering barrel, through which rises a column of hydraulic water. This enters through radial holes in the ttom, and meets the descending grains of ore. The apparatus may ler the apex of a pointed box or conical settling tank, or under an ttom of a launder. Its principal merit is that the peculiar shape ragms, and the staggered position of the slots, afford such amp

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opportunity for light grains to be caught up and
ate volume of hydraulic water needs to be added

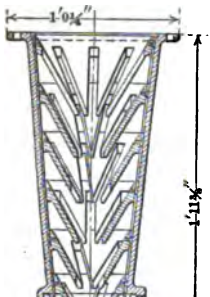

Fig 49. Deister Classifier

Difficulties of classifi
greatest difficulty, in cla
secure a uniform rising
dies to carry slime into t
of doing this: (*a*) by pro
helical current, or by usin
(*b*) by placing a teeter ch
a pulsating current.

(*a*) Vortex and revolvi
ward eddies, by distributi
any horizontal section, a
effect of friction against

(*b*) Teeter chamber and
a probationary period, d
chance of removal than if

(*c*) Pulsating current i
a rapid up current of sam
of sorting column. The
producing an even rising
current be doubled, the
quadrupled. This, often
downward eddies.

A satisfactory hydraulic classifier should be able so
Wilfley tables, that slime is completely eliminated from
remaining in their tailings.

18. Pulp Classifiers without H

These include: pointed boxes (spitzkasten),
sliming and dewatering cones, dewatering boxes
tanks. In a concentrating mill, after water has
mels, fine crushing machines, and hydraulic c
reached where it is unwise to add more water, bu
proximate classification according to size of the

Surface-current box classifier (Rittinger's
work (62).

Rittinger recommended: (*a*) For pulp of 1 mm an
per min: 1st box, 0.632 m wide by 1.897 m long; 2
long; 3d box, 2.529 m wide by 3.793 m long; 4th box, 5.058 m wide by 4.742 m long. (*b*) Ends of boxes should slope 50°, to prevent banking (sides to be steeper). (*c*) Goose-neck spigot, discharging 3 to 3.5 ft below water level for the first, and 2 to 2.5 ft below for last spigot. (*d*) To avoid banks, the launder feeding 1st box should have a slope of 1 to 1.5 in per 6 ft (Locke (45) found 5° slope for this); 2d box, 0.5 to 0.75 in p 6 ft; 4th box, 0.125 to 0.25 in per 6 ft. (*e*) Boxes 1 and

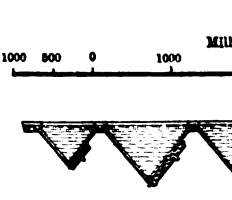

Fig 50. Ritti

4 to be hopper throughout, the spigots to be 0.25 in to 0.75 in diam, accordand. Delivery by spigots may be: 40% in 1st, 28% in 2d, 18% in 3d, 10% in overflow.

n spitzkasten. If feed pulp is flowed over the surface of clean water ak, the heavier grains will settle to the spigot and be delivered through it. e current mixes with the water beneath, and, of the increased volume of l, only a small part can overflow. The remainder passes down, and consand passing out of the spigot. Hence, though spitzkasten products are aximum size of grain, from coarse maximum grains in the first to fine s in the last box, the particles in each spigot product range from its maxfinest slime. The operation is therefore much less perfect than that of ifiers. Operation of pointed-box classifiers can be improved by addiplate parallel to discharge side of box, and rising above water level, thus lined rectangular sorting column. Additional, submerged baffle-plates this space occupied by the rising current, not only to assist further in he effect of eddy currents, but to increase settling capacity, since grains e low-velocity stratum adjacent to the upper side of every baffle plate d more rapidly to bottom of box.

ifiers (sand-slime separators). Three designs are in common use, s oscillating rakes which pull the sand from receiving tank up an scharge it at the top above water level, while the slime overflows of receiving tank. ARGALL classifier has 2 screw conveyors for sand up the incline and delivering it above water level, the slime s in the Dorr. In the ESPERANZA-FEDERAL, 2 parallel sprocket crapers draw the material upward. All these effect good sepae from sand, and deliver the sand with very little water. The er used is a little spray near top of incline, to take out as much of slime as possible.

e de-slimer and de-waterer (Fig 51). An ordinary settling box ottom may be about 5 ft wide by 12 ft long, with sides sloping orizontal. The 8-ft ering cone, with 60° m overflow 25 ft in . Thus, the width the cone is 5 times inted box, and, for of feed, the depth f overflow is ⅛ as he maximum grain ill be about ⅛ as the pointed box. the cone to settle fore much greater. the pointed box is with a plunging follows rapidly m, and stirs up the settled; a disadvannt in the cone (58).

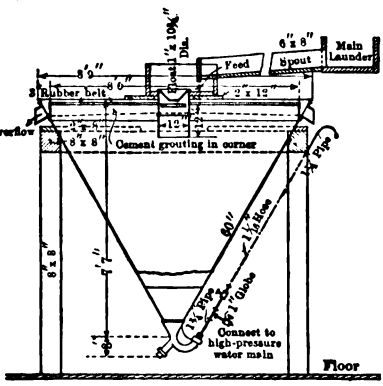

Fig 51. Callow Settling Cone (58)

f Callow cone. The cone is a whole-current classifier, in which the y constant throughout, though unavoidably somewhat stronger at cirn near center. Feed enters at center through a vertical pipe, which has subdue force of current; it then spreads and rises, all particles that can ivered to the spigot, the remainder going to the overflow. It follows ished quantity of feed, or with a larger cone, the size of maximum g-

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in the overflow will be smaller. The apparatus With a smaller cone, or a larger quantity of pulp DE-SLIMING CONE. Hence, by deciding on the size drawn, between clear water and slime, the size of give clear water are determined. In other words, can be allowed to go into the clear water and still

Callow de-watering cone. A pulp feed of water clear enough to discard or to use again of pulp would differ according to the value o ample, in treating galena (with lead at 4¢ per lowed in the clear water, than if treating tellu per oz). Again, settling is hindered by extre mill in winter, clear water can never be obtain be helped by saline solutions. In Arizona, wi even when this has been precipitated by addi in the water helps rapid settling.

Table 56. Velocities of Rising Cu

Diam of cones, ft.....	2	3	4
Diam of feed pipe, in..	6	8	10
Net area, rising current, sq in.....	424	967.6	1 731
Net area, rising current, sq mm.....	273 619	624 281	1 116 829
Gal per min in feed			Rising c
10	2.31	1.01	0.56
15	3.46	1.51	0.84
20	4.61	2.02	1.12
25	5.76	2.52	1.40
30	6.92	3.03	1.68
35	8.07	3.53	1.96
40	9.22	4.04	2.24
45	10.37	4.54	2.52
50	11.53	5.05	2.80
60	13.83	6.06	3.36
70	16.14	7.07	3.92
80	18.44	8.08	4.48
90	20.75	9.09	5.04
100	23.05	10.10	5.60

Table 56 has been prepared to aid in deciding pulp needed in a given case. Thus, for an 8-ft d feed, the velocity rising to overflow is 0.34 mm per se (Table 52) 0.022 mm maximum diameter. Table 5 is obtainable from a 7-ft cone and 20 gal per min, 5-ft cone and 10 gal per min feed. If the 8-ft con sired to lift 0.1 mm grain to overflow, the rising (from the table), the same work can be done with a 2-ft cone and 30 gal per min feed.

Caldecott cone (109), developed in connection a flat diaphragm supported by 4 lugs riveted to space between edge of diaphragm and wall of cone about 1 in. The cone is allowed to fill with sand t is adjusted to maintain sand at this level. Owing spigot opening can be large. Function of diaphra of sand and thus prevent outrushes or clogging at

ring boxes are used wherever the quantity of water accompanying too large to feed to the next concentrating machine. They are gular boxes, but hopper-shaped boxes are better if space permits. In the overflow is clean. If it carries too much value to discard, this urned to the mill circulation, at a point where the contained sand arm and the values may be saved.

rent box classifier, or settling tank, is one in which all the water moves ity from feed end to overflow. It requires a very systematic distributer at f the water plunges into the tank it may give a rapid bottom current, which ow and carries with it larger grains than it should. Sometimes there is a face current toward feed end. If properly adjusted, spigot products taken bottom will show same grading as a spitzkasten. Some prefer to use this ributer, with the last spigot product as nearly like the first as possible, in e plunging feed, quick bottom current, and absence of careful distribution s.

ontents of tank in gal C, and number gal fed per min F, then $C + F = T$, min pulp will take to pass through tank. Again, knowing depth D (in or 2 shows the diam of quartz grain which will settle D mm in T min, thus um size of grain in overflow. In discharging from bottom, the spigots di- locity and quantity of water overflowing, and also size of maximum grain. **tanks or reservoirs** are for settling sediment, so that the water can be re- circulation. Where reservoirs are used, they should be in duplicate, one receiv- r from mill and returning clear water, while the other is being cleaned out for a

Where Cal- ng cones are st be a suffi- so that none red to take safe amount ter is scarce . cones may large, and ntermittent s used. This Bote turned p with 50% a plug which ically for an 2 or 3 min. at Nacozari, cylindrical center and cted in bot- Tanks are dically, and ses too high through a valve. The tions. When ng, top plug arging, both ; when pulp w, bottom nd hydrant n a moment 4).

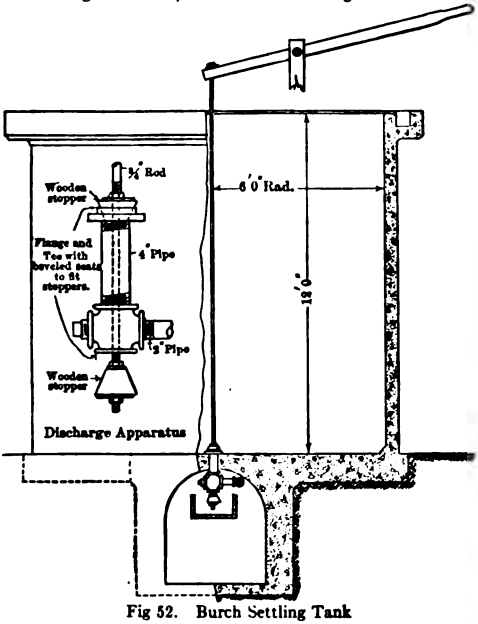

Fig 52. Burch Settling Tank

xener (108) is a circular tank 27.5 ft diam and 9.7 ft deep with and peripheral overflow; discharge is through one spigot at center A central vertical shaft, making 3.3 rev per hr, carries 4 arms

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near bottom, sloping upward at 12° from horiz
clined blades which cause the settled pulp to t
This has advantages of large capacity with sm
discharge spigot. At Anaconda, this thicken
solids in feed, at rate of 17.1 tons per 24 hr.

WET CONCENTI

19. Field of Concentrati

Principles. Modern mill work divides itself
treated by the different kinds of machines. The
or natural products. In SIZED PRODUCTS the
sieve through which they pass and the finer sie
the holes of any sieve are not more than 1.5 ti
finer sieve. CLASSIFIED PRODUCTS are of 3 kir
all the coarse, and some of the finer concentr
the later spigot products have finer concentrat
classifier or tank overflow carries the finest g
mud. NATURAL PRODUCTS are materials of r
have simply passed through a screen, defining

Kinds of concentrators. JIGS treat mate
mm, and are advantageous for sized or classif
(and allied types) treat material from 2.5 mm to
granular; they can be fed with sized or classifi
natural, products. VANNERS treat material fr
granular sand; their feed should be classified,
natural product. ROUND SLIME TABLES treat
to and including colloidal material; are best
overflow. The removal of colloidal matter f
vanners greatly lessens loss of fine free mineral

Among other machines, less commonly employed
the following: GILPIN COUNTY BUMP TABLE treats
including colloidal mud. Feed must not be class
spigots will not have sufficient size and weight of he
forward. BUILDING BUDDLES can treat material fr
for best work, they should be fed with the later spigo
and CANVAS TABLES, for best results, should be fed
down; their best work is on later spigot product
BLANKET TABLES, RIFFLES, and KIEVES have special

20. Jigs, Types and Co

The work of jigs is based on action of alterna
rents of water, through a bed of sand resting on
or PULSION, causes the mixed grains to strati
settle in water under the conditions; the heavie
downward current, or SUCTION, causes the sm
pass down through the interstices between the
screen. Pulsion is the primary action, suction
of suction is advantageous for jigging mixed
disadvantageous for closely sized products. Ji
the coarser products, from 50 mm (2 in) to
products go to Wilfley tables and vanners.

-sieve **hand jig** (65) is used in preliminary testing work, or for small-scale ot warranting expensive plant. Fig 53 shows one form, consisting of a tank, spended a sieve frame. Screen is usually of cast iron, with slots about $\frac{1}{16}$ in ieve is moved up and down by hand by lever *D*, rod *O*, lever *E* and 2 rods is loosely connected to *E*, enabling operator to give an upward blow on *E*, n imparts a sharp down stroke to the sieve, causing it suddenly to leave the orted, which then falls through the water and effects the separation. Ma- arse as 25 mm (1 in), and as fine as 6.3 mm (0.25 in) is treated, but

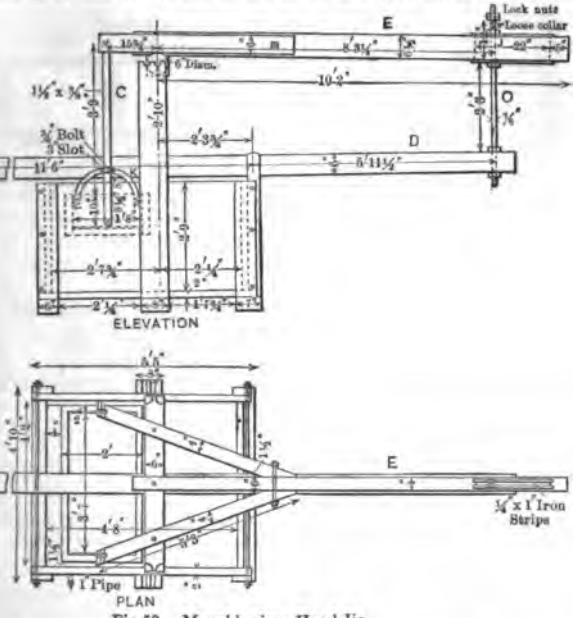

Fig 53. Movable-sieve Hand Jig

n stuff 12 mm (0.5 in) and below. It is fed by shovel to a depth of about level should be slightly above top of screen frame, insuring freer jigging. ite number of strokes (100 or more), the layers are skimmed with a short- The top layer may be tailing; the next, middling for recrushing; next, rich e re-fed with next lot; next (as at Joplin, Mo) may be clean sphalerite con- he smelter, and last layer on sieve may be galena. In Missouri, the last ide, and when a sufficient quantity is collected, is treated on a secondary jig, ean galena, the sphalerite top layer going back to primary jig. The hutch be re-treated on a similar jig with finer screen, but, as it is difficult to get result by hand jigging, the hutch product is sold to a sludge mill, which y power jig and tables. Each jig requires one man. Hand jig costs about constructed, and needs few repairs. It has the advantage over power jigs of tle water, but the disadvantage of small capacity. Rittinger gives capacity as 3 or 4 cu ft of ore per hr, for each sq ft of screen surface.

k vanning jig (65) has a tank or hutch 25 ft long, 4 ft 2 in wide, n deep (inside measurements).

le screen 2, submerged in water, is 20 ft long by 2 ft 8 in wide, divided of pockets by transverse strips, to retain the screen bedding in place. The

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screen is coarse enough to allow the largest particl
the hutch, the bedding being of concentrate of a la
The screen is suspended from cross arms 3, which a
these in turn by rocking arms 5. The latter are gi
and three-armed cam 7 on main driving shaft 8. Sp
giving 180 to 195 reciprocations. The up and do
guide 9, gives the upward-forward and downward

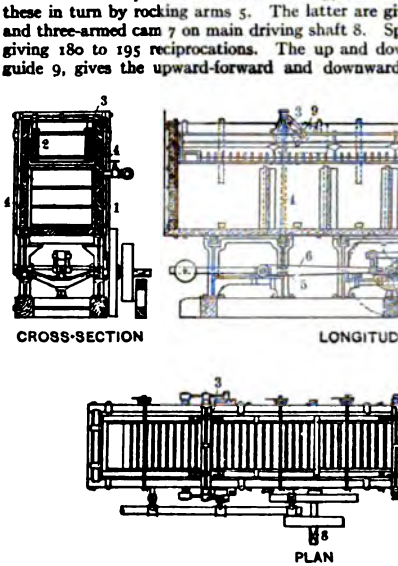

Fig 54. Hancock Vann

the forward movement of the sand toward the tail d
the capacity. Capacity often reaches 500 tons per 2
per ton of ore treated.

Power required, 4 hp. The jig gives 3 hutc
one hutch for recrushing, and one to be re-fed
make very clean tailings. Cost, $3 000 (66).
copper ores, size 14.3 to 5 mm, capacity was
sumption, 507 100 gal per 24 hr; on 8 to 2.5 m
using 1 327 gal water per ton. Alexander Veit
one case replaced 8 Harz jigs, using fewer ji
power, ¼ the water, and involving less wear
centrate was richer and the tailing poorer.

Harz jig. There may be from 1 to 7 com
is roughly U-shaped, the screen being fixed in
the other (Fig 55). The plunger is operated b
form speed. Each compartment makes 3 prod
side discharge, fine concentrate through screen
flow. Each compartment rejigs the tailing
successfully for coarse and fine ores and for
making 2, 3, or 4-mineral separations. In g
are used as there are minerals to be separated
partments are discussed later).

57. **Details of Harz Jigs, of Cananea Consol Copper Co** (67)

	1-compart jig	2-compart jig	3-compart jig
reated.....	Product from 1.0 –0.625-in screens	Product from 4.8 –2.0-mm screens	Spigots 1 and 2 on 2.5-mm feed
.....	24×40 in	24×36 in	24×36 in
.....	21×40 in	21×35 in	21×35 in
ngth.....	2.25 in	1 in	0.5625 in
r min.....	115	170	200
bed.....	8 in	4 in	average in 1 comp, 6 mesh No 16 wire
th.....	4-mesh, No 12 wire	5-mesh, No 14 wire	2 and 3 comp, 8 mesh No 18 wire
feed.....	2 10	1.00	1.10
"	2.05	.....	.....
24 hr.....	168	42	77.5
r, gal per min	96	.....	.....
ns per 24 hr	6	3.3	.....
" " "	12	8.0	25.0
" " "	150	30.7	52.5
een, days....	80	50	25–30

ows a com-
in which 2
ack to back
e diagram:
gging com-
B; plunger
t; *C*, hop-
h; *H*, dis-
for coarse
ate; *D*, dis-
ning from
iding cover
g hutch dis-
creen frame;
X, eccen-
water is dis-
n a trough
to top of
Coarse con-
charges un-
shield *I* on
through

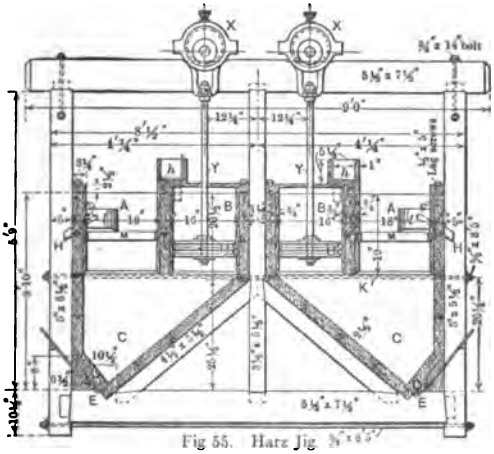

Fig 55. Harz Jig.

Table 58. Prices of Harz Jigs (68)

	Wt iron work only, lb	Price, iron work only	Wt jig complete, lb	Price, complete
		Single jigs		
	500	$75	1 500	$150
	700	125	2 300	225
	1 100	175	3 000	300
	1 500	225	3 800	375
		Double jigs (2 jigs back to back)		
	1 000	$150	2 800	$250
	1 400	250	4 300	400
	2 200	350	5 400	550
	3 000	450	6 800	700

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Hodge jig (69) is made in pairs, back to b
each side; 2 pairs placed one below the other
Each pair costs $125. Frame and tank are of
is of the sliding-block type, giving rapid down
ment. Largely used in the Lake Superior copp

At a mill of the Copper Range Consol Co, 24 2-c
material below 0.25 in. On each jig an average of 2
with the pulp, 20 740 gal of wash water per 24 hr b
17 560 gal to tail screen. Screens are 24 by 36 in, plu
per min. Length of stroke varies on different jigs, f
"skimmings," hutch, and tailings.

New Century jig (70) has a slow suction
by a cam on driving shaft, and a swift pulsio
spring encircling plunger stem; compression of
sion stroke, can be
compartments in l

Fig 56. Richards Pulsator Jig (60)

Richards pulsa
A quick pulsion is
which opens and c
times per min, by
suction is caused
flow between puls
water due to flow
tion, though small
work. The produ
and tailing. This
as an equivalent
and the passage th
rapid that it prod
in plunger jigs;
avoided. Blindi
place while runnin
to be about the sa
is inclined to thi
water to do gooc
jig (60). Approxi

Table 59. Details of Richards

Size, in	No of compts	Selling price	Extra for mule-back transport	Bo o shi
6	2	$325	$10	
6	3	360	10	
8	2	395	40	
8	3	495	60	
12	1	250	10	
12	2	470	50	
12	3	560	75	

Richards pulsator laboratory jig weigh

Woodbury jig is one of the so-called classi
the first compartment by a double shield and da

e outer shield, sand for further jigging is removed in the space be-**er and inner shield**, the portion coming from inner shield being con-. **The chief object** is to avoid diluting the slime, and to get rid of it **osequent jigging.** The jig has from 3 to 5 compartments, the first the slime and yielding coarse concentrate, hutch, and tailing. The **k of the last** 3 or 4 compartments goes to finishing jig for re-jigging.

met & Hecla mills, on conglomerate copper rock, two 5-compartment jigs **uct of one steam stamp**; 325 tons of sand below $\frac{3}{16}$ in, containing 45% of **24 hr. On Lake Superior amygdaloid rock**, three 5-compartment jigs treat **one steam stamp**, or 600 tons of sand below ¼ in, **with 40%** slime, per 24 hr. **er is driven by the sliding-block slow return on some of its compartments**, and **entric on others.**

charges. (*a*) **By skimming to remove concentrate.** To do this the **s are skimmed off first and afterwards returned.** (*b*) Shield and dam (Fig 57). Shield *E* reaches from above water level the screen. Only the bottom layer of concentrate can **ugh opening** *F* to the space behind the shield; then an **dam permits** concentrate to discharge (65). Height can be adjusted to prevent tailing from entering con-and height of dam to prevent concentrate passing out ng. **Height** of shield, and of dam above shield, are by trial. A simple computation will determine this **lly**, but the result is never attained because of vari-**rbing** forces. (*c*) Pipe and pan discharge (Fig 58) **blique pipe** *p*, passing from screen through side of **A pan**, hinged beneath it, can be raised or depressed **nb screw** on a bolt. This is not so automatic as *b*, but has succeeded **laces** (62). (*d*) Harris air-vent discharge is especially good for drawing from whole width of the tail of a screen. (*e*) Searing rocking pan and **harge** is an improvement on pipe and pan, where eddy currents around the pipe may harm the jigging. It can discharge large quantities, where the concentrate exceeds 50% of the feed. No other discharge is reliable in such case. (*f*) Shields' scoop discharge (71) lifts all concentrate above a certain adjustable level. As the concentrate can never be drawn below that level, and as very little water is discharged with it, this is probably the most per-tomatic discharge known. Monteponi discharge (Fig 59) is a pipe, ½ to **n**, according to coarseness of ore, set at a slight slope, and perpendicular **n** of jig compartment. Discharge is usually intermittent for fine ore.

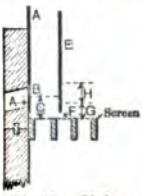
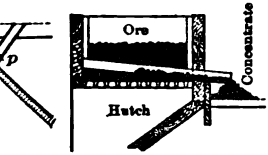


Fig 57. Siphon Discharge



Pan
arge
Fig 59. Monteponi Jig Discharge (110)

reens are best of moderately stiff wire cloth. Brass or copper lasts 60 to **rs, and** is better than iron, which rusts rapidly, lasting only 20 to 300 days. **supported by a grid** of cross bars, sides and ends of frame resting upon **ning** of screen compartment. Similar lining above holds screen in place **nd frame should be so designed** that there are no projections to cause in-**ddies.** When the jig frame is all of wood, the screen is tacked to it, but **cross bars** must not be so thick that harmful eddies are formed. In Joplin **n grating is used.** Thin metal cross-bar supports are best, with bent wires **e screen to them. For heavy jigging, punched plate is used.** Iron-wire **sts 7 to 60¢ per sq ft**, according to size; brass or copper, 35¢ to $3.

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Number of compartments corresponds in g
als to be separated with side discharges. For e
2 compartments, the first giving galena, the
sphalerite and quartz, 3 compartments, the first
ite, third middling, and so on. But, when large
the concentrate is a high percentage of the feed
needed. Thus, the Joplin jig, in separating gal
6 or 7 compartments, for securing large capac
have 4 compartments, the concentrate being a ve

Accelerated plunger motions have alread
New Century, and Pulsator jigs. They concent
serious "matting" of the screen bed. The best
ing block and crank arm. The SLIDING BLOCK
by pulley *b*. Mounted on *a* is crank pin *c*, in

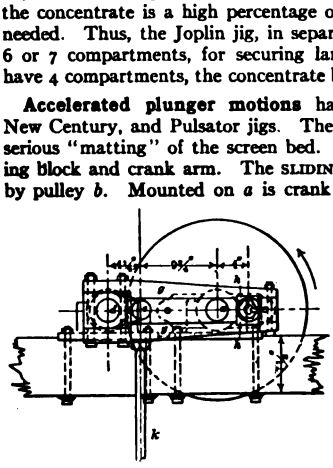

Fig 60 and 61. Jig Plunger Accelera

As *c* revolves, the arm *g*, keyed on shaft *d*, os
Also keyed on *d* is a short arm *de*, to which is pi
of this device is to impart to the plunger a slo
farthest from *d*) and a rapid down stroke (whe
ARM AND ECCENTRIC (Fig 61) does the same thi
to give somewhat better results (62).

21. Jigs, Details of O

Feed to jigs may be: (*a*) sized product, limit
which it passed and the following sieve on which i
from first spigot of a classifier, enriched by th
Fig 41 and 42); (*c*) classified product, from a lat
in small grains and light mineral in large grains;
the entire undersize of a screen, containing grai

Feed (*a*) can be jigged on a screen of moder
little suction as possible (obtained by excess o
pulsator jig); suction mats the jig bed and g
feed yields coarse discharge concentrate, very li
imperfection of screening), and tailing. The ta
quire regrinding, according to value of the met
in the gangue. Feed (*b*) may make its own be
holes should be a little smaller than the coarse
coarser concentrate may be put on the screen,
be larger than the largest feed grain. The
needed is 3 times the size of largest grain to p

ximately ¼ the size of bedding grains. The plan of using an added he advantage of not blinding the screen, and therefore makes for freer It is not uncommon to ballast the jig bed by a layer of iron balls in diam. This equalizes the currents and improves layering of the If classification has been good there should be no fine free mineral in g. Feed (*c*) has larger gangue particles and smaller grains of heavy The screen mesh should be large enough to pass the largest grain of s, and bedding should be of larger size, so that the interstices will allow s to pass through. In this case, only hutch and tailing are made. cation is good the tailing should contain no fine free mineral. Feed dled in same way as feed (*b*), and will yield same products, except that g always contains fine free mineral.

r required by jigs depends on sieve area, height of tail board, specific f feed, and length and number of strokes per min. One manufacturer s 1.5 hp for a single screen, 2 hp for a 2-sieve jig, 2.5 hp for a 3-sieve 3 hp for a 4-sieve jig. To this add 15% for friction of shafting, slip of d other losses.

60. Capacity, Water, and Power for Harz Jigs (from Practice) (72)

Size of feed	Capacity per 24 hr	Net area, each sieve	Capacity per sq ft per 24 hr	Added water per 24 hr	Water per ton of ore	Hp required per jig
mm	tons	sq ft	tons	gal	tons	
12.7–0	(*a*) 100–120	6.000	8.00–9.60	(*a*) 400 000	15.29	5.0
4.60–3.48	12	2.597	1.54	40 000	13.90	2.0
1.22–0	8	2.597	0.77	30 000	15.64	1.5
5.7–3.6	15	3.778	1.32	75 000–100 000	24.32	0.5
30–20	39.6	3.880	3.40	68 383	7.20	1.5
2–1.5	19	3.500	1.81	43 200–54 000	10.66	1.0

(*a*) These are for 10 hr instead of 24 hr.

ht of jig tail-board. A common height for medium sizes is 4 in; for es, 3 in; if the heavy mineral has high specific gravity, even 2.5 in. y coarse material, it is probable that 6 in, 8 in, or even 10 in may be For example, for 2-in lump (50.8 mm) with shield and dam discharge, ould be 3 in above the screen to let lumps through. Hence there should 5 in of gangue over the 3 in of concentrate to protect it from loss. This equire a dam 8 or 10 in high. These figures are not fully proved, the Cananea bull (roughing) jig has 8-in tail dam, and elsewhere coarse been jigged with a bed 12 to 18 in deep.

th and number of strokes. Stroke ranges from 2 in or more for very naterial, down to ¼ or ⅛ in for fine products. The chief point is to lift d sufficiently from the sieve to allow the particles to rearrange them- the water. The strokes per minute vary from about 100 to 250, the number with the longer stroke, and vice versa.

tity of water is very variable, according to whether the jig is run on minimum d maximum feed. Dudley (93) proved that a free-flowing spigot requires as 3 lb water to 1 lb sand. If sand has a specific gravity of 3, there would be 9 water to 1 volume sand. If this is also true of the jigging bed, 3 tons water to nd are required on each screen, and 3 tons water to 1 ton sand in each spigot. f the spigot products are ⅓ of the feed in each compartment, a 1-compartment ting 100 tons sand, would use 330 tons water; a 2-compartment jig, 627 tons; rtment, 894 tons; 4-compartment, 1 135 tons of water. No satisfactory data ubject are at hand.

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Laws of jigging. Ultimate effect of pulsion is to p
screen. These decrease in size upwards to the point
is so small that the large quartz grain will settle with
grains decrease in diameter, as well as those of heavy m
is coarser than the adjacent heavy mineral. Suctio
heavy mineral in the interstices between the quartz g
screen into the hutch. If jig bed be deep, and the
done, the making of hutch-product would be pract
coarse bedding with large interstices, and a coarse scre
The successful hutch-making jig must be worked ou
of bedding, and size of screen holes that produce be
peded by "matting" of the whole jig bed, due to s
plunger stroke can be only imperfectly softened up by
desirability of avoiding the symmetrical plunger stroke
eccentric. In the Joplin jigs, matting is overcome b
feed (3/4 to 1/2 in), and by excessive amount of water
copper jigging at Lake Superior, in Montana, and in l
by accelerated pulsion and retarded suction motion of
and 59). It is overcome in the pulsator jig by the al
and the soft and free working jig bed. This gives the

22. Wilfley Tabl

Features. It is a riffle table (Fig 62) abou
driving end and 5 ft wide at concentrate end
parts an oscillating, lengthwise motion, with a de
ing objects on table to move towards opposite e

On feed side near the head, the pulp is fed with wa
clear water is fed all along the remainder of **feed side.**
of about 5/8 in to 1 ft; or, on many ores, it will be 0.7
opposite or tailing side. Lengthwise, table is nearly l
0.5 in in 15 ft toward the concentrate end; this also i

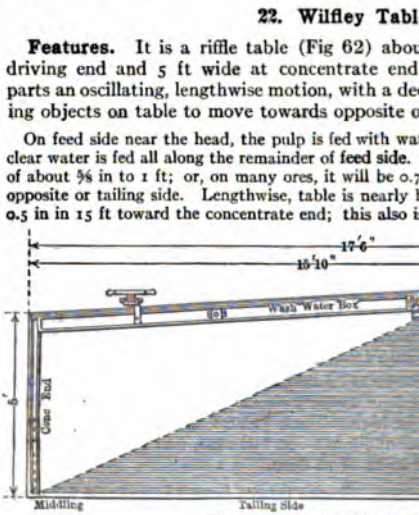

Fig 62. Deck of Wilfley T

covered with linoleum, which is just rough enough to aid
it prevents warping of the deck; it takes the wear a
On the linoleum are tacked riffle cleats, laid parallel to
the oscillating movement; they are 0.25 in wide and a
The lowermost riffle cleat runs whole length of tail
tapered from about 0.5 in high to 1/16 in or less at the
cleat is about 5 ft long, starting from head end with s
termediate cleats, 44 in all, have the same taper, and

Operation. (*a*) Feed pulp and water start down
the riffles; (*c*) pulp is stratified by the oscillations, h
gangue to top; (*d*) the loose stratified bed of pulp i

oncentrate end; (*e*) top gangue is sheared off by the downward flowing rtly to thickening of the bed in its forward progress, and partly to diminish the riffles; (*f*) when the bed reaches tips of cleats on the diagonal line, still remains. But, on crossing the diagonal, nearly all remaining gangue and, as the concentrate advances onto the unriffled portion of deck, the e particles are repeatedly rewashed as they pass from tip to tip of riffles, ng into the concentrate, some into the tailing. Thus the diagonal line of he line of control: concentrate is retained, gangue passes off. From this eries of bands results, spreading like a fan; upper band contains the heaviest thers, minerals of less and less weight. Just below the diagonal a band of cluded grains is generally found, and below that the band of tailing, more om included grains. A raff-wheel elevator (Fig 62) returns middling to if desired.

on (Fig 63) consists of an eccentric and shaft *S*, a pitman *P*, and toggles ggle *A* acts between *P* and yoke *D*, toggle *B* between *P* and the abutment ted to foundation. End ached by draw-bolt *H* to Spring *E* holds toggles eats. Owing to the relative of toggles and eccentric, ses its direction of movement ximum velocity at one end at the other. This quick the progression of the of stroke can be changed *G*. Head motion has life s of continuous service.

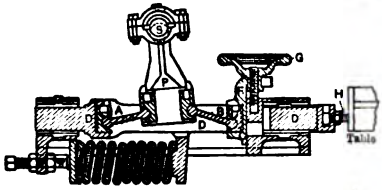

Fig 63. Head Motion of Wilfley Table (73)

ne (foundation) of Wilfley table No 6 consists of two 12-in steel channels, d 20 in apart, bolted to cast-iron cradles. This construction insures sta- erection, maintenance of alinement, and is readily adapted to concrete pier

narrow strips of redwood, laid diagonally (to avoid warping) upon a light ced with wood-filled pressed-steel stringers, and 2 steel cross trusses. The ring and riffles are described above.

vice, including support to deck, consists of cross channels, carrying slipper ed to cast-iron bolster yokes, having segmental machined surfaces and ular machined seats in main frame cradles. The yokes have a projecting ronze nut for engaging the tilting rods. One hand wheel, with miter gear perates both tilting rods with their right- and left-hand screws engaging the n yokes.

ulp as coarse as 4 mm and as fine as 0.06 mm (200 mesh) has been y table, but in regular mill practice it ranges between 1.5 and 0.5 mm. ay be: (*a*) natural product (mixed sizes); (*b*) first spigot product er; (*c*) classifier later spigot; (*d*) closely-sized product.

hould not be larger than 1 mm, or 1.5 mm, at most. This simplifies table, but has the disadvantage of loss of fine free mineral in tailing, rich material, causes serious loss. It also yields considerable ntaining coarse values, which requires a special, one-spigot classifier When (*b*) is fed, the size may be up to 2.5 mm, but much ll be formed, to be cleaned up as in (*a*). Tailing should contain mineral. For (*c*) the feed may readily be up to 2.5 mm; there ine free mineral, and so little middling that it may be returned le (by the wheel elevator), unless there is a middle-weight mineral en it must be treated separately. (*d*) Closely sized feed probably st tailing, as regards fine free mineral; and it gives but little mid- can be returned unless there is a middle-weight mineral, when it ated separately. This kind of feed may be up to 2 or even 2.5 mm;

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the objection to it is the difficulty of fine scree
cation is the best preparation for Wilfley table

Capacity, power, water, speed (74). CA
particles, and on the preparation they have h
up to 50 tons per 24 hr is not uncommon W
heard of 120 tons per 24 hr. These large feed
recrushing of tailing. With fine sizes, feed ma
POWER ranges from 0.25 hp with no feed, to
FEED WATER varies from 22 gal per min, with
pulp. WASH WATER varies from 2 to 12 gal
average pulp. SPEED of 240 vibrations per
length of stroke, 1 in; it may be shorter for fi
from 7 to 15¢ per ton, including labor, wate
cost of Wilfley table is $450, plus $15 for raff

22. Deister Ta

Deister sand concentrator No 5 (Fig 64
shaped deck covered with linoleum or oil
similar to those of Wilfley table, in ending
control of separation, but are arranged in grou

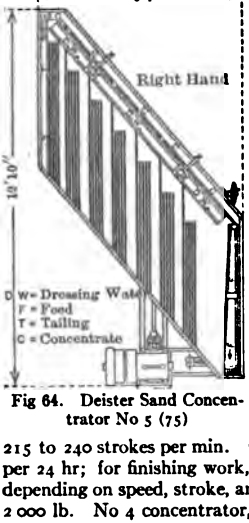

Fig 64. Deister Sand Concentrator No 5 (75)

jerky, vibratory
and causing pr
(75). Chief po
are: (*a*) thickne
as 1/8 to 1/16 in, a
gradually; (*b*) s
surface, the di
taken from the
(*c*) the rear end
on a line parall
control. To p
water, surface
under the linole
are 6 of these
slimes which wo
ably some of th
towards the dia

Feed. The
mesh (2 mm) to
make less midd
STROKE varies
215 to 240 strokes per min. CAPACITY for ro
per 24 hr; for finishing work, 15 to 40 tons.
depending on speed, stroke, and amount of fee
2 000 lb. No 4 concentrator, similar to No 5
pulp.

Deister slime table No 3 (Fig 65) is of the
(0.16 mm) and finer (75). The feed crosses th
in the wide spaces between the first riffles.
towards the concentrate side, the lighter porti
a chance for re-treatment in the second spa
several times, the material reaches a group

a portion of the remaining values is delivered at the concentrate are several wider riffle spaces, and finally another group of closely which complete the e is 3/16 in; speed 265 s per min; capacity, per 24 hr; shipping lb.

Vanners

re fine concentrators, from 1.13 mm to some cases as fine as rage about 0.75 mm. as an endless rubber ng slowly up a gentle Belt with its frame vibrate crosswise. the upper quarter, ed by the vibration, avy values to form a ubber surface. Con- us carried to head of elt travel, while the

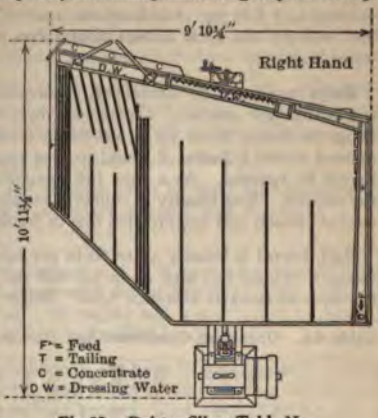

Fig 65. Deister Slime Table No 3

n upper film, which is washed down the slope. Among the best- rs are the Frue, Johnston, and Isbel, having a side shake, at right of water; and the Embrey, Triumph, and Gates, with an end shake.

1. Frue vanner will be taken for illustration (Fig 66). Belt is of 2-ply 0.175 in thick, with flanges to confine the pulp; width, either 4 or 6 ft. on a shaking frame by rollers *D*, 2 in diam and about 12 in apart (some- here belt sag is too great). The upper 2.5 ft of belt (the "cleaning plane")

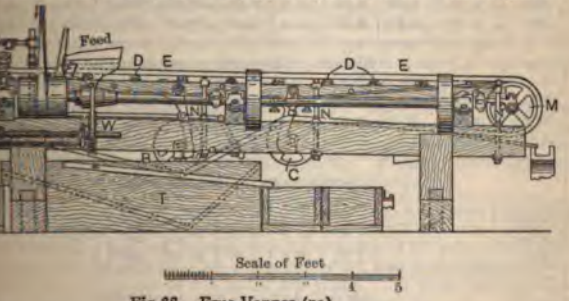

Fig 66. Frue Vanner (73)

er than the rest of the surface, or "roughing plane." The pulp feed is at these planes. At each end of the shaking frame is a roller about 1 ft roller *L* imparting the traveling motion to the belt, by pinion and worm om spouts 2, 1.5 in apart, is fed across the belt close to the head roller. around roller *L* the belt passes beneath roller *B*, which dips it in the water tank *T*. On leaving *B* any remaining concentrate is washed off both s. Belt next goes over guide roller *C*, which, by its adjustable bearings, htens the belt; thence it passes around tail roller *M*, and up the slope. ame is supported on a fixed frame of timber or cast iron, by 8 flexible *N*, *N*.

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Slope of belt is adjustable, and is 0.25 to o
and Triumph vanners. For treating very fi
as steep as 1.5 in per 12 in has been used. T
into tailing; too little carries tailing into c
water be used.

Belts are smooth or corrugated. Smooth b
corrugated for coarser. Corrugated belt h
flange to flange, 1/8 to 1/32 in center to cente
mineral grains is large; if found to give too
should be reduced. As a rule, the corrugate
the smooth. Best quality of rubber is require
cessive tension and compression due to passin

Belt travel is usually 30 to 60 in per min
vanners. Gates has used 80 in per min for
has been adopted at Morenci, Ariz. Writer h

Table 61. Operating Conditions and Details

Condition	Water, gal per min: Fed with pulp	Water, gal per min: For cleaning jets above and below	Ore, tons per 24 hr	Ratio by wt of total water to dry ore	Slope, in per 12 ft	Travel, in per min
4-ft side-shake vanners						
Max...	7.50	2.80	12.50	10.7	6.00	72.
Min...	1.50	1.00	4.00	2.4	2.25	22.
Av....	4.25	1.35	7.50	5.4	3.42	39.
6-ft side-shake vanners,						
Max...	7.00	5.00	16.67	....	4.50	72.
Min...	7.00	1.50	3.00	....	3.25	36.
Av....	7.00	3.21	7.41	....	3.75	46.
4-ft end-shake vanners,						
Max...	....	.....	3.75	16.7	3.00	26.
Min...	....	.....	3.75	16.7	3.00	26.
Av....	....	.....	3.75	16.7	3.00	26.
5-ft end-shake vanners,						
Max...	18.00	2.75	15.00	10.7	8.80	72.
Min...	18.00	2.75	5.00	7.1	5.40	48.
Av....	18.00	2.75	9.47	8.7	7.10	60.
6-ft tipping or undulating vanne						
Max...	3.00	1.50	18.00	....	6.0	...
Min...	1.50	1.00	8.00	....	4.5	...
Av....	2.25	1.25	13.67	....	3.0	...

low as 5 in per min. The catching power of corrugated belt is so unless run at slow speed, it carries too much quartz into the con- d slow running does not appear to enrich the tailing. Speed of chief adjustment; too rapid travel carries tailing into concentrate, rries concentrate into tailing, and also diminishes capacity.

s and throw. Usually 180 to 200 vibrations per min, with throw tes ran his fine end-shake vanner at 240 vibrations, with 5/8-in throw. ocking side-shake vanner has run at 120 vibrations, with 2-in throw.

ust be sufficient to maintain a perfectly soft bed.

ents all compensate one another more or less; especially true of lope; as slope increases travel must increase, and vice versa.

nges from 0.5 to 1 hp.

bed. Writer believes depth of 0.25 in is as much as should be al- is would be 12 grains deep, with 1/50-in (0.5-mm) pulp. Often a 5 to 0.2 in is preferred. Where the bed is 0.5 to 0.75 in deep, as ls, the settling is less perfect, and tailing is too rich.

ct from stamp mill is customary, the pulp passing through a copper ax to catch amalgam. If feed is from classifier, the classification good that no slime comes from spigots; then the first spigot and its h nearly half the values, but it may make tailing too rich to discard. igots and vanners yield little concentrate, but make very low tail- verflow slime is ready to go to canvas, or any other approved table. olates the two difficult products, first-vanner tailing and final slime, ordinary method leaves them mixed with the clean tailing of the later vanners.

Isbel vanners are side-shake machines, which have been more used, es- True, than any others. They make clean concentrate, due mainly to the wash-water jets (Fig 66) which efficiently wash away the quartz. On the he side-shake is apt to cause banks on each side of belt, which may be 2 or d may cause loss of values. The Isbel is a new vanner introduced by Utah and said to do good work. Its shaking frame is supported by leaf springs, Shepard and Searing, which maintain a perfectly rectilinear motion.

or suspended vanner, has a rocking or side-shake motion, due to 4 sus- which slope inwards and downwards towards the belt on both sides. The at and moved from one side, while it is depressed and moved towards e, the motion being reversed on return stroke. A distinct advantage is the de banks (sand corners) resulting from this peculiar motion.

Embrey, and Gates vanners have end shake, which produces cleaner tail- clean concentrate than the side shake. There is little tendency to form side se the flange does not interfere with the vibration of the pulp. This makes ailing, but the endwise swing of the wash water does not remove the last of rom the concentrate so well as the side shake. The Gates has a 3-ft belt, ope, rapid, short vibrations, and is used only on very fine pulp. The tailing and has to be re-treated on canvas tables.

e 62. Cost per Year for Vanner Repairs and Renewals (73)

	Maximum	Minimum	Average
and repairs on same	$103.00	$3.82	$36.90
airs.....	12.00	1.12	5.87
irs and renewals.....	115.00	4.94	42.77

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25. Overstrom Ta

Overstrom table is a rhomboid-shaped r
it is sometimes provided with the Wilfley di
the line of riffle cleats is usually cut off at rig
direction of vibration. Because of its shape
on the deck. Weight, 2 500 lb; cost, about
to 0.94 in. Having a long feed box, less wash

Gilpin county bumping table has an end bun
is fed across the table a little above the middle, an
centrate up the slope, to deliver it, while the upp
the wash water. Tables are mounted in pairs, ea
68 in is a plane surface, sloping about 3/8 in per ft
concave cylindrical surface, with axis across the tab
of lower part. It is operated by a 3-armed cam, sp
180 strokes per min of 1.5 to 3 in. Capacity ab
work where very low first cost is required and espe

26. Round Ta

These treat slime from 0.12 mm down to a
floating mud. They are best fed by classifier,

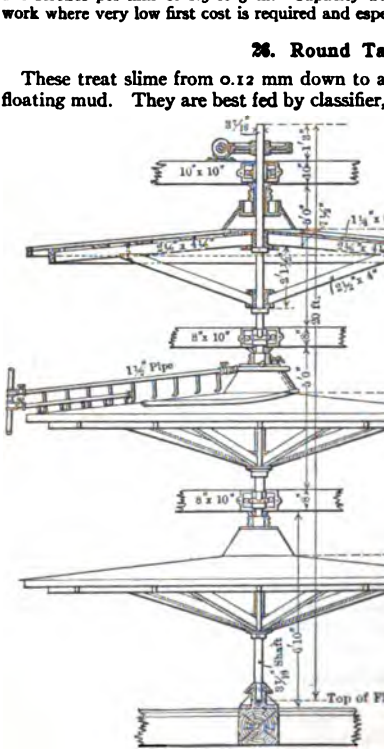

Fig 67. Three-deck Slime Table

on the table. A middling spray may be employe
mostly into the middling box under the table rim
into concentrate box. As the pulp and wash wate

ower less. This can be compensated by giving the edge of the table a steeper in H. S. Munroe's design. The wash water can be applied by concentric nter cone, or as a spiral spray, or both. Writer prefers the concentric for l spiral for coarser.

r of tables has ranged from 10 to 30 ft; 16 to 18 ft are most used, commonest. The real work of concentration is done mainly on f radius nearest the circumference.

rom 0.75 to 1.5 in per ft; ordinarily, 1 to 1.25 in per ft Finer pulp ed with steeper slope, coarser with flatter slope.

On an 18-ft table, discharging about 5 lb of water per ft of cir- per min, 33.92 gal per min are used, one-half in feed pulp, two- nainder in washing jets, the rest in the wash-off. Flow of film on smooth and regular, but is a succession of tiny waves. These aid ntration. A depth of about 0.08 in on crest of wave and 0.04 in in ommended, requiring 3 to 3.5 sec to reach circumference. DENSITY ges from 5.5 to 16% of solids. Linkenbach recommends 8 to 10%.

f table. Soft wood, cement, rubber, linoleum, and canvas have all Wood or cement have proved best, but if wood is allowed to dry, troys the surface. A moderately rough surface is advantageous, btained by using a coarse-sand cement. A good rough surface for work is made of ¼-in iron plate, coated with roofing pitch melted and mixed with 1-mm sand for roughness.

of decks. To save floor space, piping, driving gear, etc, multipli- cks is practiced in many mills. Anaconda Copper Mining Co has es at Great Falls plant and 20-deck tables at Anaconda.

rotation. The faster the speed the greater the capacity. But ex- d carries gangue into concentrate, and increasing water to prevent rry concentrate into tailing. One revolution in 41 to 155 sec ap- the proper range.

of an 18-ft table is 12 to 15 tons per 24 hr, according to fineness of he finer the pulp the smaller the capacity.

t at Anaconda (61). The 10-deck tables are 18 ft diam, sloping 1.25 in have steel frames with concrete deck, made with coarse sand to obtain . The original pulp contains 2% solids. This is thickened to 9% solids 8-ft Callow cones, now by Dorr thickeners, 28 ft diam by 3 ft deep, mounted and 90% of value is saved.

vs round tables. A vanner 4 ft wide by 12 ft long costs $350; e, $200 to $300, according to locality. Two-deck tables cost some- an twice as much as one deck. A vanner treats 4 to 5 tons of ulp; a round table, 12 to 15 tons classified pulp per 24 hr.

h stationary round table resembles the round table, except that the feed ray pipes, and wash-off jets revolve and discharge into their 3 or 4 seg- ers around the periphery. It has been largely used in Europe, but not in e surface, commonly concrete, is stationary.

lime tables have been tried and abandoned. They have very large area concentrate, small area for separating the gangue. The film deepens as it center; whence, its carrying power is greater, and tailing is richer.

27. Miscellaneous Slime Washers

rames are used for washing fine slimes at Cornish tin mines. They are gular wooden surfaces, built tandem for re-treatment of tailing on

cessive frames. After pulp has flowed over the ta
are turned up so that each table will discharge in
clean water is applied at same time, to wash dow
lever that tilts the tail boards is connected by a ro
that the two operations are performed together
washed down, the tail boards are dropped to ori
peated. The changes are made by hand or autom

Canvas tables are inclined rectangular tables,
water are distributed evenly across top of incline
sizing action of the film is more effective than o
gives a drag to the undercurrent and its meshes
the heavy particles settle. Action is intermittent,
separate launders at intervals, preferably with a br
than a hand broom and wears the canvas less.
feed, but are often fed with natural products. Th
to colloidal mud. They require about 0.042 gal w
per min of wash water per sq ft of surface area (
Gates canvas plant consists of tables 12 ft wide by
with No 6 duck, woven in one piece, with the warp
flow of a classifier, which is regulated to throw all

Carpet and blanket tables are sometimes use
a continuation of the same. They catch excess
from plates; also included grains, rich concentrate
are carefully removed every few hours, and washe

Table 63. Details of Canvas, Carpet, and B

Kind of table	Length, ft	Width, ft	Slope, in per ft
Canvas	10.0	12.0	1.50
"	16.5	1.7	0.75
"	42.0	1.7	1.12
"	39.5	1.7	0.70
Carpet	3.0	4.7	2.25
"	16.0	4.7	2.00
Blanket	3.5	1.7	2.00

C D = Cotton duck; B C = Brussels carp

Sluice riffles find their chief application in hyd

Kieves are heavy wooden tubs, with flaring sides
paddles are used for preliminary mixing of a charg
tub for final separation, as material settles. Res
layer returned with next feed, bottom layer consist
used to finish the concentration of fine products
(Richards' "Ore Dressing," p 726).

Building buddles (Richards' "Text Book," p 36
for material from 1.0 mm down, and are best fed wi
gradually builds up, until a 10 or 12-in bed has a
and separate products removed. The sizing take
finer grains (heavy mineral) are deposited near the
coarser grains of light mineral. Because of this ac
rent and of surface suitable for stopping it at its pr
conical, concave conical, and rectangular. They
Wilfley type of table, though some rectangular bu
Missouri, and are very serviceable in the first stage

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28. Flotation Processes

Flotation processes of concentration (87) are based upon the reaction of ıar forces: surface tension (of liquids), and adhesion (of liquids and gases ıd of liquids on gases). Surface tension is the force which causes a slightly to float on water. The degree of adhesion varies with different substances. presence of gas and water, or oil and water, show stronger adhesion to the oil to water. If oil and gas are both present, since oil itself has strong adhesion bble, the adhesion of sulphide to water is further diminished. The behavior inerals is the reverse; they show stronger adhesion to water than to oil or tendency which is increased by slightly acidifying the water. The attach- as bubble to a sulphide particle causes the sulphide to rise and float. These nomena, aided by mechanical apparatus, separate the sulphide from gangue e sulphide floating on the surface, the gangue settling.

on. Following minerals may be separated from their gangue: molybdenite, ena, mixed zinc-lead sulphides, chalcocite, sphalerite, pyrite, chalcopyrite, phite, tetrahedrite, argentite, cinnabar, tellurides, sulphur, and metallic and copper.

ns. Oxide and carbonate minerals have not yet been floated extensively, y closely resemble gangue minerals in adhesive properties. FILMING with h subsequent flotation of the minerals thereby coated with an artificial sul- been experimentally successful at the Magma mill, Ariz (A I M E, *Bull* 122). ontain minerals which tend to float naturally, yet are undesirable in the con- uorite, occurring with certain zinc ores, floats with the zinc concentrate, and enalized by smelters. Certain mixed sulphide ores of Scandinavia contain , which readily floats, and so impoverishes the concentrate that it is unmar- n ore in which all or part of the sulphides is bornite or chalcocite sometimes e; possibly when these occur as secondary minerals.

elprat process (87). The ore, ground to 40-mesh, is charged into V-shaped, ats. A 2 to 3% solution of H_2SO_4, or acid salt cake, preferably at a temp of roduced through pipes which reach to bottom of vat, and so aid in mixing ore n. The acid attacks the carbonates, liberating CO_2 gas bubbles; these attach o sulphide particles, which form a froth, and pass off at overflow. The gangue, by the selective action of gas bubbles, goes through a spigot in tank bottom.

process. Ore is crushed to 40 mesh, de-slimed, and agitated with cold acid 0.19 strength, in proportion of 4 or 5 parts of solution to one of ore. Me- de scum is removed, the acid solution drawn off, and the pulp agitated with to remove any remaining acid. High-speed agitation appears necessary to urfaces of the mineral particles. The cleaned material is next run into the where it is thoroughly agitated with water and oil. The oil is 1 part castor rts low-grade kerosene; charge, 2 to 3 lb oil per ton of ore, some chlorine gas d. The oiled pulp is then elevated by compressed air, and fed to the top of cone *E*, Fig 68. Pulp runs down over the corruga- ulphides floating on surface of water at *B* and over- o launder *C*. Gangue sinks and is drawn off at *D*. thoroughly all the sulphides, the gangue product ust be re-treated 2 or 3 times on similar cones and e sulphide particles are thoroughly aerated in the g, oiling, and elevating with compressed air, and eadily take up air films and float at *B*. This process fully treated ores carrying 5 to 6% carbonates, and high saving, probably due to feeding a pulp entirely lime. Capacity of cones is small, and plant covers a Initial cost is higher than for any other flotation oover gives it as about $500 per ton daily capacity, of 1 800 tons per day erected in Broken Hill district.

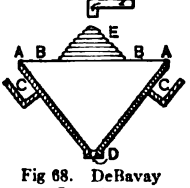

Fig 68. DeBavay Cone (87)

vacuum flotation process is also based on the selective action of oil for sul- r gangue minerals. Furthermore, if a solution of oiled pulp, sometimes con- d. be subjected to reduced pressure, bubbles of air or gas dissolved in water d, attach themselves to the sulphide particles, and increasing in size float to with the attached sulphides.

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In Fig 69, a mixture of pulp and water is run int
acid are introduced at *B*. The mixture is thoroughl
at 30 to 40 rev per min, and, upon applying a vacu
through pipe *S*, 25 to 30 ft long, to separating ve

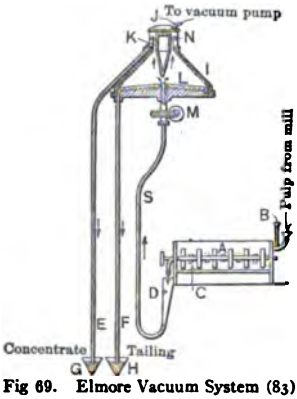

Fig 69. Elmore Vacuum System (83)

surface, due
and overflow
through tub
of separatin
outer edge b
and is discha
concentrate
With a const
but a small p
the concentr

Power re
chamber 5 f
including tha
mixer, and
1 man on va
on discharge
5-ft unit is f
complete, in
vacuum pun
f o b London
3 to 10 lb pe
among them

Texas residuum, fuel oil, tars, blast-furnace oils, oli
64 gives cost per ton at Broken Hill.

Minerals Separation process floats the
sulphides in a froth formed by use of a small
quantity of oil or other hydrocarbon, with
water, air, and acid.

Apparatus consists of a series of deep stirring com-
partments and spitzkasten. The number of these va-
ries; usually, 6 stirring compartments and 3 spitzkas-
ten are arranged as follows, regarding the passage
of the pulp: The pulp first passes through the 1st
and 2nd stirring compartments, connected with each
these it overflows into the first spitzkasten, where
froth. The gangue is drawn through the bottom
into 3d stirring compartment; thence similarly into
tated and forced to overflow into the 2d spitzkasten
and settlings for the 5th and 6th stirring compartm
spitzkasten, yielding concentrate and waste, the trea
plete. The action of the stirrers or agitators is to
mixture of ore, oil, and acid. Then, when the pulp
spitzkasten, the air bubbles, aided by the oil, exert
particles, floating them to surface, where they pass
be entirely cleaned in one operation; hence the re
about 3 parts water to 1 of ore. Charge of 2 lb oil
of ore. Live steam is introduced into the first con
to 70° C. Alcohols, turpentines, most of the essenti
available as froth-producing agents, having advanta
froth, if desired, without removal of oil.

Capacity varies with size of apparatus from
400-ton unit occupies a floor space of less than
of vertical height, and may be tended by one
for each agitator, or a total of 30 hp. In this p
as to quantity of slime in the ore treated. A ce

ly essential to the best frothing result, and the process has worked y on material 95% of which would pass 100 mesh.

pneumatic process (105) forms a froth by admission of air, under f 4 or 5 lb per sq in, through a porous bottom. Preliminary agita- p with oil and other reagents may be given in a Pachuca tank, or in ebble mill during crushing.

us (Fig 70) is a rectangular wooden box, 2 by 9 ft, 20 in deep at feed end, ailings discharge end. Sloping bottom consists of 4 layers of loose cotton rced with perforated or woven metal. Beneath the porous bottom are 8 nnected by separate valves to compressed-air header. Froth overflows both into launders; tailings discharge through gate at lowest point. When treating res, a "cleaner" cell is provided to re-treat froth from 2 to 4 "rougher" s from cleaner being returned to roughers; with high-grade ores the cleaner ecessary. Final froth is dewatered and filtered.

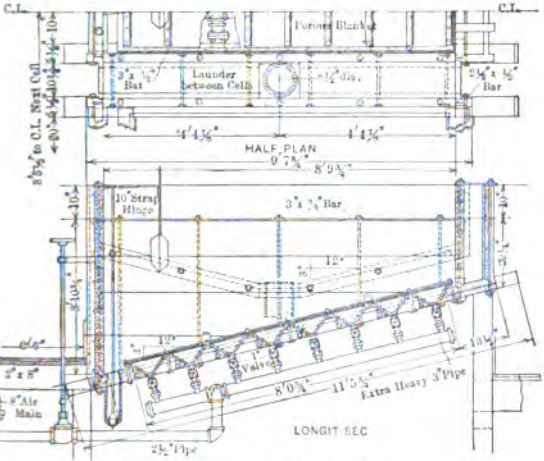

Fig 70. Callow Pneumatic Flotation Apparatus

ng details. PULP DENSITY: for mixed sand and slime, 30% solids recom- iminishing to 15% with fine slime alone. CAPACITY: 46 to 65 tons per 24 hr rougher. AIR: 8 to 10 cu ft free air per min per sq ft of porous bottom, pplied to compression of air to 5 lb in Root or Connersville blowers; 3.5 to 4 , equivalent to 10 to 12 tons per hp, or 1.25 to 2.5 kw-hr per ton. OIL: 1 to on feed, reduced to 0.3 lb by returning filtrate. LABOR: one man per shift te treatment of 2 000 to 3 000 tons per 24 hr. COSTS: for a 2 000-ton plant, nly, estimated, 6.75¢ per ton. Plant of 250 tons, estimated, 10¢. Magma , Sept–Oct, 1916 flota- ltering only, 41¢ per ton.

ng costs are difficult . Hoover, in "Con- g Ores by Flotation," s for Broken Hill dis- wn in Tables 65 and ng that same opera- U S would cost prob- % less.

Table 65. Treatment and Flotation Cost at Broken Hill (87)

Tramming to plant	$0.120	per ton
Re-crushing	0.402	" "
Flotation treatment	0.866 *	" "
Handling concentrate	0.120	" "
" residue	0.120	" "
Re-treatment of concentrate	0.362	" "
Royalty	0.242	" "
	$2.232	per ton

* Average of figures in Table 66

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Table 66. Flotation Costs per T

	Potter-Delprat	I
Power	$0.018	
Acid	0.302	
Oil	.....	
Supplies	0.120	
Labor	0.040	
Fuel for heating	0.120	
Assays	0.006	
Water	0.030	
General expense	0.174	
Total	$0.810	

DRY CONCENT

29. Magnetic and Electros

Applications. Magnetic separation is em strongly magnetic iron ores, as magnetite, b minerals. The separation of sphalerite from p nary roasting to convert non-magnetic ore in Other instances are: separation of franklini lemite, calamine, and calcite; tungsten from chalcopyrite; rutile from apatite; rutile, ga other; garnet from garnetiferous rock; ga minerals from corundum; siderite from cryol from gangue; hornblende from valuable m chromite from other heavy minerals. Sizin when minerals have nearly the same magn be fine enough to set free all the values. magnets see Richards' "Ore Dressing," p 790.

Cost of magnetic separation comprises needed, attendance, repairs, and electric cur the separation is easy or difficult. With a goo separation alone, not including the preparatio 25¢ per ton. The total cost, including prepa should not exceed 50 to 75¢ per ton.

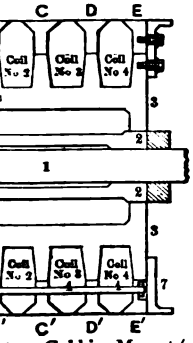
Types. (a) Magnetic separators of primary n poles causing the separation are energized direc Cobbing magnets, and Wetherill and Ball-Norton of secondary type, in which the separating magnet primary magnetic field in which they lie. Exampl Knowles, and Gröndal separators.

Cobbing magnets may be used to remov present in ordinary crude ore, before the ore magnets are suspended over a belt conveyor a off by hand at intervals. Their usual functio waste before fine crushing, thus saving cost o be run wet or dry.

Firmstone magnet (Fig 71) consists of a cylind shaft. The drum comprises: shaft 1; magnet co guttapercha ball 5, pierced to make a water-tigh

king a water-tight joint; brass rings 7; and *AA'*, *BB'*, *CC'*, *DD'*, and *EE'*, onstituting magnetic poles. Coils 1 to 4 are so wound that *AA'*, *BB'*, etc. ely north and south magnetic poles. When running dry, the material is y a conveyor belt, and carried off by a belt running on drum itself at right veyor belt. When running wet, the magnetic drum, which is water-tight, x. The water which brings in the pulp washes away the tailings, while the s removed by a belt. Other cobbing magnets are those of the Manhattan the Eastwood Improved Lifting, and the Bilharz Cobbing Magnet.

l magnet (Fig 72), made by Stearns-Roger Mfg Co, Denver, Colo, e of high magnetic intensity, for weakly-magnetic dry ores (77).



stone Cobbing Magnet (77)

Ore is distributed from feed hopper, evenly across a wide conveyor belt, which passes between upper and lower magnetic poles *aa* and *bb*. Poles of upper magnets *a* are wedge-shaped, to concentrate the magnetic force, those of lower magnets *b* being flat with rounded edges. Magnetic material jumps up-

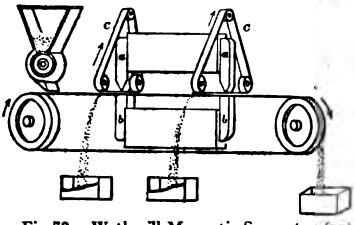


Fig 72. Wetherill Magnetic Separator (77)

stronger pole, the cross belts *cc* preventing the material from adhering to d carrying it out of the field. This machine has 1, 2, or 3 double magnets, d with a rheostat for regulating strength of current. Table 67 shows work machine on ore of different sizes, and illustrates separation of minerals of gnetic strengths on one machine.

width of CONVEYOR BELT of Wetherill separator for treating weakly- res is 18 in. The narrower the belt, the less electrical energy is re- emove a mineral of same magnetic susceptibility. Proper thickness 0.125 in for very weakly-magnetic minerals, 0.25 in for medium, and r strongly-magnetic minerals. In practice the standard machines n belt. When minerals of stronger magnetic power are to be removed e of weaker magnetic power, the poles of first magnet may be lowered, o of brass 0.125 in thick placed between it and the belt. Proper elt must be determined for each ore. The TAKE-OFF is an endless ride, and 1/32 in uniform thickness. Its speed should be such as to lear of the conveyor belt, and it depends also upon quantity of ore Proper DISTANCE APART of the MAGNETS must be determined by aries from 0.5 in on weakly-magnetic ores, to 1.5 in on strongly These adjustments govern size of feed; for weakly-magnetic ores ger than 0.03 in seems advisable; for medium-magnetic, nothing 0.25 in; for highly-magnetic ore, nothing larger than 0.375 in. L ENERGY required by No 3 machine having 3 double magnets, with 000 and 100 000 ampere-turns, respectively, is 50 amperes at 125 .4 hp. MECHANICAL ENERGY depends on speed, and varies from 0.5 WEAR AND TEAR is almost nil. Take-off belts, practically the only newal, last 1 to 12 months, and cost $1 to $2 each. Two men can to 16 machines.

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Table 67. Capacity and Conveyor-belt Sp
Separator with 18-in

Material treated	M = Magnetic product N = Non-magnetic produ
Colorado zinc-lead-iron sulphides	Zinc blende (black jack) M product
Broken Hill, Australia, tailings	Rhodonite as 1st M produ Zinc blende (black jack) 2d M product Galena and gangue as product
New Jersey franklinite ores	Franklinite as 1st M prod Silicate of manganese m erals as 2d M product Zincite, willemite, and c cite as N products
Port Henry, N Y, magnetite ores	Hornblende as M produ Apatite as N product
Roasted iron-zinc sulphide middlings from wet mill	Clean roasted pyrite as M product Pyrite and blende m dlings as 2d M produc Zinc blende and gangue N product
Magnetite ores*	Magnetite (90% of feed) M product

* A few dry battery cells were used in this case current could not be cut low enough with rheosta

Ball-Norton belt-type magnetic sepa highly-magnetic ores, combines simplicity

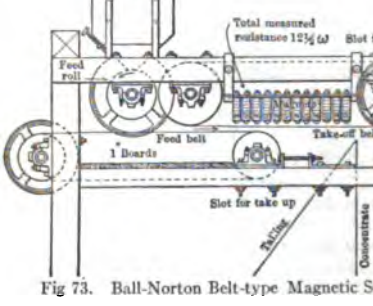


Fig 73. Ball-Norton Belt-type Magnetic S

magnetic field, where magnetic mineral is lift second poles. A large part of non-magnetic ing, and remainder is shaken off. Take-off whence the magnetic mass is spread out m separators have handled ores as coarse as o material up to 0.25 in is from 20 to 25 tons.

n drum-type separator (Fig 74) consists of 2 revolving drums of construction, and operates on the principle of alternating polarity. o first drum, and as the magnetic par- it, the non-magnetic the hopper. As the beyond the magnetic rum, they are thrown l force on to second either revolves faster agnetic strength, so articles adhering to not adhere to second. ering to second drum er a partition as soon the magnet. Thus

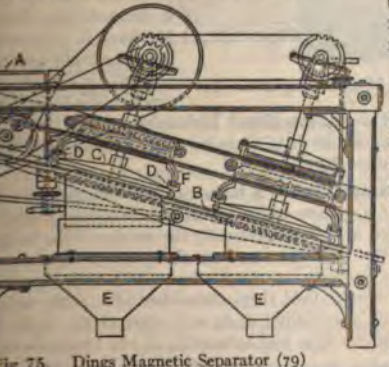
Fig 74. Ball-Norton Drum-type Magnetic Separator

delivers a non-magnetic product, a weakly magnetic middling for nd a strongly-magnetic product. Power required is 1 to 1.5 hp, for ach drum, and 0.5 to 0.75 hp for revolving the drums. Capacity, 24 hr of 0.25-in stuff, consisting of magnetite and quartz.

. Details of Ball-Norton Drum-type Magnetic Separator (77)

	Distance between drums	Drums				Capacity, tons per hr	Size of feed
		Rev per min		Current, amperes			
		First	Second	First	Second		
	.....	....	....	.....	.....	15 to 20	Through 1.00 mm
	1.0 in	40	50	10.5	13.0	12.5	" 2.12 "

gnetic separator, type "MM," is especially designed for treating ores o a low magnetic intensity. It can make in one operation 5 products, viz: non-magnetic material and 4 grades of magnetic product (78).



ig 75. Dings Magnetic Separator (79)

Ore is fed from hopper *A* (Fig 75) to a vibrating conveyor *B*, over which it passes in a thin layer through 4 zones of separation, covered by the rims of rotating disks *CC*, carrying secondary magnets *DD*. The latter attract and carry away the magnetic material, discharging it over the sides of the conveyor into hoppers *EE* for final delivery. Magnets *DD* are saturated, while over the conveyor, by the pole pieces *FF*, of mary electromagnets, but are automatically demagnetized as they pass out of c circuit to the neutral position, where rims of the wheels overhang the conerfect and natural discharge is thus given on both sides of conveyor, resulting

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in 4 grades of magnetic material. Some of the se
per day. The single- and double-magnet machi
watts; power 0.5 and 1 hp.

International separator (80) is a machi
sisting of a cylindrical secondary magnet c
shaft 8 (Fig 76) and revolving between th
shoe magnet 2,2. The armature surface ma
pending upon the permeability of mineral tr

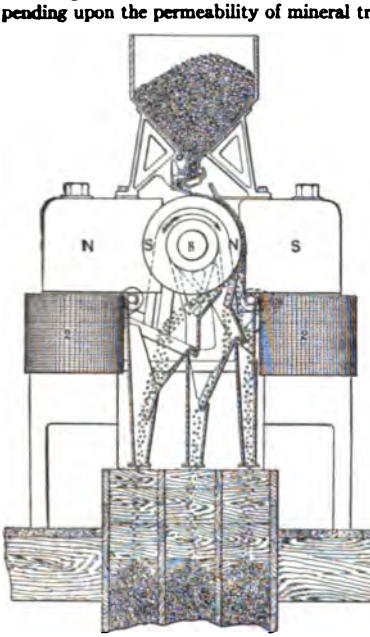


Fig 76. International Magnetic Separator (8

to material treated and size of machine. M
roasting, the cost varies from $0.60 to $1.25
ment itself should not exceed 50¢ per ton.

Gröndal No 5 separator (82) is a hori
77), which has within it a system of electro
4. Feed is brought into a weir box beneat
cles are drawn out of the water and again
action due to changing polarity, are carri
they pass out of the magnetic field they ar
for removal of tailing at 5, middling at 6,
At Herrang, Sweden, the machine has a cap

Roasting and calcining for magnetis
non-magnetic as to be uninfluenced by the
magnetic by a suitable roast. Some non-
verted into magnetic sulphides by a quick

roast be continued long enough to drive off all the sulphur; also, carbonates and oxides are made strongly magnetic by calcining s and subjecting the oxides to a roast in a reducing atmosphere.

Minerals Rendered Magnetic by Roasting or Calcination

Sulphides	Oxides and carbonates
Pyrite (FeS_2)	Hematite (Fe_2O_3)
Marcasite (FeS_2)	Limonite ($Fe_2O_3+3\,H_2O$)
Chalcopyrite ($FeCuS_2$)	Siderite ($FeCO_3$)
Bornite ($FeCu_2S_3$)	Wolframite ($FeMnWO_4$)
Arsenopyrite (FeAsS)	Chromite ($FeCr_2O_4$)

; sulphides, the practice in the U S is either to form magnetic uiring 1 to 45 min, or to roast from 2.5 to 4 hr, producing magnetic a chemical change anal- following takes place:

$_2O = Fe_7S_8 + 6\,SO_2$

inued the result is

$O = 7\,Fe_3O_4 + 24\,SO_2$

were carried further, e formed, but this may d into Fe_3O_4 by exposing ast to a reducing atmos- adding carbon to it. commonly rendered reducing the ore at red bsence of air, by solid matter, as

$C = 4\,Fe_3O_4 + CO_2$

reducing gases (usually s

$CO + H = 4\,Fe_3O_4$
$CO_2 + H_2O$

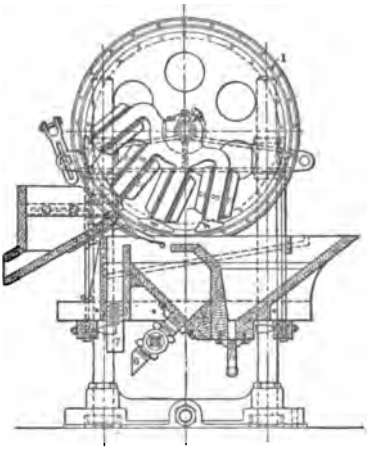


Fig 77. Gröndal Separator (82)

are decomposed thus: $FeCO_3 = FeO + CO_2$, the molecule FeO ed to Fe_3O_4 by taking oxygen from atmosphere.

The first requisite for a roasting or a calcining furnace is that of air entering it shall be readily and efficiently controlled. can be regulated by a pyrometer, but it is usually done by eye. ed in some furnaces all the time and in others part of the time. quid, solid, or gaseous. Roasting may be done in a revolving as in Joplin, Mo, in reverberatory furnaces. Furnaces of the ype are considered preferable to revolving cylinders, because of t, less space required, and lower cost of maintenance and operation. cted, $5 to $6 per sq ft of hearth area. Wilfley makes two forms e tower and the revolving circular hearth type.

tic separation. Mineral particles, when subjected to action of ity, from poles far enough apart to prevent sparking, are charged ty at varying rates according to their conductivity. The best

conductors are charged first, and are immedi
giving a means of separation.

Huff machine (79), shown diagrammatic
direct current from a dynamo. It gives
ductors, which are thrown off; (*b*) mediun
magnetic pole and ha
(*c*) the poor conduc
their box. Dimensi
cluding generator, 5
per day; power, 1 hp
best sizes of feed, 6 t
and finer; feed must
Denver; operating c
royalties.

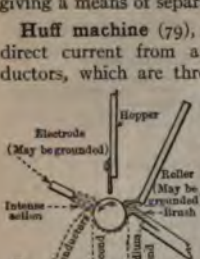


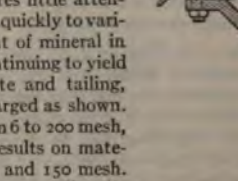
Fig 78. Huff Electrostatic Separator (79)

36. Pneumat

A light air blast is
for concentration of
removal of dust from natural or pulverized m
gravel, or ores in general, in districts where

Examples. Edison dusting chambers, for se
gangue from magnetic concentrates; Hochstedt a
Mumford and Moodie separator, which yields gra
tinuous air blast. The Pape-Henneberg, Clark
Fried gravity dry-process separators project the
other than an air blast, the grains being grade
Paddock and Krom pneumatic jigs subject the or
mittent pulsations of air, the particles settling ac
machines here mentioned are described in Richar

Plumb pneumatic jig (Fig 79) uses the s
jig, with substitution of air for water (85).
tent pulsations of air, admitted
through rotary valve *R*. It is
stratified on the screen *S*, the
heavier particles *B* going to the
bottom and the gangue *A* to the
top. The operation is auto-
matic and, once the feed is
regulated, requires little atten-
tion, responding quickly to vari-
ations in amount of mineral in
the feed, and continuing to yield
clean concentrate and tailing,
which are discharged as shown.
It treats ore from 6 to 200 mesh,
but gives best results on mate-
rial between 12 and 150 mesh.
Close sizing is not ordinarily
required. All sizes of the jig
have same cross section, 3½ in,
but vary in width. The smaller
size, with screen surface 3½ by 24 in, weighs
has capacity of 4 to 10 tons per 24 hr. T
36 in, weighs 300 lb, costs $450, and has



Fi

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, 40 to 50 lb at the rotary valve. For the air supply, about 0.5 hp er ton per day.

Steele & Steele dry concentrating table (86) resembles the e in its tilting frame and head motion. Fig 80 is a diagram (79) ion of top, in which *A* and *B* are impervious , *D* pervious cloth, and *C* supporting ribs. trips are narrow, tapering, and terminate gonal line. Above is the layer of pervious , and from below rises the air current, at re per sq in. The sand is stratified on the per strips having an effect similar to the ilfley table. The head motion propels the ns toward the concentrate end, while the s roll down the slope. Dust must be removed and ore closely sized nent. Sizes treated range from 4 to 200 mesh. For the finest mate- ers state the capacity as 1 000 lb per hr; for 8 to 10 mesh, up to 6 000 lb e table deck is 5 by 12 ft; weight complete, 2 500 lb; power required, e fan requires from 0.5 hp for slimes, to 6 hp for 8-mesh material.

Fig 80. Diagram of Sutton, Steele & Steele Pneumatic Concentrator

ater for concentration. The advantages of air over water as a g medium are: water not everywhere available, no trouble from ducts are dry and ready to ship. Disadvantages: fine crushing is vith resultant fines and dust; preliminary drying is expensive; tween close limits usually required; minerals of nearly same sp gr lt to separate than by water concentration; clayey matter can not ated nor soluble salts dissolved.

ACCESSORY APPARATUS

31. Samplers and Sampling

; is required to determine value of crude ore, tailing, concentrate, other product, at different stages of concentration.

f ore taken for a sample depends upon size of grain, and character of ore as and occurrence of values; whether rich and "spotted," or well disseminated. densed from Richards' "Ore Dressing," p 852, gives ratios between weight l size of largest permissible grain, for different ores. The first requisite for mpling is that the sample and "reject" shall be uniform in composition at he process of division. After each "cutting down" of sample, the ore should er, and mixed before next cutting.

Maximum Size of Crushed Ore Particles Permissible in Sample

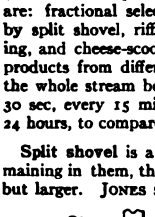
Low-grade and uniform ores		Medium ores		Rich and spotted ores	
mm	mm	mm	mm	mm	mm
200	100	75	50	30	5
65	35	24	16	10	1.7
20	10	7.5	5	3	0.5
6.5	3.5	2.5	1.5	1	0.15
2	1	0.75	0.5	0.3	0.05
0.6	0.3	0.25	0.15	0.1	0.02
0.3	0.15	0.12	0.07	0.05	0.01

e column under each class of ore implies a distinction into two varieties as. The second column of the table applies to ore that is very low-grade while the last applies to very rich and unusually spotted ore.

STANFORD LIBRARIES

Hand sampling is done by the following meth
coning and quartering. (*b*) Fractional selection b
(*c*) Cheese-scoop or pipe sampler. (*d*) Grab or
are: fractional selection by shovel, for sampling
by split shovel, riffle, and Jones sampler, for sm
ing, and cheese-scoop, pipe, and grab samples ar
products from different machines, or of mill tail
the whole stream being taken for some definite
30 sec, every 15 min. Mill sampling should be
24 hours, to compare night with day work.

Split shovel is a fork, of which the prongs a
maining in them, the rest falling between. The
but larger. JONES SAMPLER is a riffle of which t
directions, discharging the ore
any of these devices the scoo
the largest ore particle. (F
also Sec 25, Art 4, 5, 6; Sec



Mechanical sampling
cheaper than hand sampli
possesses the following fea
fectly mixed and steadily
each cut, at a uniform rat
to stream; ore once in the
out; machine is easy of ac

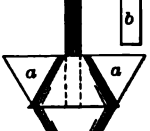


Fig 81. Vezin Sampler (88)

Vezin sampler (Fig 81
bolted together at their
spider, keyed to a revolvi
ries one or more scoops *a*
entering the scoops and pa
it is conveyed to sample b
of the ore falls into a hopp
or speed of rotation, may be regulated to
Vezin sampler is also made in a form which

Table 70. Weights and Prices

Sizes	Single	
	Weight	Price
1	500 lb	$100
2	850 "	155
3	1 800 "	300

No 1 size is for ore 1 in and finer; No 2,

32. Automatic Feeders

Hendy's Challenge feeder, for gravit
circular table *a*, inclined about 12.5°, and
beneath.

A sheet-iron hopper *v* delivers ore on one side o
certain point a fixed scraper *c* directs it into the sta
from hopper is regulated by gate *b*. Power is t
ing a buffer on rod *r*, which transmits motion to th
mittent rotating friction device at the left. Thick

et on the buffer, and thus the rate of feed. This feeder may be used for r than stamps; it is then operated by cam and spring acting on lever *k*.

der (Fig 83) con- up *a*, hung by two and oscillating be- iute *c*, on top of ed hopper is set. e of the chute is a l partition *d*, the which just clears f the stirrup. The k sides of the chute rom the partition. ps on the oscillat- e partition forces ly from each side. up is connected by crank pin, which o vary the rate of eder was designed rolls, but may be ther uses.

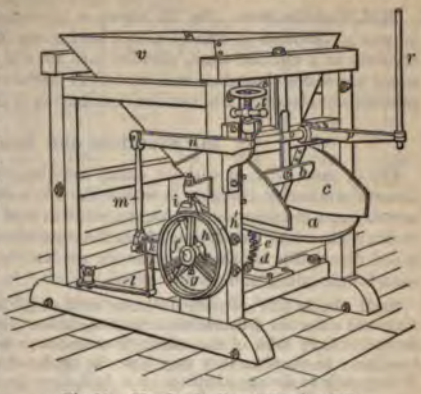
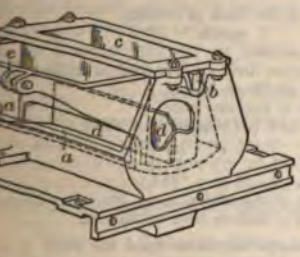


Fig 82. Hendy Challenge Feeder (88)

lers, driven by eccentrics or cranks, are extensively used in mills, the discharge gate of an ore bin.



ig 83. Gates Ore Feeder (88)

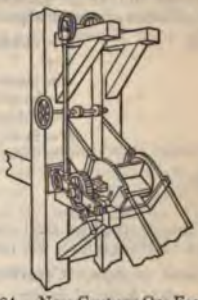
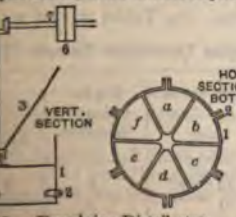


Fig 84. New Century Ore Feeder

tury feeder (American Concentrator Co, Joplin, Mo), Fig 84, is peration, with stated capacity up to 50 tons per hr; cost, $100.



5. Revolving Distributer

Distributers are used to divide a stream of ore evenly between two or more machines (mill distribution), or to distribute ore uniformly to a single machine (machine distribution). Any mode of distributing ore by taking part of the stream all the time, as by drawing consecutive spouts from a launder, gives uneven products. An exception to this is the overflow of tanks, provided the same depth at all points. Gates distributer (Fig 37) divides a tank as many spouts as are needed, of equal width and depth. They

STANFORD LIBRARIES

all carry about the same quantity of water
approximately the same composition.

Mill distributers. Fig 85 shows a rev
hopper 3, mounted on spindle 5, and having o
revolves in a circular tank, divided into sev
spout 2. Sizing launders and classifiers are
preliminary classification, besides distributing

33. Elevators and

Ore or sand, with or without water, may b
tinuous stationary troughs, in which the ore
screws; continuous-moving troughs, such as
chains; elevators, consisting of buckets on
tailed description of these appliances, see Se
used for elevating tailings and slimes.

Sand wheels (88) are vertical wheels, having fi
Sand and water are delivered to them at the lowes
on one or both sides of the wheel; they dischar
launders. In diameter they vary from the small
Wilfley table to elevate middling, up to 65 ft diam
They are usually built entirely of iron, though the
plicity of construction and operation makes runni
The wear is merely that of the journals and bucke
a belt elevator, as the material can be delivered t
The elements of a sand wheel are as follows:

Let R = radius in ft at the back o
r = " " " " " mouth
n = number of buckets,
l = length of bucket divisions
t = thickness of the same, inc
B = inside width of buckets, i
C = capacity in cu ft per revo

then $$C = B\left(\frac{R^2 - r^2}{2} -$$

The proper speed for sand wheels is ½ the critic
which centrifugal force will prevent the buckets fr

Sand pumps. Centrifugal pumps are m
any other form of pump, as they have no va
grit. The Traylor centrifugal pump is so co
the runner, solid liner, and follower, which ar
metal and are readily replaced. Definite r
lift with any given pump give best results.

Table 71. Capacities of Traylor

Sizes	Per cent of solids, cu yd solids p		
	10%	15%	20%
2 in.....	5 cu yd	8 cu yd	12 cu
3 "	10 " "	15 " "	21 "
4 "	14 " "	21 " "	28 "
5 "	21 " "	32 " "	43 "
6 "	30 " "	45 " "	60 "

. **Proper Speeds (rev per min) for Traylor Centrifugal Pumps at Specified Lifts (ft)** (88)

Capacity, gal per min	10	20	30	40	50	60	80	100
250	426	603	737	852	953	1 045	1 210	1 348
500	426	603	737	852	953	1 045	1 210	1 348
1 000	402	569	697	805	901	987	1 143	1 274
1 600	362	512	626	724	810	887	1 027	1 145
2 500	302	427	523	604	675	740	857	955

and pump, made by J. H. Frenier & Son, consists of a spiral steel g a rectangular cross-section of constant area. There are no valves; nd sand are lifted by virtue of a hydrostatic head in each turn of the pump being partly immersed in water. Height of lift depends um of these hydrostatic heads, or number of turns of spiral, and speed of revolution. The makers recommend 20 rev per min. dimension of spiral passage is about 2.5 in. This pump is made smallest has outside width of 6 in, and diam of 44 in; maximum 000 gal per hr; maximum lift, 12 ft; shipping weight, 1 100 lb; Largest has outside width of 10 in, and diam of 54 in; maximum 500 gal per hr; maximum lift, 22 ft; shipping weight, 1 670 lb; Frenier pump is considered the best for lifts within its capacity, rifugal pumps require less attention and are easier to install. But pumps are not suitable for ores containing soft minerals, upon which strong pulverizing action.

os for water supply etc, see Sec 40.

34. Launders; Unwaterers

s should convey given quantities of sand and water with minimum oss in mill height. Slope must increase with proportion of solid quid; with specific gravity of the minerals; and with increase in s in pulp. Table 73 gives slopes adopted by John Carkeek, Butte, naterial usually passing through coarse concentration works.

Table 73. Carkeek's Slopes for Launders (84)

Size of ore	Slope, in per ft
2 in to 1 in	9.33
1 " " ½ "	8
½ " " ¼ "	6.66
¼ " " ⅛ "	5.33
⅛ " " 1/16 "	4
1/16 " " vanner material	1.60
Table or vanner material	1.33

ult to state any exact formula, and it is probably best in each case nt before installing a system of launders. Table 74 gives examples from mill practice, water being used in all cases.

ers are to diminish the water carried by sand, or the sand carried Callow cones, settling tanks with overflow for water and spigot dis- and, as well as other clarifying devices and pulp thickeners, are ," and will be found under classifiers (Art 17 and 18). FLEMING

STANFORD LIBRARIES

Blake-Denison weigher is an automatic
which a short section of conveyor is suspend
passes over this section. If a 6-ft section of c
records the weight of material every time th

In Fig 86, steelyard 1 is balanced to suit the un
accurately in proportion to the load. A gripping
conveyor speed, grips the steelyard in the positio
quadrant 3 gages the weight, and the recording m
resistance of steelyard to load is provided by a ho
in such a way that it forms a dash pot 6, prever
steelyard is always in a position indicative of th
actions are effected by 2 cams upon shaft 7, oper
revolved by the conveyor itself. No auxiliary p
said to be accurate to within 0.5%.

GENERAL MILL

36. Mill Work; Mi

Certain general principles, which apply to a
operation, are of especial interest to the ore d

Location should be at the most advanta
of crude ore and delivering of products; po
disposal of tailing; room for future growth
slides. Other things being suitable, mill shou

Mill site. There are 3 classes: (*a*) side-
level site, with a sloping mill; (*c*) level site,
form of concentrating mill is single-story, bu
or a flat location. In this case, ore is receiv
pass by gravity through the crushing machi
to the trommels, whence it passes through t
middlings only being elevated for re-treatme
sign should be such that the middlings can
and simplest manner, without unnecessary d
generally followed. On the other hand, it i
separate story or terrace merely to avoid the
constitute only a fraction of the original ore.

Bins. Receiving bins should hold at leas
may provide for break-down of the mine h
bottoms are self-emptying and preferable to
bins are sometimes used, so that a stoppage i
complete shut-down. Bins should be const
and shocks due to dumping ore (Sec 12, Art

Power. Water power is the cheapest, w
motive power in most modern mills, becaus
steam. It permits a central power plant, loca
Separate motors are often used for individual
The induction motor is desirable for its qu
starting, and comparative simplicity (Sec 43).

Table 76 gives computed power required for co
Copper Co. The driving engine is a Reynolds-Co
ning at 100 rev per min with 140 lb steam pressure.
ments of power consumed by concentrator No 1.

Table 76. Computed Power Required for Concentrator (83)

	Horse-power
nmels, 4 by 5 ft and 4 by 8 ft	10
n elevators, 46 ft between pulley centers	20
16 by 36-in rolls at 80 rev per min	44
compartment bull jigs (4 active)	8
compartment middle jigs	16
e " sand jigs	16
atering trommels	1
trommels	1
el wheels, with shafting	3
rifugal pumps, 1 200 gal per min, 40-ft lift	60
Bryan mills	144
ley tables, with line shafting	25
Frue vanners, with line shafting	8
rifugal pumps	25
ing launders	3
lling elevators	5
elevators	3
n of engine and remaining shafting	80
tal on mill engine	472

Table 77. Water Consumption, from Practice (92)

al per 24 hr	Mill capacity, per 24 hr	Water per ton of ore		Remarks
mills	tons	gal	tons	
60 000	150	2 400	10	60 stamps
79 676	150	1 198	5	40 "
16 000	50	3 325	14	20 "
96 336	60	1 606	7	10 "
ls				
60 000	400	5 400	22.5	.....
00 000	1 200	3 333	13.9	.....
60 000	1 800	3 200	13.3	.....
shery				
00 000 (*a*)	1 000 (*a*)	300	1.25	(*a*) 10 hr
44 800	480	2 385	10	.....
p per-sul phide				
52 000	8 800	5 040	21	.....
00 000	3 000	8 300	34.6	.....
-sul phide mills				
40 000	1 000	1 440	6	(*b*) In mill circulation
20 000 (*b*)	.....	720 (*b*)	3	
40 000	6 000	1 440	6	
p per mills				
75 000	1 100	250	1.04	.....
50 000	500	1 500	6.26	.....
ior, native-cop-				
00 000	2 080	9 615	40	Steam stamps
00 000	1 260	9 524	40	
00 000	1 500	6 667	27.8	

erection depends upon: capacity; mill site; accessibility as regards tion of machinery; material used; type of construction, etc.

ill of North Star Mining Co, Cal, cost $74 600, including an excavation 18 000. Capacity, 135 tons per 24 hr, crushing by stamps, amalgamating, g, and cyaniding both concentrate and tailing.

STANFORD LIBRARIES

Ivanhoe mill of Minnesota Silver Co, B C, cost
hr, treating a silver-lead-zinc ore, saving all three m
Concentrator No 2, of Cananea Consol Copper
2 000 tons per 24 hr. It is of the side-hill type, t
resting on concrete foundations, iron sides and w
foundations separate from those supporting the mil
Cost of a 50-ton zinc-lead concentrator in Col
sphalerite-pyrite roasting and magnetic separation
crushing and screening plant, approximately $35 00
Table 78 gives complete cost of erecting a 1 500
Montana, exclusive of power plant. The mill is in 3
as 612.3 indicated hp.

Table 78. Cost of Erecting 1 500-ton Cop

Excavation	
Stone masonry	
Building	
153 Evans jigs	
18 double-deck Evans round tables	
Excavation for storage bins	
Masonry " " "	
Timber work for storage bins	
Trestles and approaches for storage bins	
Machinery, except jigs and tables	
Piping	
Shafting, pulleys, bearings, etc	
Wiring and electric lights	
Six 2-compartment Harz jigs	
Six 1-compartment " "	
Total	

According to W. R. Ingalls (*E & M Jour*, Vol 90,
works, computed on basis of tonnage treated per
per ton; Joplin, for treating mixed sulphide ores req
larger capacity, built of wood, 60¢; Joplin, larg
San Juan county, Colo, $1.33; Boston Cons, Uta
mill, $1.85; Ohio Copper Co, Utah, $1.50; New J
smaller magnetic plants, $3 to $4 per ton annual c

Operating costs vary even in same district,
of mill; (*b*) nature of process, the more compl
tinuity of running; (*d*) efficiency and cost of la
plies, fuel, etc; (*f*) power used and its mode of a
to delivery of ore, shipment of concentrate, an
of management; (*i*) character of ore, and its
costs are decreasing, because of the discovery
of established methods and apparatus. (See

Design and arrangement of machines.
considerations are: strength, without superfluo
number of parts; avoidance of unnecessari
machines of same make in any one mill, thus
and reducing repair stock.

Machines should be arranged as compactly as
along continuously through the breaking, coarse
sections. Many large mills have two or more si
These may be confined to crushing and coarse con
throughout entire system of concentration. This i
for repairs in a part of the mill, and also as a means

Table 79. Milling Costs from Practice (92)

f company, e or mill	District	Tons treated yearly	Cost per ton	Remarks
rine.....	Michigan.....	344 062	0.22	.....
rk.....	"	618 543	0.19	.....
.....	"	649 932	0.18	.....
pion.....	"	671 785	0.22	.....
untain.....	"	506 942	0.21	.....
et & Hecla " "	"	.....	0.40	Amygdaloid { Includes transp to mill
	"	.....	0.50	Conglomerate { Includes transp to mill
& Montana	Montana.....	.....	0.50	.....
.....	S W Arizona.....	347 020	0.94	.....
.....	" "	152 273	0.59	.....
.....	Bingham, Utah..	.....	0.60	Average cost
d r Hill & van	Wardner, Idaho...	.....	0.255	.....
.....	Colorado.....	.....	0.630	Average cost
Vest.....	Park City, Utah.	489 415	1.00	1900–1906
.....	S E Missouri.....	.....	0.30–0.75	1 000-ton plant
.....	Flat River, Mo...	.....	0.25–0.34	.....
Zinc Co...	Joplin, Mo.....	121 291	0.23	1905–1907
Tread-	Douglas island, Alaska	.....	0.17–0.36	.....
.....	Grass Valley, Cal.	.....	0.46	.....
take.....	So Dakota.....	.....	0.44	Milling and amalgamating
Bird.....	Colorado.....	.....	1.25	Cyaniding $0.61 per ton
.....	El Oro, Mex.....	1.027 282	0.77	Cyaniding $1.11 per ton
y mines.... "	Eastern U S.....	337 469	0.62	Magnetic separation
	New Jersey.....	.....	0.50	.....

nachines or groups of machines. As a rule, where two or more similar machines or a certain amount of work, it is better to arrange them in parallel than in series. nachines in couples, one to make clean concentrate, the other, clean tailing, is in practice.

ng of concentrate. After being unwatered, or settled and dried, ate should be delivered to railroad cars, or other shipping conveyance, nically as possible. It may go by wheelbarrows direct to R R cars, rrows or conveying belts to bins, and thence to the cars. Barrels or used only for small lots, or for rich ore to guard against losses, or nsiderable handling in transit must occur. It should be remembered tonnage of concentrate to be shipped is a small proportion of the tonrude ore entering a mill; hence, it is not economical to add to mill height or the sake of discharging concentrates by gravity.

sal of tailing. A mill site should provide for ample room for tailing ommensurate with probable life of mill. When necessary to reuse the iling goes to settling tanks. If there be insufficient fall for tailing to pass vity it is elevated by bucket elevators or sand wheels (Art 33). Storing for possible re-treatment should be considered.

STANFORD LIBRARIES

Percentage of extraction depends upon: easily separated from gangue give higher percer of ore. Percentage of extraction usually increa rich ore justifies higher treatment cost, and it to nearly the same value as that from a poor c profit is the most important factor. The limit o limit, and concentration should be elaborated a yield a profit over working cost. (*d*) Purity of concentrate, the greater the loss in tailing; the value of high-grade concentrate more than offset

Tendency for the future is toward graded graded washing. General rule is to save the co but to crush tailing considerably finer before c sizes, 1 to 2 in, diminishes slimes and cost of cru is important; they are difficult to treat, and ge of the losses. Graded crushing, sizing, and jigg their losses. Slimes, whenever formed, should b coarser material, and treated independently. F Wilfley tables and in fine jigging; it may be gr good classification. Middling in the coarser size to free included grains; in the smaller sizes, spe sary, as the valuable minerals are apt to be flat separate from the more compact gangue. The towards saving the finest and most difficult value after the coarser parts have been saved by ordi

Cost at which profit vanishes. It is important t furnished with tables, showing the fixed charges, for net smelter returns for the different metals, at differ charges, and the milling cost and cost of production f point at which a certain grade of ore will fail to return

Accounts and reports. Careful cost records sho subdivided, to show periodically (say once a month) venient comparison, accounts should be based on a 2 000 lb. These records may be of three kinds: (*a*) A item (labor, supplies, etc); (*b*) Distribution of expens of the work, to determine what proportion of the tota tribution of expense and income among the various certain lots are being treated at a loss, which cuts d costs may be divided under the four general headings costs, losses, amortization of plant. To the milling mining, smelting, refining, marketing, and general ex special features, calling for modifications of the cost s

37. Laboratory Testing

(See also Sec 31)

Testing may be: to determine a suitable proc termine the general efficiency of a concentrating

Testing small lots is done by certain sma similar to those of full-size machines. For ex makes a ROLL-JAW BREAKER, suitable for crus Allis-Chalmers Co makes CRUSHING ROLLS and F Thayer's small portable STAMP MILL is made by cisco. A nest of small circular hand SCREENS, (sieve ratio of $\sqrt{2}$), is invaluable for screen te

l, Ohio. For CLASSIFIER tests, the author uses a 12-spigot hindered lassifier, because of advantage of hindered settling over free settling. GING small lots, a hand jig with sieve box 1 ft square inside is most it. Round wooden screens are also suitable. Henry A. Vezin, Denver, ned a jig with sieve 3 by 4 in, which may be driven either by hand or . A small WILFLEY TABLE, with net working area 26 by 50 in, equipped regular Wilfley table mechanism, gives good results on small lots. anners and canvas tables can also be used for testing the concentrating of small lots of ores. The Cornish VANNING SHOVEL is used to sepa- y and light minerals, for products 3 or 4 mm diam down to slimes, st satisfactory on material finer than 1 mm coming from jigs, tables, ers. Enameled vanning plaques are very convenient.

ests for FLOTATION can be made by mixing the fine ore, oil, acid, and king in a closed jar, then allowing tailing to settle. The oil is poured ne added to dissolve the oil from the sulphides and the mixture heated , thus precipitating the sulphides. Or, the tailing can be dried and ecovery being computed by difference.

l 2-pole MAGNET may be made with sharp-edged pole pieces. This ed with weak current to test the applicability of the Ball-Norton con- and with strong current for the Wetherill.

amation tests for gold on small lots may be made in a "gold pan," the crushed ore with water, adding a few drops of mercury and thor- itating for a considerable time. A good amalgamating test for gold de on 1 kg of ore in a 2-gal bottle, revolved end over end with 50 grm amalgam. Most of the rusty gold will be caught. Pan amalgama- ilver may be done by charging the ore, crushed to pass a 40-mesh a small cast-iron amalgamating pan, supported on iron legs to per- cation of heat beneath, and provided with a grinder which should om 120 to 150 rev per min. The pan may be run with different reagents ring results.

g a given ore, analyses should be made of different representative samples, to composition and association of the different minerals. For instance, in ore silver, lead, pyrite, blende, and quartz, the percentage of silver carried by nt minerals should be determined. The mode of occurrence of the ore with , whether finely disseminated or in masses, is important in deciding amount of ecessary to free the mineral. Sizing and hand-picking tests are most valuable, r classifying, jigging, and table tests. Sizing tests and assays of the different ould always be made, to determine percentage of values saved and in what h ore will present different problems, and the ore dresser must use his judg- ciding what tests promise the most information. At present, many companies g distinct processes, for example, the Elmore Oil Process Co, will test a small lot heir particular process and report the results without charge, or for a small fee.

n regular mill work are made by determining, for each day, the weight e of the crude ore treated, together with weight and value of con- nd tailing. The work of different machines is tested by sizing tests, yses of the different products, and their weights.

38. Mill Flow Sheets

flow sheets which follow have been selected with the view of present- reatest variety of concentration and of method. In some instances, complexity of diagram, elevators, centrifugal pumps, conveyors, de- etc, have been omitted; for, while they are indispensable, they play n the actual separation of values from waste, though they often do disposal of both values and waste.

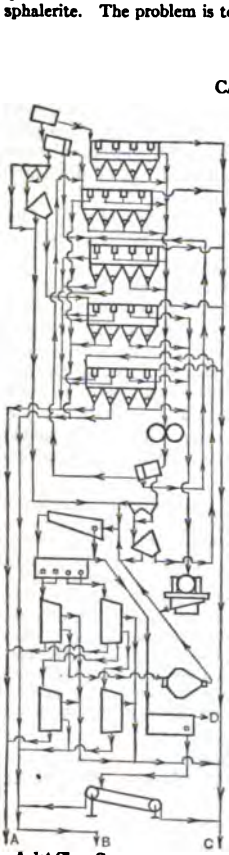
STANFORD LIBRARIES

1. Bunker Hill & Sullivan Minin
Kellogg, Idaho

The economic mineral is argentiferous galena, d
quartzite. The ore also contains unimportant
sphalerite. The problem is to save the galena.

WEST MILL

CAPACITY 550 TONS



A 1st Class Conc.
B 2nd " "
C Tailings
D Overflow

Fig 87

2 Trommels, 15-m
2 " 3 by
1 Dewaterer. 1 H
2 Bunker Hill scre
1 Harz jig
1 Harz jig
1 Harz jig
3 14 by 36-in rolls
2 Trommels, 3 by
1 Dewaterer
1 Esperanza classi
2 Bunker Hill scre
4 Hydraulic classif
1 6-ft Chili mill, w
7 Card tables; 9 C
1 6-ft Hardinge mi
2 Card tables; 1 C
28 Vanners

2. Federal Lead Co, Flat River, Mo (95)

mineral is galena, crude ore carrying 5.9% lead and 0.13 oz silver per ton. dolomite, calcite, and quartz.

CRUSHING PLANT 4 500 TONS PER 24 HR

CONCENTRATING PLANT 4 000 TONS PER 24 HR

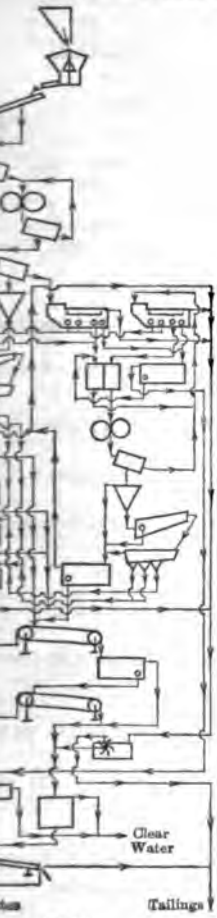


Fig 88

2 Ore bins

2 No 7½ Symons gyratory crushers, crushing to 3.5 in

2 Inclined stationary screens, 2-in round holes

4 48-in Symons disk crushers

4 Trommels, 10-mm round holes

4 Rolls, 24 by 48 in

8 Trommels, 10-mm round holes

4 Trommels, 1.25-mm round holes

2 Sloughing-off boxes

4 Standard 25-ft Hancock jigs

2 Esperanza-Federal classifiers

1 Double Esperanza-Federal dewaterer. 5 1-spigot, 60° spitzkasten

4 3-spigot Richards vortex classifiers

2 3-spigot 65° spitzkasten. 1 Roll, 16 by 36 in

2 Trommels, 1.5-mm round holes

8 No 5 Wilfley tables, 260 r p m. 1 Sloughing-off box

8 No 5 Wilfley tables, 265 r p m. 1 Esperanza-Federal classifier

2 3-spigot Richards vortex classifiers

8 No 5 Wilfley tables, 270 r p m. 1 1-spigot, 65° spitzkasten

6 Frue and Isbel vanners, 196 strokes per min

4 1-spigot, 60° spitzkasten

4 Johnston, 6-ft suspended vanners, 200 strokes per min

2 Single drag dewaterers. 1 Tailings settling box. 2 15-ft dewatering wheels

1 Concrete settling pond

18 20-ft diam by 20-ft annular overflow settling tanks

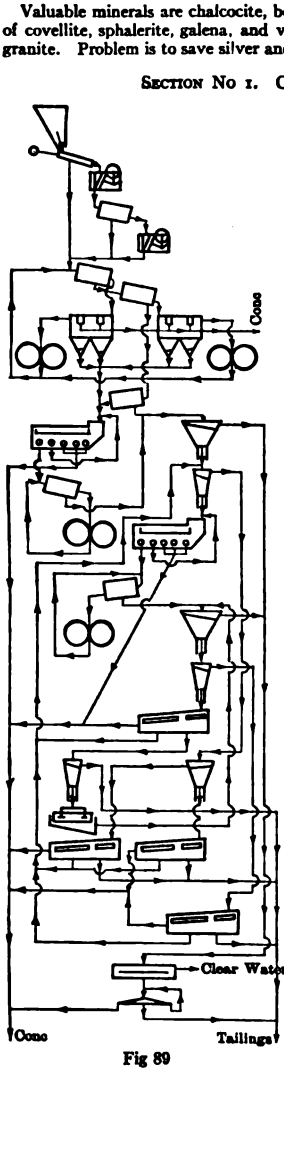
48 12 by 14-ft canvas tables

STANFORD LIBRARIES

3. Washoe Concentrator, Anac

Valuable minerals are chalcocite, bornite, enargite
of covellite, sphalerite, galena, and very little chalc
granite. Problem is to save silver and copper; no at

SECTION NO 1. CAPACITY 1 500



1 Ore bin
2 Shaking scre
1 Blake crushe
2 Trommels, 2
2 Blake crushe
4 Trommels, 1
4 Trommels 1
4 2-comp Har
2 Sets rolls, 24
8 Trommels, 4
2 Hancock jigs
6 Trommels, 4 classifiers
1 Set rolls, 16
8 Trommels, 1
2 Sets rolls, 16
6 No 4 Anacon
9 Wilfley table
2 No 5 Anacon ifiers
4 Huntington
19 Wilfley tab (coarse p
23 Wilfley tab
14 Dorr thicke
35 Round tab

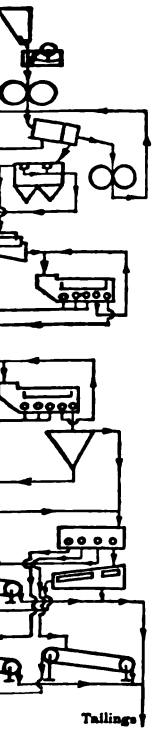
Fig 89

4. Arizona Copper Co, Clifton, Ariz (97)

nineral is chalcocite, with a little oxide and carbonate of copper, and con-
le gold and silver, in a silicious gangue. The ore averages 3.25% copper.
is to save the copper.

CONCENTRATOR NO 6

CAPACITY 700 TONS PER 24 HR



1 Bin

1 Blake breaker, 18 by 36 in, breaking to 3 in

1 Rolls, crushing to 1.25 in

2 Trommels; 2 sections, 0.625 and 1.25-in screen

1 5-comp Hancock jig. 1 2-comp Harz jig
1 Re-crushing rolls, 16 by 42 in, crushing to 0.75 in

3 6-ft Huntington mills, 5-mm screen

1 5-comp Hancock jig

2 6-ft Huntington mills, 1.5-mm screen

1 5-comp Hancock jig

1 Cone separator

2 6-ft Huntington mills, 1-mm screen

2 Hydraulic classifiers; 4 spigots, no overflow

11 6-ft Frue vanners, corrugated belts. 11 Wilfley tables

17 6-ft Frue vanners, plain belts. 17 6-ft Frue vanners, plain belts

Fig 90

STANFORD LIBRARIES

5. Detroit Copper Mining

Valuable minerals are sulphides of copper, chie
with 4% alumina. Concentrating ore carries 3
save the copper.

CAPACITY 1 100 TONS

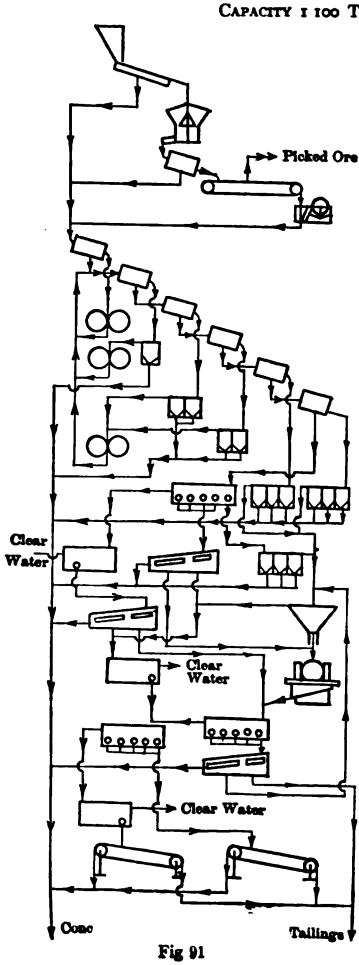


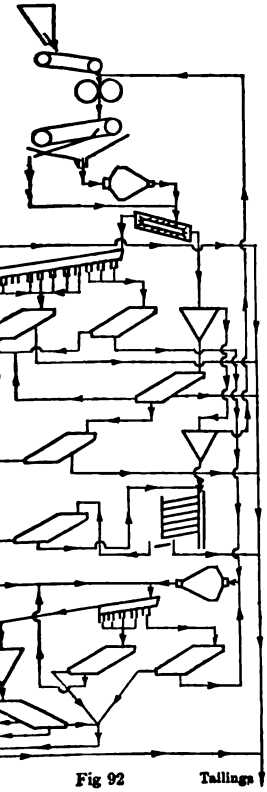
Fig 91

6. Miami Copper Co, Miami, Ariz (98)

minerals are chalcocite, with a little carbonate and silicate of copper. orphyry, more or less decomposed. Feed for one month carried 2.362% Cu, 05% was oxidized. Concentrate contained 42.22% Cu, 18.8% S, 16.5% SiO_2, , 12.5% Fe. Tailing for one month assayed 0.654% Cu and 0.223% copper een analysis of the tailing for December, 1913, was as follows:

Mesh	% wt	Assay	% Total Cu
28	2.1	0.84%	2.8
35	4.5	0.71	4.9
48	10.9	0.62	10.2
65	10.5	0.53	8.4
100	14.5	0.44	9.8
150	12.5	0.36	6.9
200	6.2	0.33	3.0
Below 200	38.8	0.91	54.0
	100.00		100.0

SECTION 6. — CAPACITY 600 TONS PER 24 HR



1 1 000-ton ore bin

1 14-in inclined conveyor

1 42 by 15-in Traylor spring rolls

2 Duplex Callow screens

3 8-ft by 22-in Hardinge mills

2 14-in drag belt dewaterers

2 11-spigot launder-type classifiers

16 No 2 Deister tables. 6 No 2 Deister tables. 2 8-ft cone tanks

2 No 2 Deister tables. 12 No 3 Deister slime tables

2 No 3 Deister slime tables. 16 8-ft cone tanks

6 No 3 Deister slime tables. 4 6-deck Deister tilting slimers

1 6-ft by 48-in Hardinge mill

1 5-spigot launder-type classifier

{ 4 8-ft cone tanks. 1 No 2 Deister table
{ 4 No 2 Deister tables

4 No 3 Deister slime tables

Fig 92 Tailings

STANFORD LIBRARIES

7. Copper Range Consolidated Co,

Amygdaloid rock contains quartz, calcite, epidote
(1.2%); problem is to save the native copper. Im
installed from time to time, based on Hodge or Wo
jig. Mortar discharge goes into ingots; bull jig and
table is the finishing machine, and Hardinge mill is
grains from the upper jigs, and fine included grains

HEADS 1, 2, 3, 4 TRIMOU
" 3, 4, 5, 6 CHAMPI
" 1, 2, 3, 4 BALTIC

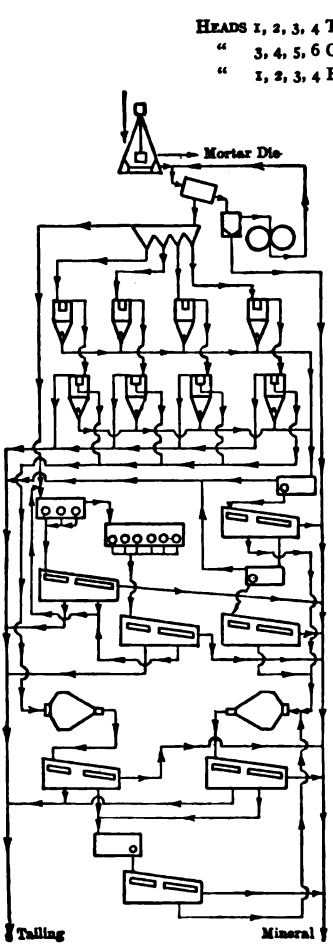
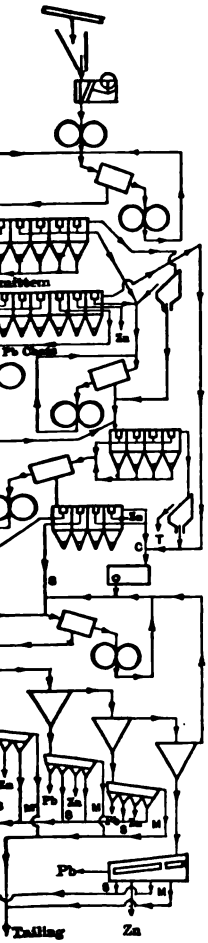


Fig 93

8. Lennan Zinc and Lead Co, Miami, Kan (100)

ch like typical Joplin ore; problem is to save galena and blende from chert angue minerals. Ordinary Joplin mill ignores too much the included grains ; this mill has been designed to save both, and has been very successful.

CAPACITY 350 TONS PER 24 HR



1 4-in grizzly

150-ton bin

1 15-in breaker

1 36-in dirt rolls

1 2-section 5/16-in trommel

2 30-in rolls

1 Rougher jig

1 Rogers dewatering screen, 1.5 mm

1 cleaner jig

1 19-in rolls. 1 3/16-in tromm

1 36-in rolls

1 Rougher jig

1 2.5-mm trommel

1 24-in rolls

1 Cleaner jig. 1 Dewatering screen

Sand tanks

1 1.5-mm screen

1 30-in rolls

2 V-box classifiers

2 V-box classifiers

2 Arbuthnots. 2 V-box classifiers

2 " 2 " " "

2 "

2 Wilfleys

Fig 94

= Mixed zinc and lead. *C* = Chats. *T* = Tailing. *M* = Middlings.

STANFORD LIBRARIES

9. Hell upon Earth, Jo

Valuable minerals are blende and galena in co
limestone and flint. Problem is to save zinc and l

CAPACITY 50 TONS P

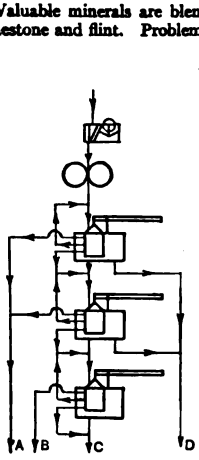


Fig 95

1 Blake breaker,
1 Rolls, 12 by 14
1 Roughing hand
1 Finishing hand
1 Lead hand jig

A = Waste. *B* = Blende conc. *C* = C

10. I Know Mining Co, J

Valuable minerals are blende and galena in co
limestone and flint. Problem is to save zinc and l

CAPACITY 100–120 TONS

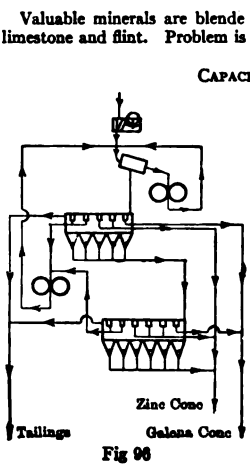


Fig 96

1 Blake breaker,
1 Trommel, 0.5-in
1 Roughing rolls,
1 5-sieve Harz jig
1 Rolls, 15 by 14
1 6-sieve Harz jig

11. New Jersey Zinc Co, Franklin Furnace, N J (101)

ists of zincite, willemite, franklinite, with other zinc-bearing minerals, and a alcite and biotite. Problem is to separate ore into four classes: (*a*) magnetic to oxide furnaces and residue to spiegel furnaces; (*b*) magnetic fowlerite, hroite, and other silicates of manganese, to zinc-oxide furnaces; residue is too smelt for manganese and is discarded; (*c*) jig and table concentrates, of zincite ite, shipped to company's works and to Europe for best spelter; (*d*) jig and gs, calcite and a little biotite, used for concrete, mine filling, or is discarded.

NO 2 MILL

ROUGH-CRUSHING DEPT, 150 TONS PER HR; FINE-CRUSHING DEPT, 100 TONS AGNETIC-SEPARATING DEPT, 50 TONS PER HR (R.i.s. = Rowand inclined screen)

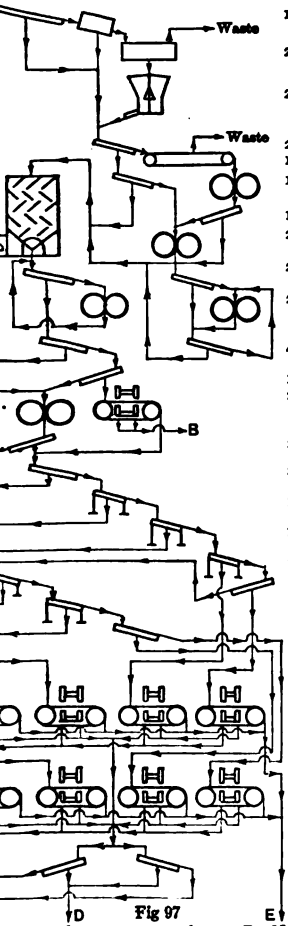


Fig 97

1 Grizzly, 5-in spaces. 1 Trommel, 1.5-in holes
2 Circular picking tables
2 McCully breakers, No 8 D, breaking to 5 in
2 Rowand incline screens, 2-in holes
1 Picking belt
1 R.i.s. 0.625 by 1.5-in openings. 1 Edison corrugated rolls, 36 by 36 in, to 2 in
1 R.i.s. 0.625 by 1.5-in openings
2 Rowand 6-ft brick tower driers. 1 Edison corrugated rolls, 36 by 36 in, to 1 in
2 Rowand incline screens, 0.109 by 0.5-in openings. R.i.s. 0.625 by 1.5-in op'ngs
2 Bacon finishing rolls, 30 by 36 in; to 0.0625 in. 1 Edison smooth rolls, 36 by 36 in, to 0.5 in
4 R.i.s. 0.01 by 0.5-in openings. 1 R.i.s. 0.625 by 1.5-in openings
1 R.i.s. 0.187 by 0.5-in openings
2 Re-grinding rolls, 10 by 18 in. 2 Rowand type "E–F" mag sep, one magnet with 100 000 ampere-turns, 125 volts
2 R.i.s. 0.032 by 0.5-in openings
2 R.i.s. 0.015 by 0.5-in openings
1 Edison tower screen, 0.015 by 0.5-in op'ngs
1 " " " 0.020 by 0.5-in "
1 " " " 0.025 by 0.5-in "
1 " " " 0.042 by 0.5-in "
1 R.i.s. 0.032 by 0.5-in openings
1 Edison tower screen, 0.058 by 0.5-in op'ngs
1 R.i.s. 0.078 by 0.5-in openings
16 6-pole Rowand-Wetherill type "E," 18-in belt, mag separators. Poles 1 and 2 have 20 000, 3 and 4 have 40 000 and 5 and 6 have 60 000 ampere-turns with 125-volt current
16 6-pole magnetic separators, details as above
2 R.i.s. 0.032 by 1.5-in openings

A = Shipping zinc product. *B* = Highly magnetic franklinite product. *C* = Low magnetic franklinite product. *D* = Slightly silicate of manganese product. *E* = Non-magnetic product to jig and table floor

STANFORD LIBRARIES

12. Tomboy Gold Mines, Tel

Ore is quartz, carrying gold, silver, and sulphides $10. Problem is to save gold, silver, galena, and py

CAPACITY 400 TONS PE

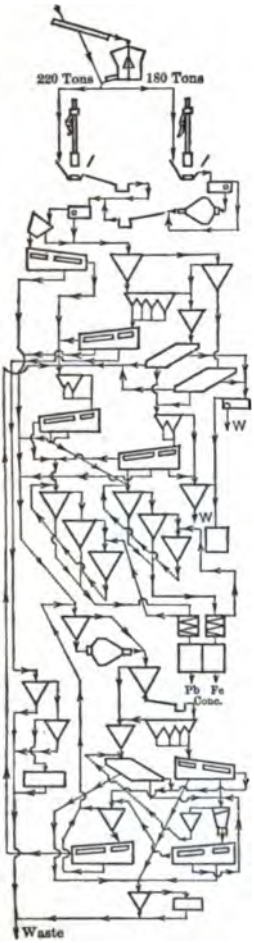


Fig 98

Grizzlies

1 No 4 Kennedy

40 1 050-lb star
3 mesh

8 Amalgam plat
ing box

2 Sloughing-off
plates. 2

4 Bunker Hill s

2 No 6 Wilfley

4 Callow tanks

2 Richards h. s.

15 Callow tanks

19 No 5 Wilfley

10 No 3 Deister

1 Richards h. s.

Sand box

3 No 6 Wilfley

2 No 6 Wilfley

1 4-ft cone. 2

1 8-ft cone. 2

1 8-ft cone. 4

2 8-ft cones

2 4-ft cones. 3

2 6-ft Hardinge

1 4-ft cone

1 Caldecott con

1 Settling cone
classifier

7 No 3 Deister

2 10 by 32-ft D

1 4-ft cone. 3

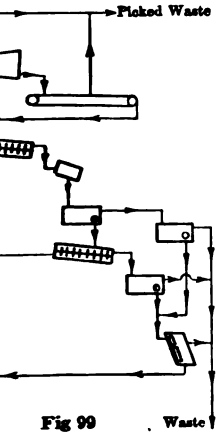
1 No 5 Wilfley

1 Thickening c

13. Oliver Iron Mining Co, Mesabi Range, Minn (104)

mineral is hematite and the gangue is a fine silicious sand. Problem is to e sand and save the hematite.

CAPACITY ONE UNIT 100 TONS PER HR



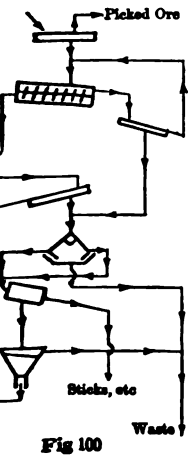
1 Ore bin
1 Grizzly, 7-in spaces
1 20-ft conical trommel, 2-in holes
1 3-ft picking belt
2 25-ft log washers
2 Chip trommels
2 Settling tanks
2 Settling tanks
2 18-ft turbo washers
2 Settling tanks
20 Overstrom tables

Fig 99

14. Longdale Iron Co, Longdale, Va (101)

ists of limonite in a gangue of clay, shale, sandstone, and quartz pebbles. to save the limonite.

CAPACITY 20 TONS PER HR



1 Flat grizzly, 2-in spaces
4 Log washers
2 Stationary inclined screens, 0.187-in round holes
4 Trommels, 0.187-in round holes
4 Stationary inclined screens, 14 mesh
1 Johnson mechanical sand shoveler
1 Trommel, 4-mesh
1 Richards annular classifier

Fig 100

STANFORD LIBRARIES

15. North Star Mines, Grass

Ore contains fine free gold and auriferous pyrite d
is diabase. Problem is to save gold and pyrite.
and tailings cyanided.

CAPACITY 135 TONS PI

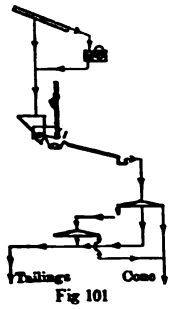


Fig 101

2 Grizzlies,
2 Blake bre
8 Challenge
8 Batteries
8 Silvered-c
8 Mercury
8 Dodd bu
1 Dodd bu

16. Port Henry Magnetic Mill, P

Valuable minerals are magnetite and apatite; gang

CAPACITY 700 TONS PI

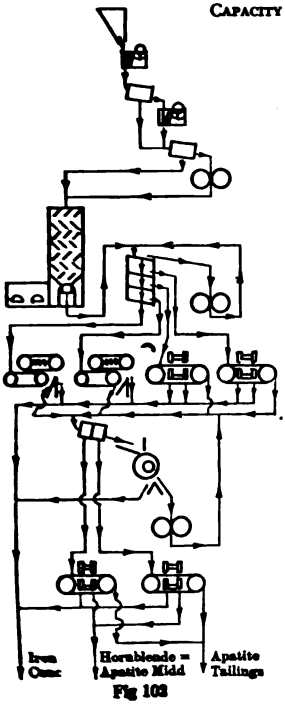


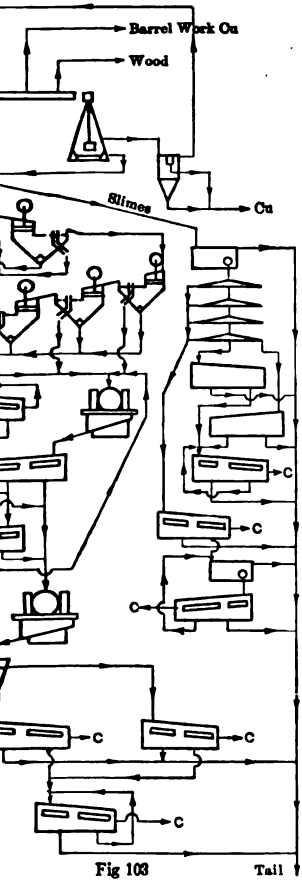
Fig 102

1 Ore bin
1 Blake cru
1 Trommel
1 Blake cru
1 Trommel
2 Rolls, 36
1 Tower dr
1 Tower sc
2 Rolls, 36
2 Ball-Nor
2 Rowand
1 Trommel
1 Wenstrom
1 Rolls, 36
2 Wetherill

17. Calumet and Hecla Copper Mining Co, Mich

is 300 tons per day on conglomerate rock and nearly 600 tons per day on Rock contains native copper and a little native silver. Conglomerate le calcite, epidote, and martite; amygdaloid carries a little calcite and epidote. to save copper and silver. Of the two general systems at Lake Superior, on the Hodge jig, the other on the Woodbury jig; this mill uses the Wood- hili mills are employed for recrushing both coarse included grains from jigs and l grains from Wilfley tables. Woodbury mortar jig takes nuggets of copper rtar.

ONE MILL UNIT AND REGRINDING PLANT FOR CONGLOMERATE



Mine bin

Picking table

1 Steam stamp, capac 335 tons per 24 hr, with mortar jig discharge to cover jig, and screen discharge to Woodbury jig

1 cover jig

2 Woodbury classifier jigs; first sieves 24 by 36 in, second 24 by 30 in

1 Settling tank

3 Woodbury jigs; all sieves 48 by 30 in

2 Round tables, 4 decks

4 Calumet tables

2 Wilfley tables. 1 Chilian mill

4 Calumet tables

1 Wilfley table. 1 Wilfley table

4 Wilfley tables

1 Wilfley table

1 Settling tank

48 Chilian mills. 1 Wilfley table

24 Classifiers

72 Wilfley tables. 48 Wilfley tables

16 Wilfley tables

ning of recrushing plant, which receives products from 11 mill units. Capac er 24 hr. Both Chili and Hardinge mills are used.

STANFORD LIBRARIES

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SECTION

ORE SAM

BY

T. R. WOODE

CONSULTING METALLURGICAL CHEMI

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1.	Conditions Controlling Ore-sampling Practice	1730
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Note. — Numbers in parentheses refer to Bi

Definition. The correct sampling of a lot of a smaller quantity which contains unchanged per lot. The commercial object is accomplished w conditions within an allowable limit of error, ha and cost, and is of the bulk, dryness and finen determination of one or more of its constituent be exact. As there are limits to the accuracy of ents it is necessary only that the error in sampling

1. Conditions Controlling O

Segregation of minerals. Constituent uniformly, but are more or less segregated. silver ores. At the mine most sulphides a than the gangue, and every subsequent h screening) increases segregation of the fin to be sampled is therefore an imperfect mi from 200 mesh or finer to pieces of several mixture of different-sized particles is impos merely decreases the degree of segregation; ods, and the inclined chute, barrel-mixer an gation. Accuracy of a sampling process de a lot of ore the same proportion of all its dif that selects the finer or the coarser particle older sampling systems depends not on in

of distribution of different-sized particles around the axis of a trunne. Later systems use mechanical means for taking frequent small from a stream of ore; this is called FRACTIONAL-SAMPLING, since the ple is composed of many partially mixed portions, any one of which can usly affect the value of the sample, but which ate approach the true average and are within ing error.

e of crushing is sufficient when the size of st particle does not exceed that permissible eight of sample obtained. Insufficient crushcially in the later steps of the process, probses most of the persistent unintentional sampling. Table 1 shows relative sizes hts adopted in the best sampling plants. athematical derivation see Bib (2).

ogy of sampling relates to the personal equation pler, who, to protect his employer, may overlook ght causes of salting (Sec 30, Art 9), permits adbe taken of segregation, or jockies with assays. nsible for use of devices similar to knee-joint and utes (Art 5), under conditions causing irregularitaccuracies evident only to the expert. Improved d customs lessen its influence, but its existence cognized in any discussion of sampling.

Table 1. Smallest Permissible Wt of Sample for Different Sizes of Crushing for Gold Ores

Size, diam, in	Lb
2	10 000
1½	5 000
1	2 000
¾	1 000
½	400
⅜	300
¼	200
3/16	100
⅛	75
6 mesh	50
10 "	25
18 "	10
30 "	4
50 "	1

2. Remarks on Sampling Methods

l. Ore is sampled by hand or mechanical methods. HAND METHODS, and simplest, require merely shovel and wheelbarrow. They are to all ores; and are preferable, usually necessary, with wet or sticky MECHANICAL METHODS, stationary or moving devices in the stream of atically divert the desired proportion for the sample. Though first cost enance of a mechanical plant are important items, its operation may be s and the preliminary sample obtained a few minutes after unloading; e ore is less important and probably unnecessary; and, when correctly d and operated, liability to error is much less than with hand methods. a mechanical plant may be so constructed that a correct sample is , thorough inspection always discloses any such condition.

nical devices are of 2 kinds; stationary, which continuously divert cerons of the stream for the sample; moving, which periodically divert the am for the sample. They respectively take part of the stream all the all the stream part of the time. Theoretically and actually, stationary dependable as moving devices. Ore does not flow uniformly, but is of many streams of varying proportions of fine and coarse. Thus, corll shells permit passage of coarse particles at certain points, and in an r angled chute, or barrel-mixer, the finer will seek the lower points. one or more compartments of a stationary device may steadily receive proportion of coarse or fine. With properly constructed moving devices, ross the entire stream, the manner of delivery is unimportant, even if s previously sized, because the sample spout is under all sections of for equal periods of revolution or oscillation.

nary and final sample. Whether sampling is continuous, from ing to final grinding, or is a series of unrelated processes, the operataining the preliminary sample from wet original ore is distinct from

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side, while the finer drops vertically. A small accumulated error after passing 3 riffles. This of the riffles across the stream; but to be effe mechanically and not left to the workman's wh small divisions to clog, the device is applicab

5. Preliminary Sample: Movin

These are so constructed and operated th of rotation or oscillation a division called t stream for the sample.

Snyder sampler. This type (Fig 3) cons ing in a vertical plane on axis *B*, with an inclin it. As the spout passes the sample such proportio circle of revolution. At and is thrown back into

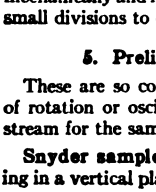


Fig 3. Snyder Sampler

These machines being of c parts, and are accessible at samples per min are taken, ting with 2 or more spouts. angles to the plate and in plan Thus, as the edges wear, the usual inclined delivery chute As the device revolves in a plate for half a revolution or from spout to the reject.

Vezin sampler is used quite generally and ods of sampling. There are many varieties Fig 4. Vertical shaft *C*, revolving with the rev per min, carries sampler spouts *AA* under *B*, and diverts sample to center and through s tom. During remainder of revolution the o reject receptacle.

ADVANTAGES: can be made accessible for cleaning during sampling; frequency or proportion of sample number or angular width of spouts; 2 intermeshin be operated for taking duplicate samples. DISA quires much head room; damp ore collects in or ma spout; the long upper edges of spouts quickly becon and worn. It is not patented and there are no restri facture or use, though the inventor, H. A. Vezin, s for a correct sample. He insisted that sides of san

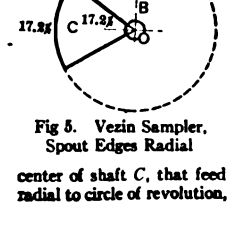


Fig 5. Vezin Sampler, Spout Edges Radial

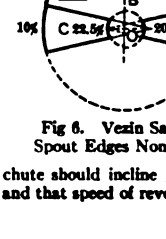


Fig 6. Vezin Sa Spout Edges Non

center of shaft *C*, that feed chute should incline radial to circle of revolution, and that speed of rev

dges, to lessen liability of particles to jump from or into sample spout. CHIEF ERROR is the position of sample-spout cutting edges, due to faulty construction, of non-radial sides. A variation from radial positions, imperceptible with machine in motion and proved only by measurements, may cause appreciable error. This is shown in Fig 5, 6, and 7, which are horiz views of Vezin samplers in actual use. Fig 8 is from a photograph of a machine just before installation. If segregating delivery chute is added, the error may be anything desired, as shown in Fig 9 (7). In the cuts: *B* is inside line of spout at center of carrying shaft; *C*, top of sample spout; *O*, outside edge of inclined chute delivering ore to sampler; and the proportion taken for sample at points indicated. Fig 5 shows all cutting edges of spout radial, with equal arcs cut on *A* and *B*, and is the only correct sample regardless of segregation in the ore stream.

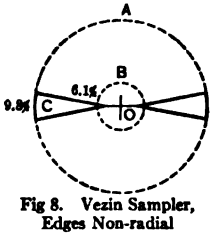
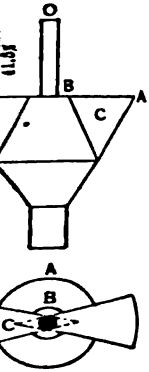


Fig 8. Vezin Sampler, Edges Non-radial

Angle between spout edges is too acute in Fig 6, 9, and too obtuse in Fig 5. Percentages (%) show proportion taken for sample at inside and outside limits of sample spout, and also at extreme points on delivery chute. Even such machines might not cause commercial errors except for use of the inclined, and more especially the knee-joint chute (Fig 9). By adjusting the angle between *D* and *D'*, and varying their lengths, control of distribution of coarse and fine is possible, as proved by samples taken across the chute. The chute shown in Fig 9 would probably cause a low sample in Fig 6, 9, and a high sample in Fig 7, 8.



e-joint Chute, with l Vezin Sampler

There are many modifications of spout and chute that will cause this otherwise excellent machine to deliver an inaccurate sample. Previous to year 1900, machines were found with sample spouts intentionally leaky, causing loss of fines; delivery chutes dropped fines just beyond outer circumference of spouts; and sample spouts were made with parallel instead of radial edges, with ore feeding from outer circumference toward the center, and producing samples of 60% or less of the actual gold value. Since 1900, similar but more refined practices have been devised, so that it is wise always to inspect carefully every installation. The machine is generally conceded to work accurately if the spout edges are constantly radial and feeding chute is vertical and at proper distance above the spout.

Chas. Snyder sampler is in principle similar to the Vezin, in that it revolves in a horiz plane and has h radial edges (see Fig 10, lettered like Fig 6). It has a special annular, livery chute, covering an arc of 90° directly over the sample spout, and narrower at top where it receives ore from ele- ls, or shaking trays. Within this spout are a short horiz iron rods, to scatter and delay the and destroy any previous segregation. The as 4 sample spouts, assuring continuous flow em, instead of the intermittent flow of other Errors of construction noted under Vezin sam- , but in less degree because of narrowness of uts.

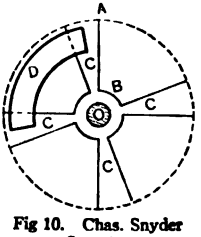


Fig 10. Chas. Snyder Sampler

vibrating sampler. Fig 11 shows front and Ore is delivered through chute *A*, narrowing d *B*. Operating mechanism drives diverting n positions *C* to *C'* and back again, diverting ore to the reject and es *R* and *S*, respectively. Usual sample taken is 20%, but a simp permits any desired percentage. Correct feed is necessary for

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accurate sample, and as insisted upon by
assured by the narrowing chute or by use o

Possible inaccuracy through cutting a large st
indicated by Fig 12, representing an actual case.
than at C', and takes an appreciable time to cha
more ore near C' than near C. Ore from rolls A,
delivered over full area of chute D, the coarse sc
slides to opposite side. Therefore sample S mig
but too much fine, and would probably be high i
side of chute, or a different length of portion D, n

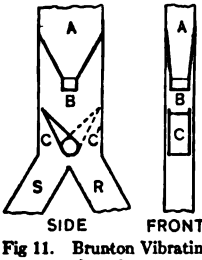

Fig 11. Brunton Vibrating Sampler

Fig 12. Vibrati
pler (Incorrect

Brunton oscillating sampler (Fig 13).
the moving portion suspended from shaft C, a
Ore is diverted for the sample through spout
sitely inclined spouts on both sides of S.

This machine is free from the undesirable featu
part of sample spout is rectangular, making it
cutting edges are removable and quickly replaced;
at speeds (regulated by size of largest particles)
min, with the greater accuracy due to small fr
motion tends to free the spouts from wet ore or
including several machines with intervening rolls
Usual sample is approx 20%, though 5% may be

6. Final Sa

When diam of largest particles is 0.125 to
the sample is taken to the bucking room
preparation for assay. (Sec 30, Art 1.)

Further reduction is made by any hand
or by riffles. BRUNTON SHOVEL rapidly and

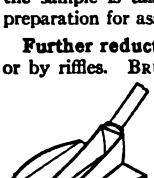

Fig 14. Brunton Shovel

the quantity desired. Fig
with 1 sample and 2 reject
large samples of medium-g
since, when shoveling from
select fines. The 7-compa
4 reject divisions, may be
samples, and for original a
preferable on small quanti
done on the flat, Jones, or
The Jones (Sec 30, Art 1)
angle of 45° to the horiz. Taylor and Brunto
Jones, with short stiff horiz dividing edges,

nd to avoid the bending, leaking, and clogging of other forms. Riffling od results only if the device is kept in repair, the ore moved across the uring feeding, and partitions do not overflow.

ng. When the sample weighs from 2 to
ut preferably not less than 20, it is dried
n or electricity.

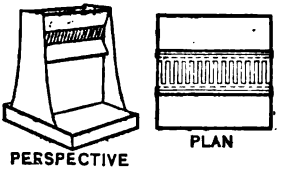

Fig 15. Taylor and Brunton Riffle

plates are steel boxes through which steam s, having a top drying area of 6 to 20 sq ft, h the sample is spread. Steam coils are in a brick or steel closet, the pipes forming or holding pans containing sample. Electric nsist of resistance coils in a closet containing or holding sample in pans. Temperature is nt and should be slightly above the boiling point corresponding to elevation above . Thermometers are used for steam, and automatic regulating devices for electric Insufficient or excessive drying causes trouble and error in assaying and valuation e (Art 7).

ding of the dried sample to 40 to 60 mesh is done by rolls, or grinders Englebach cone- or Braun disk-type. The ENGLEBACH TYPE consists of shaped cone, revolving horizontally within and beneath a stationary g flaring at top and bottom. The cone rests on a vertical driving shaft, e ring is carried on a hinged frame so that parts are quickly swung open ning. 8 or more channels are cut vertically in the grinding surfaces of one and ring. Ore is fed through top of ring and channels. Fineness of g is controlled by regulating closeness of contact between cone and ring by sting screw and lever attached to driving shaft. BRAUN DISK-TYPE consists cular iron plate, revolving in a vertical plane with its face in contact with e of a similar stationary plate, hinged at the bottom to permit separating aning grinding surfaces. 6 or more feeding channels are cut in faces of lates, beginning about 1 in from circumference and deepening at center. fed into a hopper passing through center of the stationary plate. An ng screw on shaft carrying the revolving plate regulates pressure between tes and consequent fineness of grinding.

lity of iron used in grinders is important. If hard it soon polishes and fficiency; if soft it may lose enough iron to the sample to lower the value iably, or may even take up particles of valuable metals from one sample op them into a succeeding sample. Imperfect parts must be promptly led; filling cavities with babbitt is not permissible, as metals are removed he sample by the soft filling. The ground sample may be further reduced, ed, by methods described above.

ening is done through sieves varying from 80 mesh for low-grade to sh for highest-grade gold ores, average being 100 and 120 mesh (Sec 30).

s should be free from missed wires and holes, and so constructed that all solder- on the outside. Sieves with broken or misplaced wires should be promptly dis- and not repaired with a drop of soft solder which may gather metals from the While it is not desirable, it is accepted custom to use loose iron washers in the prevent balling of dried sample and to hasten screening. Some plants even use tion a stiff paint brush. If hastening is overdone, especially near end of process, res of copper, gold, or silver may pass through and cause puzzling differences in g. Single wires of gold have been found heavy enough to cause 0.5 oz gold ce in check assays (7). With such ores, washers should be discarded and more ken to grind the pieces of metal. Oversize from the sieve is either reground or l on the bucking board (Sec 30, Art 1). Quality of iron in bucking board and r is important (see above).

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Metallic particles resisting grinding, and ore (or copper caps or iron from the machine assayed (Sec 30, Art 3).

Mixing the ground sample, now called PU ing into a rapidly revolving gold pan, by turni ing through an Anaconda Mixer, or by rolling paper or oil cloth (Sec 25, Art 4). Dividing by coning and quartering, by the spatula, by or preferably by riffling (Sec 30, Art 1, 3). P are used for holding this assay pulp.

7. Moisture San

Moisture samples. Ordinarily 2 to 4 sam from each carload, while high-grade lots ma may gain or lose water rapidly, moisture samp to the time of weighing the lot. In case of gr obtain proper proportions of all sizes, as fines u than coarse pieces.

As individual grab moisture samples may vary a an average. To avoid this difficulty samples may b chanical device, during or after the sampling and w smaller. As ore becomes drier during sampling, it i 10% to the moisture as determined from the crush mospheric humidity. While all modes of determini mations, the sample taken after crushing is preferab

Weighing the wet sample requires care a use of either the finest or the coarsest materia ing should be avoided. Special scales, for show percentage loss without calculation. Sa drier and at same temperature used for assay differences in moisture or in roasting of oxidiza can not be accurately computed.

8. Multi-samp

Check sampling. Custom plants usually settlement is agreed upon. Owing to limited bed or store the bulk of a lot, retaining intact and usually only a few hundred pounds. Lo determine the amount so held, though the Sta ute for retention of 2.5%. It is the practice SAMPLES, called ORIGINAL and DUPLICATE, or ev the division being made when sample has been exposes gross errors in later work, detects salti resampling, and averages errors of faulty meth

At some plants anything less than a specified p of multi-samples makes settlement on the average co automatically calls for resample of the retained p 5 to 20%, usually 10%. Close checking of multi-s in the portions so handled, but does not prove corre On the other hand, large or constant differences indic methods, and make possible undesirable "jockying" 11). Many plants are so constructed and operated pling of any sized lot may be less than 2%, with on

9. Synchronism

ition. When two or more mechanically-operated sampling machines nged in series, there is a constantly recurring cycle in their relative . If one horizontally-revolving machine is directly beneath another ne rotative speed, the spouts will have a constant angular difference, n no intervention the second sample spout may be arranged to receive rt or none of the first sample. At different speeds one spout will pass r at regular intervals and, if not obstructed, the sample from the first will regularly strike certain points on the second machine in any

Table 2. Example of Synchronism

Record representing 0.2 of complete record												
.....	5	10	1	6	1	4	1	1	4	3	6	1
.....	5	1	5	1	5	1	1	7	1	5	1	5
.....	8	1	4	1	6	1	7	1	4	1	8	1
.....	1	5	1	3	1	6	1	7	1	5	1	5

esired. Table 2, containing data from a large plant, illustrates this. ow figures S show number of consecutive samples from first machine partly or entirely into second sample spout. Lower row figures R mber of consecutive samples passing entirely into the reject of second . These should be read, S 5, R 5, S 10, R 1, S 1, R 5, and so on.

alue of any sample portion depends on at least part of it reaching the ultimate its complete loss being equivalent to omitting from the shipment the portion which it represents. Intervening rolls alone do not correct synchronism, but , the continuous delivery of the Chas. Snyder sampler, closed hoppers, barrel- nd shaking trays, lessen or prevent it. The BARREL-MIXER has the disadvan- egregating coarse material and corrugating the following rolls. The SHAKING ses ideal delivery, besides being a generally useful device. It delays the ore ly to prevent synchronism, gives uniform feed, can be made to correct uneven olls and to stop pieces of steel or hammer heads that might injure the machinery.

10. General Considerations

ghing. Small lots, or high-grade ores, are weighed on special bullion Carload lots are weighed on railroad scales, preferably housed and with etween platform and coping covered by old belting to prevent jambing es of ore. Railroad scales should be tested monthly by special test approx weight of the empty and loaded car, and daily or weekly by 40 ghts totaling 2 000 lb. Great care is necessary in checking with small ghts, as the 20-lb minimum error of the scale is equivalent to 2 000 lb a load of 100 000 lb.

hould be uncoupled and stationary when weighed, and time allowed for scale swing both up and down. Rapid balancing with rider slightly over and under- or gross and tare will cause 100 lb or more error in net weight. Balancing scales g fine ore or shot on rider may cause an error of several thousand lb. Wagon o short for both team and wagon, must have roadway at each end level with , and brakes and tugs must be loose during weighing.

mplete lots. Portions of a lot may be received at widely different necessitating the questionable practice of taking several preliminary to be combined later for the final sample. With mechanical methods to error is not serious, but with hand methods different sets of work- king slightly different proportions may cause errors never discover- o bedding of the bulk of the lot.

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A combination may be made of SAMPLING WITH ROA No 3), though this is of doubtful desirability. There excessive dust, and general lack of care when sampling

Cleaning. Plants should be reasonably tight ore chutes tightly enclosed to prevent the too In hand methods, cleaning with brush and broom mechanical mills, cleaning immediately follows proceed from first to last crushing and sampling through all following machinery to insure a pro

Many mills supplement brush and broom clean piped to convenient points. After the regular cleani are fed with barren rock or slag, and again cleaned It is good and economical practice to discard cones, diately after grinding a very high-grade ore.

Salting. Deliberate salting by addition of to salting at the mine (Sec 25, Art 9). Device have been quite elaborate, including high-gra necessary to cause the desired salting, and pro cies or for following a lot from sampler to sm for nearly a year, was largely responsible for known smelting company. Of greater impo machinery, producing either a high or low sam may employ trained men, called MOOCHERS, t sampling company may lock sample-hoppers important machines. The most effective pre chanical mill, operated according to scientific

Confusion of samples. This is a worryin many large and small lots, of unknown and they are begun and finished by different workm

Accompanying is a record made at the Taylo plant, Murray, Utah, on a form designed primarily strong cardboard, punched for loose-leaf practice, reports from each department. Upper margins wer reference. MILL CARD, signed by foreman Quist, sh the mill in 2.5 hr, at cost per ton (wages at 25¢ per h reloading into 2 other cars. Had the ore arrived i halved. Cleaning took 0.5 hr, increasing cost 20%. livered a theoretical wt of 183.2 lb for the prelimi that Davies (ass't sampler) began cutting (using B of ore was unloaded, and finished 5 min after clean tially-dried sample of 22 lb, instead of the theoretica from the reject at this point, averaged 4.2%. SAM man, Spoonley, and prompt placing in steam drier, It was then ground in No 1 grinder, riffled 3 time Finished pulp weighed 42 oz instead of the theoretica barren sand before and after this lot, eliminating da Wyoming lot 195. Cost of sampling an ore receive varied physical conditions, is instantly available fro efficiency of foremen and workmen. Its use genera

Manner of sale of ore. Local custom the procedure. In outlying camps a lot may made at the mine by buyer's agent. But ore or smelter, is transit-sampled on its way to sm Valuation may be based on sample and assays made by buyer and seller on the custom sam

Sampling Mill Cards (devised by T. R. Woodbridge)

Nov. 8. 1907 *O MILL*

d Lot Number *Grand Central — 55* DAY ~~NIGHT~~ Shift

15367, 28118 OUT Cars *17171, 27075*

of IN Cars Box Dump [Wooden Gonds] *2* Steel Gonds

Sacked [Coarse] Fine Concentrates Wet [Dry] Frozen

utters Used *4*

of UNLOADERS *6* LOADERS *1*

loading Started *7 : 00 A.M.* Finished
mpling Started Finished *9 : 30 A.M.*

n Lot *146560* Next Lot *Wyoming, 195* Started *10 : 00 A.M.*

re this *6262* Signed *Quist*

.....

O CUTTING ROOM

d Lot Number *Grand Central — 55* DAY ~~NIGHT~~ Shift

ced Cutting *9 : 35 A.M.* Finished Cutting *10 : 05 A.M.*

Cuts before Grinding *3* After Grinding *0*

Taken to Sample Room *22 lbs*

Moistures taken by *Davies (4.1, 4.3 = 4.2%)*

e this *6262* Next Lot *Wyoming 195*

Signed *Davies*

.....

O SAMPLE ROOM

Lot Number *Grand Central — 55* DAY ~~NIGHT~~ Shift

11. 8. 1907 Time *10 : 10 A.M.* By *Spoonley*

yer *1* Time *10 : 20 A.M.* By *Davies*

inder No. *1* Time *3 : 00 P.M.* By *Spoonley*

1. 8. 1907 Time *4 : 40 P.M.* By *Spoonley*

of Pans in Dryer *3* Number of Times Cut before Bucking *3*

of Sacks put up *6* Weight of Pulp Bucked *42 oz*

e this *Sand* Next Lot *Sand*

ies Signed *Spoonley*

on Reverse Side any Delay, Cause, Presence of Ore Rep, etc

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In last case, seller may use the custom sample
wisdom of calling for a resample by the smelt
advantage, some buyers request or demand a
even considering seller's objections.

Resamples may be demanded by either buy
resamples of entire lots are freely granted, usua
charge if the 2 samples check within an agreed
on low-grade to 2% on high-grade ores. Other
plants, have various rules depending on exigenc
fidence existing in methods employed. A comm
on unchecked ore, but only in case of 10 to
check-sampled. Several resamples may be de
by a specified difference. To be of value re
whole lot, rather than of the small portion usu

Most custom plants and some smelters make pay
buyer and seller on same sample. Others pay on b
without knowledge of sellers figures, permitting the se
assay, or a resample, with correction in payment if t
ciently large. As the seller is unlikely to make dem
the buyer, under this arbitrary system, takes a certai
ing duplicate assays made by the same or a different
average, as his judgment and conscience may direct
ment assay in cases of repeats, umpires, and resample
assay may be selected, or the whole series averaged.

Table 3. Comparison of Gold Assay
(*O* and *D* are original and duplicate sa

Seller		Buyer		S
O	*D*	*O*	*D*	*O*
0.98	0.92	0.87	0.86	2.65
1.67	1.87	1.66	1.78	2.16
1.27	1.11	1.06	1.24	2.32
1.51	1.51	1.46	1.46	2.71
1.22	1.21	1.18	1.18	2.40
1.58	1.58	1.52	1.54	3.28
1.24	1.24	1.19	1.19	4.48
2.70	2.78	2.65	2.64	5.80
Average of entire series				2.37

Table 4. Comparison of Gold Assa
(Brackets indicate resamples; *O* and *D* are original a

Seller		Buyer		Seller		Buy
O	*D*	*O*	*D*	*O*	*D*	*O*
1.26	1.24	1.26	1.16	1.14	1.10	1.16
1.54	1.74	1.52	1.66	1.59	1.63	1.72
1.22	1.11	1.16	1.06	1.02	1.10	1.00
1.25	1.28	1.18	1.25	[1.61	1.75	1.58
1.50	1.48	1.38	1.38	[1.82	1.74	1.64
1.41	1.43	1.40	1.38	1.35	1.39	1.28
1.23	1.17	1.18	1.08	1.04	1.00	1.02
0.92	0.98	0.88	0.94	3.00	2.90	2.92
1.03	1.09	0.99	1.02	2.62	2.68	2.56
1.37	1.36	1.30	1.33	2.88	2.92	2.79
Average of entire series.....						

11. Comparison of Assays

arison of assays of large shipments and over long periods of time are where special assays or selected resamplings are indecisive. Tables 5 contain assays representative of the entire shipments from 1 mine pling plants for approx 1 year, in lots of 30 to 150 tons. The following rules for settlement existed, but were not rigidly enforced. Pulps yed by both seller and buyer, and in some cases by an umpire. Set- was made on buyer's assays, when results were within limits varying oz gold to 10% of buyer's assay. For greater differences, even after or umpire assay, a resample was made of the retained portions (Art 8). and duplicates were made in all plants. The tables show that seller's e generally higher than buyer's (Art 9).

Table 5. Comparison of Gold Assays on Buyer's Sample

indicate resamples; *O* and *D* are original and duplicate samples; values in oz)

eller		Buyer		Seller		Buyer	
	D	*O*	*D*	*O*	*D*	*O*	*D*
	1.26	0.90	1.26	2.44	2.44	2.46	2.32
	1.14	1.04	1.14	2.28	2.60	2.20	2.62
	1.18	1.10	1.12	2.39	2.40	2.40	2.46
	1.02	1.16	0.96	2.80	2.72	2.64	2.64
	0 74	0.84	0.68	3.02	2.58	3.04	2.60
	1.50	1.32	1.50	2.80	2.18	2.76	2.24
	1.58	1.28	1.60	3.33	2.71	3.32	2.76
	1.27	1.08	1.28	3.52	3.22	3.47	3.18
	1.33	1.38	1.40	3.11	3.13	3.04	3.04
	1.25	1.20	1.20	3.08	3.34	3.04	3.28
	1.33	1.26	1.24	2.79	3.03	2.84	2.88
	1.58	1.54	1.40	5.34	5.40	5.26	5.34
ge of entire series				2.04	2.05	2.01	2.00

shows assays of shipments from 1 seller to 1 buyer for 15 months, and illustrates pling and assaying. In Tables 3, 4, 6, resamples were made only on the re- rtion of 200 to 500 lb, and were not a check on the whole lot (Art 8). Many of ling machines figured in preceding pages are from plants represented in the Table 7 shows assays of shipments from a custom sampler to a smelter using hand Averages in Tables 3, 4, 5, are of much longer series, shown here only in part.

Table 6. Comparison of Gold Assays on Buyer's Sample

(Representative of 400 assays)

indicate resamples; *O* and *D* are original and duplicate samples; values in oz)

r	Buyer		Seller		Buyer		Seller		Buyer	
D	*O*	*D*	*O*	*D*	*O*	*D*	*O*	*D*	*O*	*D*
0 49	0.47	0.47	1.27	1.28	1.33	1.32	2.06	2.07	2.03	2.04
0 67	0.64	0.67	1.03	1.06	0.98	0.98	2.17	2.24	2.16	2.18
0 74	0.80	0.80	1.15	1.16	1.11	1.11	2.62	2.61	2.64	2.64
0 77	0 75	0.76	1.00	1.00	1.01	1.04	3.48	3.42	3.41	3.44
0 63	0.62	0.62	1.25	1.20	1.28	1.26	3.57	3.65	3.55	3.68
0 49	0.52	0.52	1.22	1.21	1.11	1.13	3.60	3 62	3.44	3.42
0 46	0.47	0.47	1.06	1.05	1.00	0.97	3.33	3.31	3.34	3.30
1 06	0.98	1.02	1.00	1.02	1.00	1.00	3.10	3.02	2.89	2.91
1 66	1.56	1.56	1.17	1.16	1.16	1.16	3.05	2.95	2.93	2.83
1 81	1.92	1.89	1.71	1.74	1.75	1.79	4.53	4.56	4.55	4.55
1 06	1.08	1.06	1.73	1.63	1.74	1.72	3.77	3.72	3.82	3.86
1 64	1.63	1.63	1.88	1.89	1.88	1.87	4.02	3.91	3.96	3.91
rage of entire series							1.41	1.45	1.41	1.40

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Table 7. Comparison of Gold Assays on Buyer's Sample taken by Hand Methods

(Brackets indicate resamples. *O*, *D*, *T*, and *Q* indicate Original, Duplicate, Triplicate, and Quadruplet Sample, taken on last Quartering; values in oz)

Buyer					Seller					Umpire				
O	*D*	*T*	*Q*	Aver	*O*	*D*	*T*	*Q*	Aver	*O*	*D*	*T*	*Q*	Aver
19.65	19.54	20.16	19.96	19.83	19.87	19.69	20.20	19.85	19.90	.....	.....	.....	.....	.....
20.83	19.98	20.32	20.33	20.36	21.35	20.43	20.76	20.84	20.84	20.98	19.86	20.23	20.52	20.40
19.87	19.87	20.38	19.77	19.97	20.17	20.38	20.54	19.97	20.27	19.89	19.88	20.55	19.57	19.97
18.98	21.12	19.60	19.89	19.90	18.94	21.42	19.72	20.30	20.10	19.02	21.30	19.78	19.96	20.01
20.83	19.98	20.32	20.33	20.36	21.35	20.43	20.76	20.84	20.84	20.98	19.86	20.23	20.52	20.40
21.43	21.98	21.73	21.29	21.61	21.56	22.34	22.12	21.44	21.87	21.58	21.86	21.85	21.16	21.61
39.74	39.60	38.69	40.39	39.54	39.50	39.58	38.84	40.50	39.58	.....	.....	.....	.....	.....
41.62	40.64	42.61	40.41	41.32	42.30	41.26	43.04	40.84	41.86	.....	.....	.....	.....	.....

eying with assays. Assays reported by the seller are usually higher
e buyer's, the relation persisting through reassays and resamples (Tables
6). On high-grade ores larger differences occur (Table 7). Eliminat-
deliberate falsifying of the mining camp telephone-system of compari-
gures, there still exists the jockeying with assays frequently found in
ing practice. Simplest example is the cupeling at an intentionally, but
inally, high or low temperature, giving correspondingly low or high
vith no particular strain on assayer's conscience.

incipal and more specious excuse for jockeying is the fact that assaying of ordi-
o is not an exact science. Due to conditions shown in Art 6 (Screening), the
scientious and skilful assayer may be unable to weigh from the pulp two or
rges containing exactly the same amount of gold or silver. Hence it is cus-
o assay from 3 to 12 separate charges from each pulp, to weigh all the buttons
and report the average assay only. But the large differences occasionally
between the buttons furnishes a plausible excuse for weighing them separately
rding, in the average, such as differ excessively from the majority. This elimi-
ay be done by the assayer, or by the settlement clerk, neither of whom may
responsibility. It is natural that judgments of buyer and seller will differ as
uttons should be rejected; but even a conscientious umpire is at a loss what to
nen buttons vary by several oz per ton. Possibilities for jockeying are greatly
when 2 to 4 multi-samples, with repeats, umpires, and resamples are made,
consideration of perhaps 160 separate buttons for one settlement.
r opportunity for jockeying is the relation between assays and treatment charges,
-called "splitting" points with assays. Assuming that an ore contains approx
per ton, and treatment charge is $3 if below 1 oz, and $4 if 1 oz or more, with
o per oz; the buyer prefers a final average of 1.00 to 1.045, and seller anything
5 to 0.955, each preferring the critical assays of 1.005 or 0.995 oz gold respec-
This is shown as follows:

045 oz	= $20.90	0.995 oz	= $19.90
ss treatment ch'ge	4.00	Less treatment ch'ge	3.00
	16.90 *A*		16.90 *D*
00 oz	= 20.00	0.955 oz	= 19.10
ss treatment	4.00	Less treatment	3.00
	16.00 *B*		$16.10 *E*
005 oz	= 20.10	Seller prefers *D*, but can stand *E*.	
ss treatment	4.00		
	$16.10 *C*	Values of 0.955 and 1.045 oz (0.09 oz difference) are equal commercially.	

yer prefers *B*, but can stand *A*.

tion of the splitting point on RR freight classification arises in a similar manner, where the ore is bought f o b at shipping point.

12. Flow Sheets

sheets 1, 2, and 3 are selected from a series of 55 compiled for the U S of Mines (7). The form is a modification of that used by D. W. Brun-
Each line represents a distinct operation. Single spacing indicates passes from the operation shown in one line to that in the line below, spacing indicates that the flow of ore divides at that point, different taking courses shown by arrow points. Figures after crushers and indications of standard sizes and fineness of crushing. Percentages r sample are shown. At certain stages approx wt of sample is given, n original lot weighing 100 000 lb. Discard, unless followed by lines e spacing, indicates elimination of that portion from further sampling.

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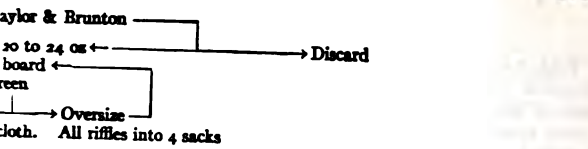
Hand-sampling Mill (Flow

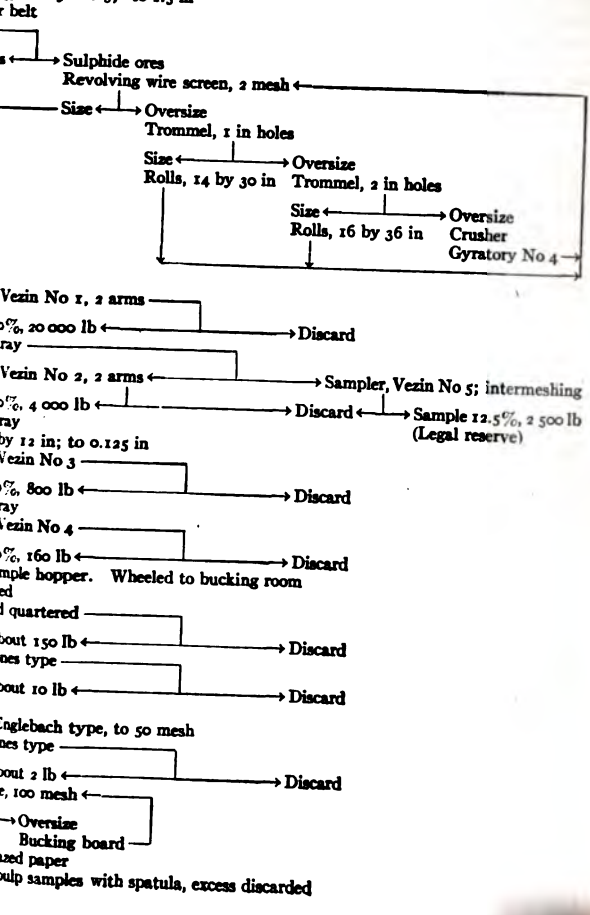
Railroad beside storage bins
Shovel sample
Sample 10%, 10 000 lb, left in car ← → I
Shovel sample
Sample 50%, 5 000 lb, left in car ← → I
Car taken to coning floor
Spread in ring, and coned over wooden cross insid
Quartered
Sample, 50%, 2 500 lb ← →
Spread in ring
Coning and quartering repeated until
Sample weighs 150 to 300 lb ← →
Taken to roll room, and partly dried if necessary
Rolls, 12 by 20 in; about 20 mesh
Riffle, Jones type
Sample, 10 lb ← →
Dried
Grinder, Englebach type, to 50 mesh
Riffle, Jones type
Sample, about 2 lb ← →
Hand sieve, 100 mesh ←
Size ← → Oversize
↓ Bucking board
Roll on glazed paper. Put in 6 sacks with spatul

Mechanical-type Sampling Mill

Railroad car over hopper. Unloaded into hopper
Crusher, 13 by 24 in; to 2.75 in
Rolls, 16 by 42 in; to 2 in
Shaking tray to elevator No 1
Sampler No 1, Brunton oscillating
Sample 20%, 20 000 lb ← →
Rolls No 2, 16 by 36 in; to 1 in
Shaking tray
Sampler No 2, Brunton oscillating
Sample 20%, 4 000 lb ← →
Rolls No 3, 14 by 27 in; to 0.375 in
Shaking tray
Sampler No 4, Brunton oscillating
Sample 20%, 800 lb ← →
Rolls No 4, 8 by 20 in; to 0.125 in
Shaking tray
Sampler No 5, Brunton oscillating
Sample 20%, 160 lb ← →
Rolls No 5, 8 by 12 in; to 20 mesh
Locked sample car. Taken to sample room
Half-shovel, Brunton
Sample, 20 to 40 lb ← →
Steam dryer
Englebach grinder, to 50 mesh



Sampling and Crusher-roasting (Flow-sheet No 3)



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18. Costs

Table 8 shows operating costs of customs s
Montana, and Nevada, during 1902 to 1914
price of labor, distance from supply market
service rendered. The probability of receivi
high-grade gold ores of camps like Cripple
necessitates a distinctly better mill and opera
ups, more assaying, and generally greater
grade gold, silver, lead, and copper ores, of s
Plants exclusively for transit sampling may
tomers, and need small clerical force; wherea
assaying, clerical force, administration, demu
increasing with value and quantity of ore so
smelting points.

Table 8. Costs of

Plant	Period, months	Tons in period	hr per day	Common labor, wages, dollars	Costs, cents				
					Mill labor	Mill machy and supplies	Mill power, light, etc	Total mill	Assay
A	3	22 600	10	2.50	36.2	7.5	2.1	45.8	4.5
B	24	111 004	10	3.00	19.4	1.5	2.4	23.3	6.7
C	3	15 433	8	3.00	35.6	4.7	2.3	42.6	4.1
D	3	35 076	9	2.75	13.8	2.6	1.7	17.1	0.2
E	1	9 779	8	3.00	11.0	2.0	1.7	14.7	0.1
F	3	52 899	8	3.50	16.4	3.6	1.1	21.1	2.2
G	3	49 712	8	3.50	13.9	4.6	2.4	20.9	1.9
H	24	199 537	8	3.00	23.3	4.1	1.4	28 8	3.3
I	9	105 252	8	3.25	25.5	3.9	4.0	33.4	3.8
J	3	35 094	8	3.25	23.4	6.0	1.9	31.3	4.0
K	3	17 647	8	4.00	26.8	4.5	4.1	35.4	9.2

w = ore transferred by wheelbarrow
t = ore transferred by tram
m = ore transferred mechanically
g = gold ores exclusively
s = silver ores exclusively

h
o
c
p
•

Table 9 shows one month's mill-labor costs, take
day-shift run only. This form was designed for t
being exchanged monthly between managers of 6 p
Platting the cost figures for a year's run develops int
to efficiency of management.

Hand methods at smelters are usually con
and the preliminary fractional shoveling may
crushing, or some other department. Therefo
difficult and of doubtful value. Different sm
per ton. Assuming a 100-ton lot, crushed to o
cost of fifth-shoveling this to 20 tons, half-sho
quartering to 250 lb, will be about $1.35 per
tramming the reject a reasonable distance
include mill supplies and machinery, maki

Table 9. H Mill. Labor Costs for Month Ending .....

9 648 tons	Days, hr	Cost, $	Cents per ton	Remarks
g.....	31	93.00	0.96	
n.....	31	165.00	1.71	
ng.....	190 1	570.60	5.92	
.....	1 7	5.80	0.06	11.5 cars (321.5 tons) per shift; 48 tons per lot. 200 clean-ups. Mill idle, 6 days, 2 hrs
.....	24 1	72.40	0.74	
.....	95	285.10	2.95	
room.....	71 2	248.85	2.58	
ps.....	75	225.00	2.33	
.....	42 3	183.00	1.90	
g mill.....	6 2	18.75	0.20	
g site.....	15 2	45.75	0.48	
an.....	31	124.00	1.28	
.....	614 2	2 037.25	21.11	30 shifts

nor any supervision or administration. At 25¢ per hr, the single of half-shoveling 10 tons costs about 5¢ per ton. Were it feasible to large a lot as 100 tons by coning and quartering only, labor cost 30 to 50¢ per ton, depending on care required for the grade of ore. l, the cost of coning and quartering varies from 5 to 10 times the cost nical sampling.

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SECTION

ASSAYI

BY

E. J. HAL

PROFESSOR OF ASSAYING, COLUMBIA SC

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Note. — Numbers in parentheses in text refer

Introduction. This subject is treated from t equipment and methods, with particular reference districts. Rare minerals, and some others of low assay or analysis requires special laboratory fa methods, see Fulton's "Manual of Fire Assayin Ore Analysis."

In a remote region, the question whether to tak the samples, will generally be decided by the nu factory fuel for fire assay, and distance from an e case, to secure reliable results, adequate equipme reduction of samples if transport of large sampl and, unless individual samples are reduced to sm that of assay equipment.

It is often important, in making mine examina results on certain samples, in order to resample in c sampling may be curtailed by advance informat samples is large, or the mine is in a remote distric ative, notwithstanding that it is rather bulky and assay work in the field is often beset with difficulti ment, simpler and more convenient methods of sought, but there is no satisfactory substitute for

e may be used for identification and qualitative tests (Sec 1); quantitative uire much practice and skill, and the results are unreliable. Further, fewer be made per day by blowpipe than by standard methods, and the work tious. An example will indicate the inherent shortcomings of the blowpipe Employing the usual charge of 100 mg for a 1-oz gold ore, 0.0034 mg of gold will

Combining results of 10 fusions gives only 0.034 mg, a quantity too small to dy measured without a microscope, and requiring a delicate balance for weigh- otal wt of ore used, 1 gm, is insufficient for a representative sample, and each d require 2 to 3 hours. The blowpipe is more applicable to high-grade silver ase metals. The so-called "pocket smelter" gives no more accurate results lowpipe.

seful for testing gold-bearing gravel, or pulverized gold ores, particularly those e base metals are oxidized. In experienced hands, it will give better results ores containing coarse gold than the ordinary assay on 0.5 A T (1 A T, or assay 6 gm) charges. (See Art 8; also, Sec 25, Art 7, Sec 31, Art 5.) The pan is re the gold is combined as telluride or is enclosed in sulphides, unless the sample ted.

1. Equipment

ng and grinding devices. If power is available, and samples are numerous, mechanical devices for reduction of sample should be used. e time and energy, and diminish tendency to excessively large cuts ore, which vitiate accuracy of sample. In selecting a JAW CRUSHER, strength, weight, and convenience in cleaning are important; for ores ack (due to moisture or talcy nature), a crusher with forced discharge le. For fine grinding, the DISK PULVERIZERS are efficient. There are akes; those having a stationary and a revolving disk give excellent hard and granular ores, though for talcy or heavy sulphide ores, they atisfactory. Ores of this class are better handled by a PLANETARY in which a revolving disk travels in an orbit, and discharge is positive. uction of coarse ore by manual labor the JAW CRUSHER is most efficient. small jaws and heavy flywheels of large diameter are necessary. a knack in operating these machines, slow feeding and high rotative ng essential. A simpler device is a STEEL PLATE AND HAMMER (cast ss very heavy, is apt to break), and a 3 or 4-in section of 10 or 12-in revent the pieces from flying; or, instead of the hammer, a "tamper" ler piece of pipe will be found more efficient. For added convenience, r may have a long handle, projecting through a hole in a board over- the pipe section screwed to a flange for sta- or fine grinding, nothing is better than BUCK- MULLER, which, to minimize salting of samples ust have a finished surface and be free from kboards with rim or flange on 3 sides have an surface. Some assayers substitute a piece of channel iron for the buckboard, using a muller roove. Mullers should not be over 4 in wide, ter bearing surface diminishing the unit crush- re. The Guanajuato Development Co has long buckboards and cobble-stone mullers, with usands of samples have been ground. They are a silicified rhyolite, and it is stated that the ited less than 0.5%. (See also Sec 25, Art 4.)

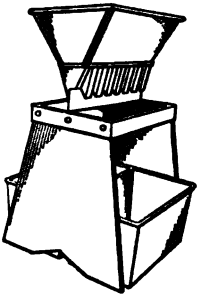

Fig 1. Jones Riffle Sampler

ng devices. A convenient and satisfactory the Jones riffle (as made by Denver Fire Clay . It has short spouts, facilitating cleaning. If it is set on the work hat its center line coincides with the edge of a rectangular open-

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the bench, the rejected portion is delivered c
The opening is readily closed with a cover
are to be saved. The old trough and space
use; when carelessly used, results are unre

Furnace depends on method of assay, av
tion, and amount of work to be done. Mu
all work; gives a more uniform and cons
effects of furnace gases. It usually gives t
or 3 times as long as when in direct contac
wood require this type. Disadvantages: l
limited, and destruction of muffles compara
cost of muffles is generally offset by savin
tain life may be the deciding factor when
tages of separate crucible fusion and mu
tically the reverse of above. COMBINATION
separate furnaces into one compact unit, u
cupellation, but it is difficult to maintain
perature. It is recommended, however, fo
transportation is poor. It is generally pre
obtainable for practically all fuels.

Furnace built in field, for solid fuel, is rect
mixture of ⅔ burnt and ⅓ raw clay. Ordinary b
less. Furnace is bound with angle irons at corne
sheet-iron case. Transverse iron bars at front an
project into the draft opening, for convenient c
grate bars is an opening, through which the muf
at the rear. Muffle opening is closed by a plu
between muffle and furnace walls must be sufficl
If old crucibles are available, they may be grou
water and 10 to 20% of clay, and rammed into a
shape desired for interior of furnace. Slag adher

Fuel is coal, coke, or charcoal, but the above f
burns rapidly, requires constant attention, and g
is apt to coke and form a solid mass, which must
ature is low until most of volatile products have
sults in muffle furnace, thickness of bed being

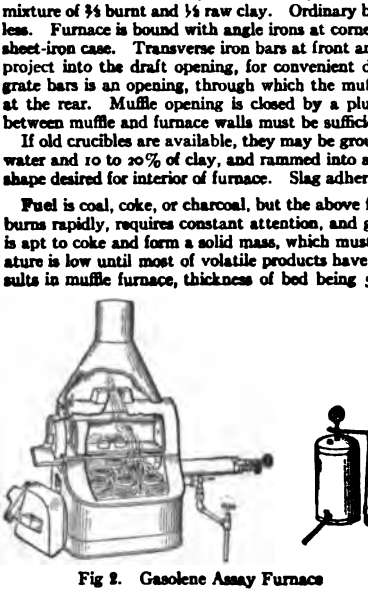

Fig 2. Gasolene Assay Furnace

while heating to required temp than subse
avoided. Two of the largest manufacturer
Colo, and Braun Corp, Los Angeles, Cal.
type, gasolene consumption for which is sta

taking 6, 8, and 10 20-gm crucibles respectively. The ⅛ or ¼-in pipe ws usually furnished with these furnaces can be replaced with advan- small-bore copper tubing, known as hollow wire; it is easier to install nates numerous joints, with their liability to leakage.

ces. A cheap spring-balance, costing 15¢ to 25¢, with a pan suspended ngs, answers for weighing stock charges. PULP BALANCE should have ling at least 1 A T (assay ton = 29.166 gm), and should be capable of mounted and packed in the case drawer. They have either all-steel or steel knives and grooved agate bearing plates, the latter being prefer- r FINE BALANCE, following points should be considered. Since silver not require extreme accuracy in weighing, a very sensitive balance, always slow and liable to derangement, should not be chosen for this ne. Gold determinations are most exacting, and for a large number of , the balance should be of the best. The most sensitive balances are cate, and will not stand rough handling. The inverted or "no column" ecommended. Several makers have special portable balances, which but less satisfactory than the standard types, and should not be used anent equipment.

hts. For GOLD AND SILVER ASSAY, combination set of 0.01 to 100-gm o 4-A T "second grade" weights is advisable. The ASSAY TON (A T) as many milligrams as the avoirdupois ton contains troy ounces; there- g of gold or silver obtained from 1 A T represents 1 troy oz in 1 avoirdu- The short (2 000 lb) and long ton (2 240 lb) will be represented by an aining respectively 29 166 + mg (29.166 gm) and 32 666 + mg (32.666 EAD WEIGHTS comprise 1 mg to 1 gm, and should be of the highest grade. ights, as furnished by supply houses, are usually inaccurate and should . Slight discrepancies can be disregarded for silver weighings, but a e of 0.01 mg causes an error respectively of 20¢ on 1 A T, and 40¢ on harges of gold ore. ANALYTICAL WEIGHTS for base metal and miscella- terminations should include 5-mg to 100-gm first-grade weights. If ses of work are to be done, the 5-mg to 1-gm weights may be omitted analytical set. It would be possible also to eliminate the gram weights set, but this is inadvisable.

ore is weighed in METRIC TONS, gold and silver are reported in grams . As there are 1 000 kg in a metric ton and 1 000 mg in 1 gm, 1 mg silver obtained from 1 gm of ore will represent 1 kg per ton. Example: of ore gave 15.5 mg of silver and 0.35 mg of gold, the ore contains 1.55 and 0.035 kg, or 35 gm, of gold per metric ton. Where metric system ved the A T weights are useless.

2. Reagents

rge (PbO) is employed in crucible assay to furnish lead for a button; many charges, as a basic flux, oxidizer, and desulphurizer. It should be r silver by fusing 10 A T with silica, borax glass and soda, and enough agent to give a 35-gm lead button. If appreciable bismuth is present, indicated in cupellation by a brown stain at the point where last of the idized. Bismuth causes inaccurate results, for which correction can not , as in case of silver.

m carbonate (Na_2CO_3). Commercial carbonate or soda ash (Na_2CO_3) ently pure for assaying and should be employed. Bicarbonate is un- y, costs more, and is more bulky per unit of Na_2O. Na_2CO_3 acts as a x and in certain cases is a desulphurizer.

Test lead is a solvent for gold and silver. to litharge, which is the principal flux. It m be tested by scorifying four 80-gm portions, until sufficiently reduced for cupellation.

Potassium carbonate (K_2CO_3) performs used with the latter it gives a mixture of low expensive, not essential, and, being hydrosc in moist climates.

Borax ($Na_2B_4O_7 + 10\ H_2O$) is a fusible ac solve many basic oxides. It should always b prepared by fusing commercial borax and pul

Silica (SiO_2) should be considered as an a its melting points, in combination with the than those generally used in assaying. Whe the fusible fluxes is necessary.

Potassium cyanide (KCN) acts as a red is useful in base metal determinations, but should be of good quality. Cyanide conta cyanate is absolutely worthless for some dete

Flour is the most convenient reducing age of lead.

Niter (KNO_3) is used to counteract exces has an oxidizing power of about 4.25 gm lea a determination may be necessary, preferabl occurring in the ore, usually pyrite. Mak pyrite, or enough ore to reduce 15 to 20 gm lea litharge. In one of these charges introduce red heat. Reaction should be completed in and divide difference in weight by 2, giving also acts as a basic flux.

Stock fluxes are advisable when many ass ores, using the proportions given below und desirable to omit niter or reducing agent, ad assays as required.

Reagents for wet analysis and parting unless otherwise stated.

Water. If distilled water can not be ob collected after a portion of the precipitation this, chemically treated water should be used silver assays, where chlorine, generally pre seriously. To prepare water for parting it sh of silver nitrate, heated, and allowed to stand decanted. This water is used for diluting HN portion of water not treated with silver sho and used for second washing.

2. Sampling (See

General procedure. Working down of often more difficult than making a satisfacto and care involved this operation is frequen sample, as usually taken, is apt to be much

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(For grinding and sampling devices see Art 1.) The original sample reduced to small size at place of taking, unless suitable mechanical available elsewhere. Table 1 assumes that the assay office will deal with samples of over 20 lb (preferably 5 lb or under).

Minimum Permissible Weight of Sample for Given Size of Ore (1)

mple	Low-grade and uniform ores		Medium ores		Rich and spotted ores	
Lb	Mm	Approx mesh	Mm	Approx mesh	Mm	Approx mesh
20	3.6	6	2.4	8	1.00	17
10	2.5	8	1.7	10	0.71	24
5	1.8	10	1.2	14	0.50	32
2	1.1	16	0.76	22	0.32	45
1	0.8	20	0.54	30	0.22	65
0.5	0.57	28	0.38	38	0.16	88
0.2	0.36	40	0.24	58	0.10	150
0.1	0.25	55	0.17	82	.....	.....
0.05	0.18	76	0.12	125	.....	.....
0 02	0.11	150	.....	.....	.....	.....

ethod of working down a sample can be ascertained only by experifor a number of similar samples it should be determined. The size icle indicated in Table 1, for a given weight of sample, is much finer sary in many cases. The screens used are generally multiples of 10. bor, the number of cuts when grinding by hand will be greater than e work (Table 2).

able 2. Working down Samples by Hand and by Machine

	Hand work			Machine work			
wt e	Mesh	Number of cuts	Final wt of sample	Original wt of sample	Mesh	Number of cuts	Final wt of sample
	4	.....	20 lb	20 lb	4	0	20 lb
	8	2	5 "	20 "	10	2	5 "
	20	2	1.25 "	5 "	40	4	5 oz
	40	2	5 oz	5 oz	120	assay	.....
	100–120	assay	.....	.....	.....	.....	.....
	40	3	2.5 oz	.....	.....	.....	.....
	120	assay	.....	.....	.....	.....	.....

ck accuracy of sampling (see Art 9), mix rejected portions thord repeat operation. If sampling procedure is correct, results will hin limits of assay accuracy. Thorough mixing and mode of dividing e as important as size of ore particle. The coning and quartering while capable of giving good results, is liable to serious error and the es (Art 1) is much preferred. Riffles should have an even number of d the sides should be kept in perfect alinement. Ore coarse enough to dging must never be fed on a riffle, and a flat-bottom scoop, over ore is uniformly distributed, should always be used. After splitting, ortions of a sample should differ but slightly in weight.

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Metallics in an ore necessitate following
large to insure an average value must be taken.
ore, and can be determined only by trying seve

Assume an ore containing coarse gold to be grou
to 150 mesh and screen. Wt of metallics and coa
with 40 gm of lead and cupel; wt of gold = 800 m

$-\frac{3}{40}$ gm = 14.508 gm = proportionate amount of
wt of gold = 200 mg.

Weight of gold in metallics from 1 A T of
" " " " siftings " "
Weight of total gold in 1 A T of ore

or 440 oz per ton.

If metallics on screen give a bead of more th
pling procedure is as follows. Ores containin
Ont, deposits, are extremely difficult, and in s
usual methods. A large sample must be take

Assuming it is necessary to take 50 lb of a 6-mes
sample: Weigh out 64.3 lb = 1 000 A T (1 lb =
Metallics on screen weigh 500 gm. Flux with sod
oxidizable constituents such as sulphur, arsenic, or
Sample the bar and cupel 500 mg. At same tim
copper, equal in wt to the silver and base metal
silver in cupellation of latter to bead from bar sa

coarse metallics contain 380 gm silver. Of the 3

gm (291.66 − 5) = 286.66 gm, the proportionate
ore, and grind to 120 mesh. Metallics on screen
lead to a button of at least 60 gm, since rich gold-a
button and cut sample from different portions of
10 gm of lead foil. Weight of bead = 450 mg;
silver in metallics.

Of the 120-mesh siftings, weigh 0.5 A T − (
(0.25 + 0.4) = 13.933 gm, or 0.5 A T minus the p
and 120-mesh metallics. Resulting bead = 350 m

Weight of silver in coarse metallics from
" " " " fine " "
" " " " siftings (120-mesh)
Weight of total silver in 1 A T of ore

or 1 530 oz silver per ton.

Low-grade, spotty gold ores (2), illustra
Black Hills, S D, have their values so segre
charge of 100-mesh product is unreliable.
(opening in a 100-mesh screen) weighs 0.032
which the particles of gold are of that diam, t
0.5 A T, whence the absurdity of expecting to
this material is apparent. It is necessary to
as this amount is large to flux, the coarse gol
mercury, with enough water to make a slu
globules of mercury in suspension. This is g

ly of porcelain, or small pebble mill, then thinned with water, and to recover the amalgam. If the mercury does not collect well, add a uantity of sodium amalgam (Art 8; also Sec 31, Art 7). Place the in a parting flask or cup, dissolve the mercury in HNO_3, anneal and e gold. The gold may contain some silver and should be inquarted and The tailing is dried, mixed, and assayed, using 1 or 2-A T portions. results are combined in proper proportions.

composition tests. Approximate composition of an ore must be known, to proper charge. When ore is in lump form, this may generally be determined tion. If pulverized, take 5 to 10 gm in a pan or clock glass, add water and our off fine slime and repeat if necessary. Add more water and stratify remain- minerals by panning motion, then determine minerals and their approximate by inspection. Do not attempt to effect a concentration; instead, remove inguishable fine minerals, leaving the coarse particles so segregated that they entified and estimated. Treat about 1 gm of ore with HCl; rapid evolution of tes CARBONATES. Allow complete action, noting percentage dissolved, which ssed as base. Odor of H_2S may give the impression that the action is due to s. Mineral sulphides, however, react slowly with HCl and do not give a rapid of gas. HNO_3 will not answer for this test, as it attacks sulphides vigorously not possible to distinguish between the action of carbonates and sulphides. of chlorine denotes MANGANESE dioxide, and the amount of this mineral present udged by color of sample, remembering that pyrolusite is black. Add strong about 0.25 gm ore, boil, dilute, make ammoniacal, filter or allow to settle. Blue ndicates COPPER, and the percentage may be approximated by the depth of o test for ARSENIC and ANTIMONY, separate the concentrated metallic minerals ning test, and heat with blowpipe, or in porcelain capsule, mixed with a little Characteristic fumes of arsenic and antimony will denote their presence.

4. Crucible Method for Pure Ores

f lead button. Crucible method is recommended for field work, as olicable to all ores, permits use of large, representative samples, and multiplication factor. While the charges vary for different ores the is the same for all. The ore is mixed and fused with such fluxes as will mplete decomposition and solution at a reasonable temperature, thus g the gold and silver. The oxidizing and reducing agents are proper- o produce 20 to 25 gm of lead, which collects the gold and silver in one A lead button of 25 gm is sufficient to collect the values in 1 A T of arger button is not in itself detrimental, but it must be realized that harge is made to yield a button of definite size, all lead reduced in excess amount diminishes the quantity of litharge available as a flux, and reduction may thus seriously affect results. Lead buttons of less than ay or may not collect all the values, and, if adopted, this point should be It is essential that the gold and silver shall be completely liberated l the lead has settled.

n. Manner of fusion is important and varies with different ores. Close n should be given to instructions on this point, which accompany the charges. As a rule, a fusion should be complete in 25 to 35 min. at too low a temperature will affect recovery of both gold and silver; a temperature usually affects only the silver. Do not place crucibles rnace until it is up to working heat.

. A glassy or vitreous slag, which can be drawn into threads when is generally desirable, since a slag of opposite characteristics may indi- omplete decomposition, and may also cause poor collection of lead. lags are produced by having sufficient silica and borax present to form olution with the bases. Character of slag is controlled by these reagents.

Charges for pure ores. Pure ores are rock-forming elements, with small amounts viz, S, As, Sb, Cu, Zn, etc. The problem i that the ore will go completely into solu Na_2CO_3 and PbO, are capable of readily dis bases is rather limited. To decompose basi to combine with the bases in the ore and fl varying amounts of SiO_2 to produce a sla This is indicated in Table 3. In first colum in third column, the charge is for 100% of th for what may be called the neutral ores, whi the base and give a good slag. It is not su compositions indicated in these charges; but base, it is easy to vary the charge accordin than one base is present the charges can be stituents. There is considerable latitude in ore, if enough SiO_2, or even a moderate e combination, and then enough flux to give good results will be obtained. Flour is the u 10 gm lead per gm. Amount required for a the ore often contains reducing or oxidizing a total reducing action, in terms of lead; not or amount of lead desired. (Remarks follow fore making use of the charges.)

Table 3. Typical As

Acid			Calcium or Magnesiu	N
SiO_2, 100% (*a*)			$CaCO_3$ o	Si
Ore	0.5	A T	.....	
Silica	0	gm	.....	
Sodium carbonate	20	"	.....	1
Litharge	45	"	.....	4
Borax glass	5	"	.....	
Reducing agent =	20	gm lead	..... =	2

			Iron Oxid	F
SiO_2, 100% (*a*)				Si
Ore	0.5	A T	.....	
Silica	0	gm	.....	
Sodium carbonate	20	"	.....	1
Litharge	45	"	.....	4
Borax glass	5	"	.....	
Reducing agent =	20	gm lead	..... =	2

			Manganese Dio	M
SiO_2, 100% (*a*)				Si
Ore	0.5	A T	.....	
Silica	0	gm	.....	
Sodium carbonate	20	"	.....	
Litharge	45	"	.....	4
Borax glass	5	"	.....	1
Reducing agent =	20	gm lead	..... =	4

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Table 3 (*Continued*). — **Typical Assay Charges**

ALUMINUM OXIDE

SiO_2, 100% (*a*)		Al_2O_3, 60% SiO_2, 40% (*h*)		Al_2O_3, 100% (*i*)	
	0.5 A T		0.5 A T		0.5 A T
	0 gm		0 gm		10 gm
arbonate	20 "		15 "		25 "
	45 "		45 "		75 "
ss	5 "		5 "		8 "
luoride	0 "		5 "		8 "
agent =	20 gm lead	=	20 gm lead	=	25 gm lead

LOW-GRADE ACID ORES AND TAILINGS (*k*)

Ore	3.0 A T	
Sodium carbonate	120	gm
Litharge	150	"
Borax glass	15	"
Reducing agent =	35	gm lead

s added SiO_2 is considered as ore for the purpose of composing charge, and fusible added accordingly. These charges form silicates and borates with the bases. A T of ore + 15 gm SiO_2 requires same flux as for 1 A T of acid ore. Excess e that required will do no harm, provided sufficient soda and litharge are used. fore apparent that charge (*c*) may be used for 1 A T of acid ore. Borax glass mitted in many charges for siliceous ores, eliminating the more expensive flux. ven when not required, does no harm. In the absence of large amounts of im- da and litharge may be used in these charges to a certain extent interchange- atio of 2 to 3, without objectionable results. Ores frequently contain more As, Sb, Cu, Zn, which should be oxidized. As this can be done only with he charges as given will have the greatest general application. There is no vantage in a charge of the following type, which is in common use: Ore, 0.5 A T; 5 gm; K_2CO_3, 15 gm; borax glass, 15 gm; litharge, 30 gm; flour, 1.75 gm. f litharge in above amount makes it impossible to obtain a lead button too large ation, but this is of doubtful advantage. Any saving in cost of litharge is offset e of potassium carbonate and increase of borax. The amount of reducing agent depends on presence or absence of reducing and oxidizing agents in the ore. e½) up to 13% of the ore, or its equivalent, is permissible with these charges; replaces all the reducing agent, giving a 24-gm button. In charges (*c*), (*g*), (*i*), rger lead buttons are produced because of increase in size of charge. In (*d*), (*e*),), a lead button of 20 to 25 gm is desired, but excess reducing agent is necessary to t oxidizing effect of Fe_2O_3 and MnO_2. In (*f*) and (*g*), manganese as MnO_2 does ilily into solution; high soda interferes with its reduction; borax glass mate- sts dissolution. Charge should be brought rapidly to fusion temperature, the being 1 050° C, or light yellow. Al_2O_3 in combination is a frequent constituent harges *k* and *i*); it slags with difficulty, and interferes with collection of lead. of fluorspar (CaF_2) assists in taking Al_2O_3 into solution. If omitted, soda and ss must be increased 100%.

5. Crucible Method for Impure Ores

include ores containing enough S, As, Sb, Zn, Cu, Ni, Co, and Te, se slag and cupel losses, unless their effect is counteracted. For this ing charge is necessary, and as litharge is the only oxidizing agent which sed in excess, it is employed freely.

pyrite ores containing less than 13% pyrite have been discussed in Those containing over 13% pyrite are treated by either niter or nails

NITER METHOD gives more reliable results for gold and silver, and ad- presence of other impurities. It usually requires a preliminary assay mine reducing power of the ore, and this, together with the large flux nakes it slow and costly. The NAILS CHARGE is capable of direct appli-

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cation, but only to sulphides comparatively f
results by this method are fairly reliable, wi
results, except in the case of small amounts o
the loss increases with percentage of pyrite,
When in doubt whether an ore is pure or pyri
satisfactorily, even though the pyrite is less th

Niter method. In this the KNO_3 oxidizes
quired to reduce the desired amount of Pb, th
by litharge. To obtain the required definite
large amount of base is necessary, in which
to SO_3.

Preliminary charge. The reducing power of a
the amount of niter required. A knowledge of the
with sufficient experience, it may be determined by
erally necessary, using following charge and calculat

Ore	0.1 A T		(
Sodium carbonate	7	gm	
Litharge	50	"	
Wt of lead reduced = W			or

Charge should be fused in 15 to 18 min, at a light y
long a time in the furnace, gives low results. The ch
or HAWLEY'S METHOD may be employed. Weigh

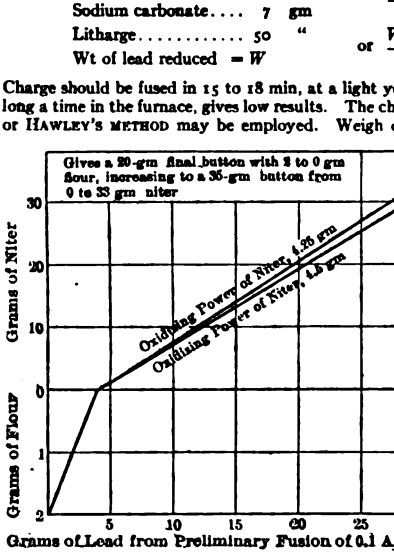

Fig 3. Reduction Diagram for Impure Or

Table 4. Final Ch

Over 50% SiO_2 in ore			
Ore	0.5 A T		Ore
Sodium carbonate	10	gm	Silica
Litharge	75	"	Sodiu
Potassium nitrate	X	"	Litha
			Borax
			Potas

sion. Lead button of 20 to 25 gm is desirable for ores containing
% pyrite; for more than 50%, the button should weigh 25 to 35 gm,
reasing with proportion of pyrite. Success of this charge is largely
on manner of fusion, especially in high-pyrite ores. For low to me-
nts of pyrite, a temp of 1 000° to 1 050° C (yellow heat) is satisfactory.
ercentages, a high initial temp is essential, say 1 050° to 1 100° C (light
intained for 10 min, or until charge is thoroughly liquid. Temp of
hen lowered, and fusion continued for 25 to 30 min. Charge should
ved to reach original high furnace temp, or silver losses will increase.
should be made in a high-grade crucible, like the Battersea soft clay
be confused with the hard clay crucible). There is but little boiling
ions, and the dry charge may fill ¾ of the crucible. From 75 to 100
is used to insure complete oxidation of S to SO_3, which combines with
ing Na_2SO_4. This floats on slag as a thin liquid, forming a white
oling.

ethod is essentially reducing, with no attempt at oxidation. Sulphur is
slag, size of lead button is regulated by adding only sufficient litharge,
is desulphurized by nails. It is therefore apparent that impurities can
ized, and have a tendency to concentrate with the lead. Charge is:

Ore	0.5	A T
Sodium carbonate	30	gm
Litharge	25	"
Borax glass	10	"
Nails (10 d)	4	"
or 1 R R spike		

gh in pyrite, or contains a large amount of base, borax glass should be
5 to 10 gm. Fusion should be at temp of 950° to 1 000° C (yellow),
25 to 40 min. Avoid a strongly oxidizing furnace atmosphere, which
unsatisfactory fusion. In withdrawing nails, shake well while their
mmersed in slag, and not simply knock against side of crucible.

ores. Presence of copper up to 5% does not interfere in charges
in litharge in excess of that which is required for a lead button (Art 4).
s not interfere in the fusion of an ore, but has a strong tendency to
e the lead button. If the button contains about 8% Cu, or more,
e if cupeled at 700° to 750° C ("feather temperature"). If button
or less of copper, it will cupel at a lower temperature than pure
though copper is present in amounts insufficient to cause a freeze,
ental by increasing loss. While some copper remains in bead, the
ss than the induced loss, giving low results (especially for silver).
ss, and the amount of copper retained, increase with proportion of
e lead button. Therefore, copper in excess of 2% of the lead is not
Copper is eliminated from the lead button by large quantities of
her added in the charge or formed by scorification. Amounts of ore
rges indicated below are intended to give final lead buttons contain-
n 2% copper; 2 to 3% copper will stain cupels gray green, shading
color at center. A blackish green indicates presence of too much
hile 175 gm of litharge is given for all charges, it should be reduced
r is lower than stated. If a scorification is not feasible and ore is
er, the ore charges must be smaller, without diminishing quantity of

er 20%, use 0.5 A T
45%, " 0.5 " and scorify lead buttons + test lead.
60%, " 0.25 " ; run 4 charges, combine buttons 2 by 2, and scor[if]

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Note. — In SCORIFYING BUTTONS, add sufficie
depending upon amount of copper; 70 gm lea
the copper. Add about 2 gm SiO_2 to preve
increase copper in the lead button, for, if no
from the scorifier. Temperature should be l
When lead buttons are not too large for cupell
time is saved by melting the buttons with a co
in a scorifier, and pouring. About 75% as mu
scorification.

Oxidized ores include those which do
of that required to produce suitable lead

ACID GANGUE

Ore	0.25 or 0.5 A T	
Sodium carbonate	15 gm	
Litharge	175 "	
Borax glass	5.0 "	
Reducing agent	= 25 gm lead	

Note. — It is a disadvantage to produce lea
especially when scorification follows, because w
it increases in total amount.

For copper ores with basic gangue, se
to add; then increase soda and litharge
which gives a good fusion should be used,
large amounts of litharge.

Sulphide ores, or those which have a

Ore	0.2
Sodium carbonate	
Litharge	
Potassium nitrate	
(see above note)	

PRELIMINARY CHARGE:

Ore	0.1 A T	For 0.25 A
Sodium carbonate	7.0 gm	
Litharge	50.0 "	For 0.5 A
Lead reduced = W gm		

(See iron pyrite, niter method, for p

SiO_2 or borax glass should not be adde
creasing the litharge. Fusions are made
clay crucibles, as they withstand the act
silica.

Arsenic and antimony ores, being si
gether. When present in an ore to any
stibnite (Sb_2S_3) and mispickel (FeAsS).
in silver minerals is generally insufficie
should the amounts present require atten
Sb_2S_3 and FeAsS. The tendency to high
larly of silver, and the resistance of oxides
flux, are the principal difficulties in assay
charge is generally required to decompo
not always necessary, but the use of nail
carrying values, will be produced when a
5 to 10% As or Sb, the charge may be c

onstituents, possibly increasing the litharge, and varying the amount of gent. Above 5 to 10%, a large proportion of litharge should be used.

	(*a*) FeAsS to 75% Sb_2S_3 to 50% Acid gangue	(*b*) FeAsS to 100% Sb_2S_3 to 75% Acid gangue	(*c*) Sb_2S_3 100%
.....	0.5 A T	..... 0.5 A T	..... 0.25 A T
.....	....	..... 6 gm	..... 6 gm
bonate....	15 gm	..... 15 "	..... 15 "
.....	75 "	..... 90 "	..... 75 "
nitrate....	*X* "	..... *X* "	..... *X* "

of KNO_3 is determined by preliminary fusion. For principles, and formula (*a*) and (*b*) see iron pyrite charge (Art 5). For charge (*c*) use,

$$\frac{(W \times 2.5) - 2.5}{4.5} = X$$

es containing no more than 5 to 10% Zn do not affect method of assay. ilty in assaying ores high in zinc is to slag the zinc completely after it , so as to prevent loss of gold and silver by volatilization with the zinc, g and cupel absorption. To accomplish this, the proportion of flux sually large. An oxidizing method is necessary where zinc is present in gold and silver ores. If the zinc is in an oxidized state, a charge ilar to those for pure ores (Art 4), the exact composition depending ortion of acid and base, but increasing the relative amounts of fusible or sphalerite ores containing up to 35% ZnS, having excessive reducing niter charge as for iron pyrite (Art 5). With zinc above that amount, da and litharge, adding SiO_2 and borax glass if ore is of low acidity. QUANTITIES, for 100% ZnS, are as follows:

Ore	0.5 A T
Sodium carbonate	15 gm
Litharge	120 "
Silica	5.0 "
Borax glass	10 "
Potassium nitrate	*X* (see above)

le ores. Tellurium, when present, is nearly always combined or h gold or silver. The smallest amounts therefore require attention. 0.1 or 0.2%, its interference is confined to slag losses; but as it in- re is a tendency to concentration in the lead button, resulting in in- pellation losses, with a possible splitting of the bead. In extreme y cause complete absorption of gold and silver by the cupel.

	(*a*)	(*b*)	(*c*)
.....	0.5 A T	..... 0.25 A T	..... 0.1 A T
rbonate..	0.33 "	..... 0.33 "	..... 0.33 "
.....	3.5 "	..... 3.5 "	..... 3.5 "
.....	0 gm	..... 2.5 gm	..... 5.0 gm
s.....	10 "	..... 10 "	..... 10 "
agent..	= 25 " lead	..... = 25 " lead	= 25 " lead

up to 300 oz of gold and silver use charge (*a*), above that amount use for specimens composed largely of telluride, in which case (*c*) is Charges should be fused at moderate temperature, though it is have too low a temperature, giving low results, probably due to in- ecomposition of the tellurides. Charge of litharge is high, to give a g, highly oxidizing slag, which oxidizes the tellurium, liberating gold Gangue of telluride ores is generally acid, but if basic, the bor

glass must be increased or SiO_2 added. SiO
excess of requirements, as it diminishes the
defeat the desired object. SiO_2 is added in
clean slag.

Lead ores are the simplest to assay for
present acts as a reagent. Oxidized ores sho
depending upon the gangue. As the lead
flux decreases. If the ore has excess reducing
results, though the nails method may be used
the scorification method is the simplest for l

Nickel and cobalt ores, carrying appre
are uncommon except at Cobalt, Canada, a
the gold is negligible. Ores of this district
large amounts of As, Sb, and Bi, besides the N
of considerable size is an important feature
are readily reduced with lead in fusions, and,
with or prevent cupellation, by formation o
Experience with these ores indicates a choic
metallic scales are present they must be remo
For ores up to 100 oz silver, crucible metho
and scorification methods give equally goo
above 500 oz, scorification is more reliable.
of impurities, particularly bismuth, it may
method (3), treating with HNO_3 either the o
obtained from fusion of the ore. These state
latitude should be allowed in their applicati

	Crucible (*a*)		Crucible (*b*
Ore	0.5	A T	0.25 or 0.5 A
Sodium carbonate	0.66	"	0.75 "
Litharge	2.5	"	3.5 "
*Silica	0.3	"	0.33 "
**Borax glass	0.33	"	0.66 "
Reducing agent	=20 gm	Pb	=25 gm P

* Charge of SiO_2 depends on quantity in the o

For the less impure ores charge (*a*) may be used
using ¼ or ½ A T, depending upon value and tot
tion of the lead buttons is necessary or desirable.

6. Scorificat

Limitations. This method is treated br
it is less convenient and has a smaller range
It is not suitable for highly basic ores, nor
unless several portions of the sample are
reduce the multiplication factor and the per
due to the small amount of silver present.
ores, but it is best applied to impure ores in
poses the impurities and throws them into t

Charges are closely similar for all ores. T
50 to 75 gm test lead and 1 to 3 gm borax
character of impurity present. Ore is mixe
of scorifier and covered with other half, ov
Charge is placed in muffle at light yellow

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hen charge is thoroughly melted, the muffle door and damper are give a good draft, and temperature is slightly lowered. Slag should y fluid at all times; it should appear red hot from the lead almost to contact with scorifier, and not merely where the hot litharge from f lead runs into slag. If sulphur is present, little grease-like patches will float on the slag, and the furnace should be hot enough to keep en, and, as the assay progresses, to cause the slag to run up side of capillary action. With an excessive amount of reducing minerals e assay will not show rapid decomposition and solution of the ore, incipal flux is litharge, and this can not form to any extent until the gents have been oxidized. Oxidation is carried on until the lead is hirds to completely covered by slag; charge is then poured.

7. Cupellation

eristics of cupels. Separation of lead from gold and silver by cupel- nds upon the high surface tension of the latter metals, which prevents sinking into the pores of the cupel, while the fluid PbO is readily Any material with a high melting point, not attacked by litharge, nular form, may be used for making a cupel. Magnesia, Portland d other substances are used to a limited extent, but bone ash is the ion. Several brands of manufactured cupels are on the market, but special advantage in using them. Bone ash in bulk will stand all f transportation, even to a thorough wetting; and, as cupels can be a hand mold at the rate of 100 per hr, the final product is cheaper. ould have a smooth surface, and readily absorb its own weight of thout cracking. Cracking is usually due to the cupel being too ssively fine bone ash, too much water or pressure, favor this condi- naking cupels there is no better bonding substance than water, the d being so proportioned to the pressure that the be firm enough to handle, but will break when or 10 in; they become much harder on stand- avity of the cupel should approximate the surface , with a radius equal to half the diam of cupel two-fifths of radius; the rim may be angular or rounded (Fig 4). This will give a capacity for to 25 gm, 30 to 35 gm, and 40 to 45 gm, for a 1-in, 1.25-in, and 1.5-in ctively. Weight of the cupel should be greater than its capacity

Fig 4. Shape of Cavity in Cupels

on. Ordinary bone ash, air-dried cupels should be placed in the together in transverse rows, sufficiently far back for uniform temper- oken rows should always be in the rear, and an empty row, or a fire- ould protect the front row from drafts. After heating 5 to 10 min ce at 850° to 900° C (light cherry), insert the buttons. As soon "uncover," open muffle draft and lower temp to 700° C (below full , as the burning lead produces considerable heat. When near the emp to about 800° C. This finishing temp is extremely important; large losses will occur and small beads may entirely disappear. A ade cupellation will always show a ring of litharge crystals surround- d at a distance of ⅛ to ¼ in. In case of gold ores, silver should be re cupellation (Art 8), saving subsequent inquarting and practically gold losses.

g is caused by vigorous expulsion of oxygen on rapid solidification **-metal button.** It is often difficult to prevent, but is minimized by

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a high finishing temperature and slow cooli
particles of the bead are completely ejected

Freezing is caused by too low a temp, c
temp freeze, if remaining lead is less than
this amount and raise temperature. A pi
assist "reopening." When cupellation h
rapidly; best done by placing in the muffle
If properly reopened, the results, though
Freezes due to impurities may sometimes
test lead and raising temp. Results are unr

Beads are cleaned by squeezing in stou
and then brushing. Beads need not be w
exact weight, except when the multiplicatio

8. Parting and

Parting of the gold and silver is done p
(Fig 5), 45 mm diam, using first 1 to 12 H
until action ceases, decantin
treating with 1 to 1 HNO_3, h
decanting and washing once o
(Art 2). Always decant clos
muffle or over flame. Take
breaking up; hence, at first
Flattening the bead to ¼ i
increasing surface of attack. Parting cup
gold should have yellow color after ignitio
of silver to 1 of gold, complete separation
described. But, as little as 2 of silver to 1
by boiling in 1 to 1 HNO_3, if the bead is les
ing more than 25% gold may be treated
without breaking up. It is impossible alw
a bead will part; 60% gold produces very s
the parting ratio limit, the acid action and
whether the bead parts or not. A pure g
criterion of complete parting. A properly
has lost at least 80% of its weight, without
ing. This is the best indication of complet

Fig 5. Parting Dish

Inquartation, or the addition of silver t
the bead with silver, in ratio of 1 gold to 4
and cupeling. The temperature must be
the button, being small, will finish at the
may be added directly to the ore charge, or
ured amount of a standard $AgNO_3$ solution
Since all the silver is not recovered, the valu
by assay of a blank charge.

Gold in pannings may be separated from oth
perfectly dry. This method is not recommende
very fine gold. It is better to add a globule of m
mortar, and rub them together. If the gold fai
sodium amalgam. (SODIUM AMALGAM is prepare
¼-in cube or less, to mercury, pressing the pieces
bottom of container with pestle or rounded stic
objectionable explosions will occur. Continue ad

p the amalgam under gasolene or light oil.) There is no objection to adding
um amalgam to the original mercury to cause a slight adherence to iron. The
m is placed in a parting cup, treated with dilute (1 to 1) HNO_3, washed, dried,
(see Parting). Residue is not pure gold; its fineness is determined by taking
inquarting (Art 8) with silver approximately equal to 2.5 times the actual
. Hammer or roll the bead to less than $\frac{1}{32}$ in, and part in a flask with dilute
O_3, giving two separate treatments, boiling each for 10 min. Wash, anneal,
l is of varying composition, the total product from pannings must be melted
l sampled. The fineness obtained represents the acid-treated product and
ness of original gold. A fineness determination made on original gold should
in calculations based on the weights obtained from acid-treated amalgam.

9. Check Assays and Salting

lant results on duplicate assays are not conclusive evidence of accu-
ially when samples are run under the same conditions. Greater weight
en to checks when varying weights of ore are taken and run at differ-
Checks obtained by DIFFERENT METHODS are the best indication of
esults. When it is undesirable to run duplicates on many samples,
s a whole may be checked by weighing out extra and equal portions
ore when the original charge is made up, and carefully mixing and
ese extra portions. If the work has been carefully done, this com-
ple should give results that check to the second decimal the average
om the individual samples. (See also Sec 29, Art 8–11.)

assays. To determine whether an assay sample is salted is often
The crude plan of introducing metal filings may easily be detected
and microscope. When fine native metal is employed the problem
ficult and conclusions less certain. If ore is coarse, the finer material
screened out and assayed separately from the coarse. The fines will
ariably run high, but if the discrepancy is great there may be reason
on. Where the metal used in salting has a different origin than that
a test of its fineness may bring out this point. Pulverized samples,
rich concentrates derived from the same ore, will pass as genuine
ot be detected. If the sample is not ground, the fines should be
ut and the coarse product crushed to produce another portion of fines
al to the first, which is then separated from the remaining coarse
In a normal sample, assay results of the two fine portions should not,
vary greatly. When salting is done with solutions, some of the rock
s will frequently decompose the original salt, rendering the metal
nd frustrating any effort at detection by solution of the metallic com-
he ore may be tested for the acid radical originally associated with
Gold and silver are the only metals likely to be introduced by this
ith a chloride radical for the former, and a nitrate for the latter.
apt to contain chlorides, and the nitrate may be decomposed. But,
which assays high in gold or silver and, even under the microscope,
ree gold, nor silver minerals, nor minerals which usually contain or
ted with these metals, may be strongly suspected of having received
s from unnatural solutions. (See also Sec 25, Art 9.)

l salting is more frequent; in fact, it occurs to an extent unappreciated by
engineer. This is particularly true where miscellaneous samples are being
he most common source of contamination is in the grinding apparatus, and
imum when small samples are being successively ground, some of which con-
s. It is almost impossible thoroughly to clean grinding appliances without
rein a prohibitive amount of barren material. Pores in the grinding surfaces
l with ore; particularly when metal is present, which is gradually fed out
surfaces are abraded. Separate apparatus should be used for low- and high-

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grade samples. Where this is not practicable, bar
material, and in case of gold and silver this materia
are important. Another source of salting is in us
objectionable if high- and low-grade pots are kept
cupels should be protected from dust, especially
Blank assays should be made occasionally on thes

16. Platinum M

Qualitative tests. To differentiate and deter
considerable equipment and a skilled analyst. M
accurate results, and others only under certain
palladium are the three most important. The first
the last occurring more frequently, though in small
formations. As compared with gold and silver, t
as follows: Melting point: silver, 961°; gold,
1 755°; iridium, 2 300° C. If cupellations are fini
temperature for gold and silver, 5% of palladium,
produce a frosted bead. Frosting can also be pr
chilling just before the finish, but solid litharge
surface. Larger quantities of the platinum metals
ance. HNO_3 dissolves silver and palladium, givi
brown solution, depending on concentration; 10 c
0.15 mg of palladium, in a parting cup, has suffici
against distilled water. When alloyed with silv
though not completely. The solution is only sligh
to colloidal platinum. Gold and iridium are
dissolves silver and slowly attacks palladium; t
regia converts silver to chloride and dissolves p
iridium, except with vigorous treatment or when
ments, except silver, give colored solutions, whic
dark red, depending on the combination and co
precipitates gold from slightly acid solutions, b
monium chloride in concentrated solutions will pr
gold or palladium in its usual condition. Potassiu
gold, gives free iodine; with platinum, brownish
with palladium, a black precipitate soluble in e
delicate. Dimethylglyoxime precipitates palladi
Silver precipitated as chloride in presence of palla
these metals. Platinum metals concentrate in le
usual method of assay, but unless present in suf
parting are apt to be ove-looked. If parting a
hundredths of a mg of palladium will produce a
parted residue is likely to be finely divided. Iridi

Methods of assay. For ores and black
are frequently necessary to yield sufficient m
portions (Art 4 and 5), and lead buttons are c
concentrates are fused with lead, producing
alloy, which is pulverized for sampling and
of the metal content is necessary for good r
only as preliminary. Alloy for cupellation
much gold as platinum metals, and silver 2.
if necessary. Wrap alloy in sheet lead, cu
remain in muffle 10 min after bead has set.
palladium is absent, part with boiling conce
inserting a glass rod. (A glass tube 3 mm
2 mm from the end, leaving a small cavity,
— A yellow or orange solution indicates pall
is not applicable.) Decant and repeat. W

d weigh. If silver ratio has been very high the parted residue will be divided that some may decant with solution; hence, solution must d, filtered on ashless paper, and ignited with main residue. Loss in silver. Inquart with silver 2.5 times the weight of gold, flatten bead in dilute (1 to 1) HNO_3, giving two treatments, boiling for 15 min in lask, washing twice. Repeat inquartation and parting. Anneal and ld, plus iridium, if present. Loss in weight is platinum. When only antity of platinum is present, one inquartation and parting with HNO_3 To test for iridium dissolve the gold in cold dilute aqua regia (HNO_3 1, ater 5, parts); residue is iridium. Results obtained as above are fairly and if a check of pure metals, duplicating the assay, is run at same ors are largely eliminated. When palladium is present, the H_2SO_4 s omitted, and the bead treated with HNO_3 as described. In absence um the HNO_3 is nearly neutralized, and palladium is precipitated with glyoxime, heating for an hour and allowing to stand over night. Filter, nd weigh palladium. When platinum is present, repeat inquartation ing if necessary, dilute the solution considerably, precipitate silver with r, evaporate solution nearly to dryness, add HCl, evaporate again, and Add a saturated solution of ammonium chloride and allow to stand over Filter on ashless paper, wash with 75% alcohol, ignite, and weigh. n may be precipitated as described above. Unless errors have compensults will be only approximate.

11. Copper and Lead

llowing methods for copper presuppose that the metal is soluble in Ores which are not decomposed as described require use of hydrofluoric F) or a fusion.

cyanide method" is not suitable for all ores, but where interfering such as Ag, Zn, Ni, Co, and certain other elements to a lesser extent, nt or present in small amounts, the results are remarkably satisfactory. t useful when a large number of determinations are to be made on fairly ore, and a high degree of accuracy is not required. Success with this depends upon a strict uniformity of conditions, as to quantity of chemd, volume and temperature of solution, and rate of adding cyanide. solution, containing 22 gm c p KCN per liter, or one-half that amount rade ores, is standardized against an average sample of the ore to be a check on the solution being run every few days with the ore samples. ontent of standard sample is accurately determined by electrolytic method.

.5 to 1 gm of ore with 10 c c HNO_3, of 1.42 sp gr, in a 6-oz copper flask, or 20 c c 1 to 1) HNO_3 containing 50 gm $KClO_3$ per liter. Boil until all the lower oxides of re expelled, add 25 c c of water and enough ammonia to give an excess of 5 to ually 20 c c. Cool, titrate to a faint blue, filter, drain (washing unnecessary), titration of filtrate. The end point is not particularly sharp, and a definite ust be selected. For large amounts of work, a dozen burettes will be found t. titrating this number of assays at one time, as it is necessary to allow the rate to settle when the end point is approached, in order to observe color of solu- pensing burettes for reagents are a convenience.

method, for accuracy, is not surpassed even by the electrolytic method y carried out, and, as it requires less expensive and elaborate equipment, the preference here. A solution of sodium thiosulphate containing 19 ter, or one-half that strength for low-grade ores, is necessary. STAND- y dissolving 200 mg of copper, or 100 mg for the weak solution, with 1 to 1) HNO_3, in a 6-oz flask. Boil until lower oxides of nitrogen are

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expelled, add 5 c c of bromine water and co
eliminated. Add ammonia until the soluti
vigorously for a few minutes, or until the an
acid is then added, 2 or 3 c c in excess, and
the solution is boiled. Cool, dilute to 50 c
titrate with hypo until the brown color fades
tion until the solution becomes a decided blue,
the end. (STARCH SOLUTION is made by dissol
c c of boiling water and allowing to cool. La
sion of ordinary starch into boiling water, bo
cool. Starch solutions do not keep well, but
of cassia will preserve them perfectly.)

Decompose 0.5 to 1 gm of ore with 10 to 15 c c
hydrochloric acid in a 6-oz wide-mouthed Erlenmey
pear. Add 6 to 10 c c of concentrated H_2SO_4 and c
then give the flask a circular motion over a free fla
the mouth of flask. Allow to cool, add 20 c c of wa
may be precipitated from this solution, after dilu
pieces of aluminum foil, 1.5 in square (bend corne
opposite directions). After complete precipitation
taken to prevent oxidation and re-solution. Dissol
foil with dilute (1 to 1) HNO_3, and titrate as in s

If a MUFFLE FURNACE is available, the following
phate solution is diluted to 50 c c, brought to boil,
is poured in slowly until the precipitate which forms
supernatant liquid is clear, filter, and wash with
and place in scorifier or clay annealing cup and ig
completely oxidized. Transfer to flask, dissolve
standardizing. The thiocyanate or permanganat
deservedly popular and is extensively used on the

Lead; fire assay. Weigh 10 gm of ore
following flux in 10 or 15-gm crucible: Na
glass, 15; flour, 9 parts. Insert 4 10-d nails
750° C (cherry red), and hold for ½ hour, or un
raise temp to 1 050° C (yellow to light yellow)
tion, which has become more vigorous, ceases,
around nails. This usually requires about
of heat, varying somewhat for different ores,
fore the temperatures and times given abov
Results by this method are in the average 1
other reducible metals they may be very high

12. Tin

Vanning test is unreliable as an assay, on
the retention of foreign minerals. Shipmen
Nigeria, as determined by this method, were
that metal. The test may be of use in ascer
requires a knowledge of the percentage of tin i
GRAVITY of a tin concentrate is a fair index of th
tion is ascertained for a given product a close a

Fire assay (6) is applicable only to a hi
reducible metals. For ores, this involves a p
attendant loss which may reach 30% of the t
concentrate are, however, fairly accurate. T

liberate the tin minerals, taking care to produce a minimum of slime. portion to yield about 10 gm of concentrate, and pan. If the concen- tains oxidizable material, it is roasted and soluble constituents removed g in strong nitric acid. After drying, mix the concentrate, or a 10-gm of it, with 30 gm of pure potassium or sodium cyanide, and place in a having 5 gm of cyanide in the bottom; cover with another 5 gm. Charge or 20 to 30 min at a good red heat (750° C), cooled, and crucible broken. uncollected tin are recovered by crushing slag and panning.

metric assay. Best wet methods are those of J. J. Beringer (7) and ow (8).

-Low method. Ore must be crushed to at least 100 mesh, and portions of 0.2 are weighed. Add to 5 times its weight of sodium peroxide, and fuse in a nickel Should the ore contain readily oxidized material, melted peroxide is allowed to nd the ore gradually added to the hot mass. The crucible is then given a cir- ion over a burner till a clear melt results. Hold the outer surface of crucible with 50 c c of water in a beaker, till cool; then immerse, and when melt has remove crucible, and wash. Transfer beaker contents to a 600-c c flask. Add Cl, using a portion of this to clean the crucible further, and enough water to otal volume of 400 c c. Strips of nickel having a surface of at least 10 sq in are e flask closed with a stopper fitted with a small funnel or Bunsen valve, and boiling until tin is completely reduced; 25 min is sufficient for 40 mg and 1.5 hr g of tin. Add a crystal of calcite (4 or 5 gm), cool in water. Remove stopper el strips after calcite is completely dissolved, and titrate with iodine solution g 11 gm iodine and 20 gm potassium iodide per liter for a rich product, or one- strength for poor material.

ndardize iodine solution, weigh 0.2 gm pure arsenious acid (As_2O_3), and dissolve a 20% NaOH solution, dilute, acidify with HCl, neutralize with sodium carbon- litmus paper as an indicator; then add 20 c c cold saturated $NaHCO_3$ solution te to a blue color, using starch as indicator. The arsenic standard multiplied by ves value in tin. Iodine solution should be standardized at least once a week.

ering elements are bismuth, titanium, and tungsten. BISMUTH is precipitated during reduction and should be filtered out, the reduction being completed with kel. To eliminate effects of TITANIUM, add the weighed ore, small portions at a 5 gm of $KHSO_4$ melted in a platinum or quartz crucible, and fuse for 5 min. in 60 c c water and 7 c c H_2SO_4. Filter, dry, ignite, and fuse residue with sodium as described. TUNGSTEN, if present, gives a blue precipitate during the reduc- nickel. This is filtered off, and solution again reduced before titration. When and titanium occur together, acid sulphate fusion is described above, and washed residue treated with m carbonate solution, filtered, and washed. This ignited, fused with sodium peroxide, and so on.

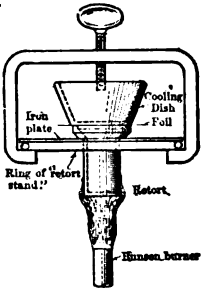

Fig 6. Whitton's Apparatus for Mercury Assay

13. Mercury

ton's apparatus affords results less accurate ose obtained by wet methods, but is better for the technical assayer. Illustration of appa- ig 6) as manufactured by Braun Corporation, eles, is self explanatory. Take an amount of aining not over 50 mg of Hg (0.15 to 2 gm), mix m of iron filings sized between 50 and 80 mesh, m grease and dried at a high temperature; add re as a cover. Clamp in place silver foil and dish filled with water. Heat retort with a low r 15 min and allow to cool. Remove silver foil, gh it; gain in weight is mercury. The deposit should be white, and, after g, may be removed by volatilizing, when the foil is again ready for

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14. Antim

Fire method, fusing 10 gm ore mixed with 4
approx results by compensation of errors, but is

Permanganate method (4). DECOMPOSITIO
HCl, followed by small additions (0.2 gm) of K
decomposed. Better results are sometimes obt
with 15 c c HNO_3, evaporate to 4 c c, add 20 c c
contains oxides, insoluble antimony (Sb) is like
fuse residue or ore with 15 times its weight of
preferably nickel, to complete decomposition (5
cular motion while cooling, thereby causing melt
sides. Place crucible in beaker of water and
crucible, wash well, using dil HCl, if necessary, a
on insoluble residue, add resulting solution to th

SEPARATION. Add 3 gm tartaric, dilute to 5
400 c c, pass H_2S until sulphides are completely
water. Test filtrate by passing more H_2S. Un
sulphide, wash from filter into beaker without
twice with hot alkaline sulphide solution, appro
in water or passing H_2S through a NaOH or
paper, and wash with dilute alkaline sulphide
Sb, and Sn from other sulphides. Acidify filtra
water. If As is absent, treat antimony sulphide
wash precipitate from paper into beaker, using a
mum of wash water. Dissolve sulphides in HC
and transfer to pressure bottle. (Clamp-top be
the volume of conc HCl (sp gr 1.2), saturate c
water, bring water to boiling, and allow to stand
ordinary flasks may be used; after saturating l
allow to stand cold over night. Filter out arse
597). Wash with HCl diluted 2 to 1, and satura
volumes, and complete precipitation of Sb with
H_2S water. Test filtrate by passing more H_2S.
flask, removing traces of precipitate from paper
5 c c of conc H_2SO_4 and 3 gm $KHSO_4$. Heat over
until water is expelled and precipitate complet
and slowly add 125 c c H_2O, and cool.

TITRATE with $KMnO_4$ solution. The end-p
containing less than 30% Sb, use a solution of a
over 30%, use double that strength. To STAND
oxalate for the weak solution, or 0.4 gm for th
10 c c strong H_2SO_4. Heat to 70° to 80° C, an
first additions are not decolorized readily and m
the reaction has started the permanganate solut
Sodium oxalate value multiplied by 0.8963 giv
to standardize with pure Sb when this can be
H_2SO_4 and $KHSO_4$, as described for antimony s

Iodide method (10). Final titration is mo
rately, done with KI and hypo. Dissolve the
small additions of $KClO_3$. Boil off 25 to 50%
to a constant boiling strength. Now add enou
of 90 c c and dilute to 600 c c with cold water fro
going steps are important.) Add 3 gm KI. Al
hypo solution, adding starch indicator near fini
and Sb proceeds slowly to completion, and if hy
color may be completely discharged, only to reap
that the blue color gradually fades to colorless, wh
Hypo solution is preferably standardized against p
for antimony sulphide. The solution may be sta
results by 0.9549, not by the theoretical factor 0.

15. Coal

mate analysis of coal (9) requires certain standard conditions, as the is arbitrary and results vary with procedure. Crush sample to 0.25-in l if moisture content is likely to change, place a sample in stoppered or a separate moisture determination. The remainder should be air-eating moist sample, if desired, to 70° C, and allowing to cool in the air. portion to 80 or 100 mesh and place in stoppered bottle.

ture. Place 1 gm in crucible and heat to 104° to 107° C for 1 hr. Cool cator, and weigh covered.

ile and combustible matter. Heat 1 gm for 7 min in a 20 to 30-gm cov-tinum crucible, supported by a triangle 6 to 8 cm above top of a Bunsen aving a flame 25 cm high. This test may be made in a muffle at a moderate (950° C), using a clay or porcelain annealing cup, though results will not be l. Moisture must be subtracted from loss in weight thus ascertained.

Burn off the carbon completely from sample used for moisture deter-n, using a low heat at the beginning. This operation may be conducted in um crucible over a Bunsen, or in a clay or porcelain cup in a muffle. If, rying, the sample loses an appreciable amount of water, a determination be made on the original moisture sample, and results corrected.

this is necessary, and percentage of moisture is not over 10%, correction may be l, though not in accordance with the standard method, by weighing out 1 gm e percentage loss of weight in air-drying. Results may then be pointed off in % EXAMPLE: Loss of wt in air-drying, 2.5%. Wt of sample for determination, n. Ash. — Wt of ash, 0.0975 gm; percentage of ash, 9.75%. Moisture. — Loss 0275 gm; moisture in air-dried sample, 2.75%.

16. Laboratory Equipment for Gold and Silver Assays

wing list includes equipment and reagents necessary for 500 crucible Prices are approximately those in the large cities of the U S. Quantity s required will vary with character of ore assayed and the charge used. re estimated mainly for acid ores on 0.5 A T samples, with excess PbO Prices in column 1 are for higher-grade equipment.

olene combination furnace, capac six 20-gm crucibles and 24 1.25-in cupels.		
out 10 assays per hr on 0.66 gal gasolene. Gasolene at 15¢ to 50¢ per gal		$27.50
olene burner for above, 2-in		13.50
k outfit		20.00
gs, crucible		0.80
gs, scorifier		0.80
gs, small for parting cups		0.80
gs, cupel		0.50
ffle scraper		0.50
ole Assay molds	$2.40	1.80
il		1.00
nmer, 2-lb, for Pb buttons		0.30
el tray		0.75
d trays, 24-hole		1.00
ton pliers		0.40
ton brush		0.40
rosene or gasolene stove for parting	10.00	4.00
estos stove mats		0.40
ets of asbestos, 1/16-in		0.75
rcelain parting cups, 1.5-in		5.25
nd cupel molds, 1.25 and 1.5-in		3.00
ller's cupel machine, dies 1.25 and 1.5-in	22.00	
Battersea soft-clay 20-gm crucibles		6.00

(these crucibles will generally stand 5 or more fusions with acid slags, fewer with basic)

15 Scorifiers, 2.5-in, Bartlett shape; 10 Scorifiers, 3
3 Battersea annealing cups, A
3 Parting flasks, 50-c c
1 Blowpipe
1 Alcohol lamp
6 5-in Roasting dishes
1 Still and condenser
1 Demijohn for distilled water
3 ft 5/8-in Glass tubing, 1.5-mm wall
2 ft 3/16-in Glass tubing, 1.25-mm wall
3 ft 3/8-in Rubber tubing
2 Pinch cocks
3 32-oz Wash bottles
2 6-in Watch glasses
2 2-in Watch glasses
12 Test tubes, 6 by 3/4-in
1 Test-tube brush, 9 by 3/4-in
1 500-c c graduated cylinder
1 Nest beakers, 100 to 600-c c
1 Funnel, 6-in
1 Funnel, 1.5-in
2 5-pint Acid bottles, empty
1 pkg Filter paper, 8 in
1 oz c p Ag foil
40 lb Soda ash (Na_2CO_3)
50 lb Litharge (PbO) silver-free
10 lb Borax glass ($Na_2B_4O_7$)
3 lb Niter (KNO_3)
2 lb Silica (SiO_2)
3 lb Test lead, silver-free
1 lb Lead foil, silver-free
40 lb Bone ash
14 lb Nitric acid (HNO_3) c p
1 lb Hydrochloric acid (HCl) c p
1 lb Sulphuric acid (H_2SO_4) c p
1 lb Ammonia (NH_4OH)
1/4 lb Charcoal, powdered
6 Charcoal sticks for blowpipe
1/2 lb Reddle, for marking
1 Small ore crusher
1 Buckboard and muller, dressed surface
1 Jones sampler, 8-in
or ordinarily riffle sampler, 9-in
1 Nest sieves, 20–40–60–100-mesh, 8-in, tin
1 Sieve, 10-mesh, 12-in, wood frame
2 Brushes, 4-in for buckboard
2 doz Tin sample pans, 6-in
500 Paper sample bags, 4 by 7-in
1 Scoop, 4.5 by 5-in, flat
2 7-in Steel spatulas
1 Horn spoon
1 Camel's hair brush, flat, 1.5-in
1 Camel's hair brush, pencil
1 yd Rubber mixing cloth
1 Piece of surfaced mixing cloth, 50 by 50-in
1 Spring balance for weighing stock flux
1 Button balance
1 Pulp balance
1 Set assay and gram weights
1 Set button weights and riders, 1 mg to 1 gm
Totals

STANFORD LIBRARIES

17. Assay Supply Houses

. W. H. Hoskins, 81 S. Clark St. E. H. Sargent.
Denver Fire Clay Co. Mine & Smelter Supply Co.
. Mine & Smelter Supply Co.
eles. The Braun Corporation.
rk. Eimer & Amend, 205 Third Ave. Mine & Smelter Supply Co, 42 Broadway.
e. Mine & Smelter Supply Co.
ncisco. Braun-Knecht-Heimann Co.
Braun-Knecht-Heimann Co, 205 Colman Bldg. Stewart & Holmes Drug
07 Third Ave.
. C. M. Fassett & Co, 209 Wall St.
s. Henry Heil & Co.
City. Hoffman-Pinther & Bosworth, Calle Nuevo.
Baird & Tatlock, 14 Cross St, Hatton Garden.
Maison Wiesnegg (P. Lequeux), 64 Rue Gay-Lussac.
Vereinigte Fabriken für Laboratoriumsbedarf, Scharnhorst Str 22.

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SECTION

TESTING O

BY

J. E. CLEN

METALLURGICAL ENGINEER WITH CHAS. BUT

AND

EDWARD K.

ASST PROF OF MINING, COLUMBIA SC

Art		Page	Ar
1.	General Considerations	1776	7.
2.	Preparation of Sample	1777	8.
3.	Screen Tests	1777	9.
4.	Classifier Tests	1783	10.
5.	Concentration Tests	1785	
6.	Flotation Tests	1789	

Note. — Numbers in parentheses refer to bibl

1. General Cons

Importance of testing. Before erecting necessary to determine what process will extr profit; this is done by tests on an average desirable to test separately the different class it is practicable to give different treatments economy in the matter of testing is of freq companies have suffered loss by erecting m ores; on the other hand, in some cases large siderable expense to determine points which c rately, and at much less cost and trouble, on

Scale of testing. In general, it is desira a gradually increasing scale. The first test well equipped chemical laboratory, on 1 or of 100 grm or less. The indications obtain subsequent procedure. Results may be co ore per charge, conducted in improvised or together by intelligent mechanics and carpe on ½ to 1 or 2-ton lots, in a small experimen on a large scale may be imitated in most erect such experimental plant may require is complicated. Finally, trial runs may b conditions, in a small plant erected for the voted to experimental work.

2. Preparation of Sample

racy. For precautions necessary to obtain an accurate sample, see Art 31, and Sec 30, Art 3. It need only be pointed out here that results equent tests are misleading unless care has been taken to make sample presentative of material which it is proposed to treat. This applies not the percentage of valuable metal in the ore, but also to the physical er and chemical composition of both valuable and worthless minerals. nature of the ore is liable to vary in different parts of a mine, and arly at different depths, it may often be necessary to test separate representing these various classes of ore. If the mine development already disclosed variation in character of ore, the probability of a ced alteration near ground-water level should not be overlooked.

uction of sample. In reducing a portion of ore for experiment, it is l to avoid treating it in any way which might alter its character; that sample must be so prepared that its behavior in subsequent tests will y indicate the behavior of the ore when similarly treated on a working The process of grinding a sample introduces an appreciable amount of the grinding parts of the machine are of steel. If the presence of this iron would materially affect subsequent tests, it may be removed by ic separator, provided the original ore contains no magnetic material. shing samples, those machines are best which permit quick and thorough g, to avoid danger of enriching a sample of poor ore.

ssible, the sample should be quartered, mixed and reduced to the re- size without drying. Where this is impracticable, the drying should be the lowest possible temperature, and any roasting of sulphides or de- on of hydrated minerals must be carefully avoided. Ores which rapidly o oxidation or other changes, on exposure, should be stored under cover, mple, in closed tins, from which the portions required for small-scale nay be taken immediately before beginning operation. Precautions oxidation are especially necessary with ores on which it is intended to t flotation tests.

3. Screen Tests

e principal objects of screen testing are: (*a*) To judge the character- f a given crusher or system of crushing; for concentration works, in gen- at machine is to be preferred which yields the minimum proportion of zed material, fine slime being particularly objectionable. (*b*) To ascer- lative richness of different sized products of a given ore; the coarsest nts, after a preliminary crushing, may prove to be so nearly barren that n be discarded as waste, while the finest may be so rich as not to require concentration; in same manner, the size below which it will be uneco- l to crush will be indicated. (*c*) To ascertain mechanical efficiency of a rusher; this is based on theory that work performed is in some manner tional to reduction in average diameter of ore particles, after crushing; ties are not agreed as to exact relationship (1).

en openings, mesh, etc. Test is conducted by screening a weighed of ore, sampled in such manner as to insure an accurately representative e of sizes (Sec 28, Art 31), on a set of sieves arranged in order of descend- e of opening. AVERAGE SIZE of a product is then considered to be the etical mean of the diameters of opening in the two screens, through the product has passed and on which it rests. Theoretically, round-hole would be preferred, as yielding more uniformly sized products than -hole or slotted screens, but for practical reasons, since the finer si[illegible]

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Table 1. Diameter o

No of wire (*d*)	5	6	7	8	9	10	11	1
Diam, mm	5.26	4.88	4.50	4.11	3.77	3.43	3.06	2.(
Mesh, wires per lin in								
1	20.14	20.52	20.90	21.29	21.63	21.97	22.34	22.
2	.....	7.82	8.20	8.59	8.93	9.27	9.64	10.
3	.....	.....	.....	.....	4.70	5.04	5.41	5.
4	.....	.....	.....	.....	.....	.....	3.29	3.
5	.....	.....	.....	.....	.....	.....	.....	..
6	.....	.....	.....	.....	.....	.....	.....	..
7	.....	.....	.....	.....	.....	.....	.....	..
8	.....	.....	.....	.....	.....	.....	.....	..
9	.....	.....	.....	.....	.....	.....	.....	..
10	.....	.....	.....	.....	.....	.....	.....	..
12	.....	.....	.....	.....	.....	.....	.....	..
14	.....	.....	.....	.....	.....	.....	.....	..
16	.....	.....	.....	.....	.....	.....	.....	..
18	.....	.....	.....	.....	.....	.....	.....	..
20	.....	.....	.....	.....	.....	.....	.....	..
22	.....	.....	.....	.....	.....	.....	.....	..
24	.....	.....	.....	.....	.....	.....	.....	..
26	.....	.....	.....	.....	.....	.....	.....	..
28	.....	.....	.....	.....	.....	.....	.....	..
30	.....	.....	.....	.....	.....	.....	.....	..
35	.....	.....	.....	.....	.....	.....	.....	..
40	.....	.....	.....	.....	.....	.....	.....	..
45	.....	.....	.....	.....	.....	.....	.....	..
50	.....	.....	.....	.....	.....	.....	.....	..
55	.....	.....	.....	.....	.....	.....	.....	..
60	.....	.....	.....	.....	.....	.....	.....	..

must be made of woven-wire cloth, square hole screens. Use of the word MESH (meaning numbe discouraged in connection with screen tests, sinc ing unless diameter of the wire is also stated. Tal shows ranges in size of opening corresponding to wire. In reporting a screen test, it is always pr

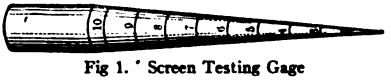

Fig 1. ' Screen Testing Gage

tained, if coarse, by measuring a considerable n with calipers or the instrument shown in Fig 1 microscopic examination (4) with a micrometer e enlarged micro-photograph (5). A fine screen c brated by measuring diameter of largest grains p it, using a machinist's micrometer fitted with br pieces (6). Table 2 (7) gives relationships be and diameter of wires, diameter and area of ope centage of open space per unit of screen area, on same sized wires and that number of wires per i

Mesh and Diameters of Wire, in Millimeters (*b*)

21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	36	37
0.81	0.73	0.66	0.58	0.52	0.46	0.44	0.41	0.38	0.36	0.34	0.33	0.30	0.26	0.24	0.23	0.22

eters

....	....	....	....	....	....	....	....	....	....	....	....	....	....	....	....	....
....	....	....	....	....	....	....	....	....	....	....	....	....	....	....	....	....
....	....	....	....	....	....	....	....	....	....	....	....	....	....	....	....	....
5.55	5.62	....	....	....	....	....	....	....	....	....	....	....	....	....	....	....
4.27	4.35	4.42	4.50	....	....	....	....	....	....	....	....	....	....	....	....	....
3.43	3.51	3.58	3.64	3.71	....	....	....	....	....	....	....	....	....	....	....	....
2.82	2.90	2.97	3.05	3.11	3.17	....	....	....	....	....	....	....	....	....	....	....
2.37	2.45	2.52	2.59	2.66	2.72	2.74	....	....	....	....	....	....	....	....	....	....
2.02	2.10	2.17	2.24	2.30	2.36	2.38	2.41	....	....	....	....	....	....	....	....	....
1.73	1.81	1.88	1.96	2.02	2.08	2.10	2.13	2.16	....	....	....	....	....	....	....	....
1.31	1.39	1.46	1.53	1.60	1.66	1.67	1.70	1.73	1.76	....	....	....	....	....	....	....
1.01	1.09	1.16	1.23	1.30	1.35	1.37	1.40	1.43	1.46	1.48	1.49	1.51	1.55	....	....	....
0.78	0.85	0.93	1.00	1.07	1.13	1.15	1.18	1.21	1.23	1.25	1.26	1.29	1.32	1.35	....	....
....	0.68	0.76	0.83	0.89	0.95	0.97	1.00	1.03	1.05	1.07	1.08	1.11	1.15	1.17	1.18	....
....	....	0.61	0.69	0.75	0.81	0.82	0.86	0.89	0.91	0.93	0.94	0.97	1.01	1.03	1.04	....
....	....	....	0.57	0.64	0.69	0.71	0.74	0.77	0.80	0.82	0.83	0.85	0.89	0.91	0.92	....
....	....	....	....	0.54	0.59	0.62	0.65	0.68	0.70	0.72	0.73	0.76	0.80	0.82	0.83	....
....	....	....	....	....	0.52	0.54	0.57	0.60	0.62	0.64	0.65	0.68	0.71	0.74	0.75	....
....	....	....	....	....	....	0.47	0.50	0.53	0.55	0.57	0.58	0.61	0.64	0.67	0.68	....
....	....	....	....	....	....	....	0.43	0.46	0.49	0.51	0.52	0.55	0.58	0.60	0.62	....
....	....	....	....	....	....	....	....	0.34	0.37	0.39	0.40	0.43	0.46	0.49	0.50	....
....	....	....	....	....	....	....	....	....	....	0.30	0.31	0.34	0.37	0.40	0.41	0.42
....	....	....	....	....	....	....	....	....	....	....	....	0.26	0.30	0.32	0.34	0.35
....	....	....	....	....	....	....	....	....	....	....	....	....	0.24	0.27	0.28	0.29
....	....	....	....	....	....	....	....	....	....	....	....	....	....	0.22	0.23	0.25
....	....	....	....	....	....	....	....	....	....	....	....	....	....	0.18	0.19	0.21

in = 25.4 mm; 1 mm = 0.0394 in.

Table 2. Relationships of Screen Factors

		Disch area a, diam wire y	Disch area a, aper-tures z	Disch area a, No of holes x	Diam wire y, aper-tures z	Diam wire y, No of holes x	Aper-tures z, No of holes x
arge area,	$a =$	.....	.....	.....	$\left(\frac{10z}{y+z}\right)^2$	$100(1-xy)^2$	$100z^2x^2$
r wire, deci-of inch,	$y =$	.....	$\frac{10z}{\sqrt{a}} - z$	$\frac{10-\sqrt{a}}{10x}$	.....	.....	$\frac{1-xz}{x}$
r of aperture, als of inch,	$z =$	$\frac{y\sqrt{a}}{10-\sqrt{a}}$	.....	$\frac{\sqrt{a}}{10x}$	.....	$\frac{1-xy}{x}$	.....
r of holes n in.	$x =$	$\frac{10-\sqrt{a}}{10y}$	$\frac{\sqrt{a}}{10z}$	.....	$\frac{1}{z+y}$	.....	.....

er, in decimals of an inch, of a steel wire is approximately $y = .00315\sqrt{w}$, = weight in milligrams per lin in of wire.

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Standard screens. For convenient compa
ard series of screen openings are desirable;
SCREEN RATIO (quotient obtained by dividing
next finer screen), sometimes modified by pr
manufacture of screens. Best known series
harz, Institution of Mining and Metallurgy,

Rittinger series (9) has the constant ratio $\sqrt{}$
in two successive screens, therefore, area of large
testing work on finer sizes of ore, this series has
between the (approx) 20 and 200-mesh screens
this ratio, see Sec 28, Art 16.)

Richards series (also known as "double Ritt
with 1 mm as starting point. For testing work
fication tests) this series gives 14 sizes between (a
ore, it is unnecessarily complete.

Tyler series (10) is based on Rittinger ratio, b
with opening of 0.074 mm, which has been adop
testing cement, etc. The 100-mesh (0.147 mm
also come within the specifications of the Bur
testing in the finer sizes, 3 screens are interpola
0.208 mm and 0.074 mm.

Table 3 compares diameter of opening, i
screen series:

Table 3. Diameters of Opening, in M

Rittinger—$\sqrt{2}$=1.414		Tyler—$\sqrt{2}$=1.414		I M
Mm	Approx mesh †	Mm	Approx mesh †	
32.00	.....	26.67	.....	...
22.63	1	18.85	.....	...
16.00	1	13.33	.....	...
11.31	2	9.42	.....	...
8.00	2	6.68	3	2
5.66	3	4.70	4	1
4.00	4	3.33	6	1
2.83	5	2.36	8	1
2.00	7	1.65	10	0
1.41	10	1.17	14	0
1.00	15	0.83	20	0
0.71	18	0.59	28	0
0.50	25	0.42	35	0
0.35	35	0.295	48	0
0.25	50	0.208	65	0
0.18	70	*0.175	80	0
0.12	100	0.147	100	0
0.09	130	*0.124	115	0
0.06	200	0.104	150	0
0.04	300	*0.088	170	0
.....	.....	0.074	200	0

* Interpolated, on ratio $\sqrt[4]{2}$ = 1.1
† Wires per linear inch.
‡ Equivalent to 1 in.
1 mm = 0.0394 in; 1 in = 25.4 mm

rz series is designed for general application; ratio is $\sqrt[5]{10}$; whence, in a series screens, the diam of smallest hole will be 0.1 that of the largest.

screens, ratio	1.75.....	1,	1.8,	3.1,	5.4,	10			
" "	1.60.....	1,	1.6,	2.5,	4.1,	6.5,	10		
" "	1.47.....	1,	1.5,	2.2,	3.2,	4.7,	6.9,	10	
" "	1.40.....	1,	1.4,	1.9,	2.7,	3.8,	5.3,	7.5,	10
" "	1.33.....	1,	1.3,	1.8,	2.4,	3.2,	4.2,	5.6,	7.5, 10

uch series can be extended in either direction by shifting decimal points.

ution of Mining and Metallurgy series (11) is not based on a fixed screen ratio, approximating the Rittinger series, but is composed of screens commonly used in and readily obtainable. Wires are of same diam as holes; whence, diam of hole ÷ 2 × number of wires per in (e g, holes of a 20-mesh screen are ¼0 in diam); also, openings is 25% of screen area. Screens made to this specification are limited sizes (2.5 mm and less) because of mechanical difficulty of weaving thick wire; the finer end of the series, also, the wire becomes so fine as to lack durability.

er series (12) is based on fixed ratio $\sqrt{2} = 1.26$; whence average volume of a ain in any product will be double that of a grain in next finer product. Also, opening in one screen is twice that of the third following screen. This series eveloped with equal ease in either English or metric units, since it happens that, with 1 in, the fifteenth screen is 0.0394 in, the legal equivalent of 1 mm.

polation of screens. In any series of screens having a fixed geomettio, the following relation exists (13): $S = \sqrt[n+1]{A \div B}$, where S = ratio, mber of screens to be interpolated, A = diam of coarsest hole, B = diam t hole. For example, to interpolate 4 screens between 2 in and 0.25 in, ll be $\sqrt[5]{8} = 1.516$, and the diams will be 0.25, 0.38, 0.58, 0.87, 1.32, 2 in.

autions in test screening. In passing the weighed quantity of crushed ough successive screens, care must be taken to avoid violent rubbing, might enlarge the openings or tear the screen. With fine screens, rable time is required to get the particles to pass, even though crushed ıtly fine; in most cases, long continued shaking, at the same time tape edge of the screen, or lightly rubbing the material over its surface lat cork or rubber stopper will accomplish required separation. No hard as a metal washer or disk, should ever be used for this purpose, as it evitably crush particles which would not otherwise have passed the and will quickly wear holes in wire cloth of fine mesh. U S Bureau of rds, for cement testing, specifies a shaking and gently bumping motion, ıt 200 strokes per min, to be continued until not more than 0.05 gm (200-mesh screen) in 1 min. Investigation has shown (14) that a somewer shaking, about 150 per min, puts a larger ultimate percentage of a ample through the screen; also that use of washers or steel balls in the ields the least reliable results, due to uncertainty of stopping point. nature of material allows, fine screening may be more rapidly and accuccomplished by a jigging motion under water, assisted by a jet of water; ts are settled and dried, after decanting surplus water. If screened dry, ts should be collected in a tightly fitting pan, and screen closely covered, d loss by dusting. A COMBINATION METHOD (15) is to wash and jig the in a pan of water, in the finest screen it is proposed to use for the test, bly using a screen having a metal frame. Fine material is settled, dried, ighed. Material on screen is dried, in the sieve, at a moderate temp oid decrepitation), and then screened in ordinary manner on the coarser This method is more accurate and requires less time for a complete nce stopping point is more distinct on washed sands.

ghing and assaying screen products. Pulp balance is accurate enough ghing. In general, the coarser products should be re-ground to suit

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fineness for accurate assay or analysis. In so
merely as to physical nature of the crushed mat
Screen tests are often made to compare distrib
and after treatment, for example, the heads an
nide percolation. Comparison of assays in th
percentage extraction in each sized product, a
ing or further classification of the ore will be b

Reporting results of screen tests. Muc
results of screen tests, aside from that due to
instead of "mesh." For uniformity, the me
ample is recommended:

Size, mm	Weight, grams	Pe
Over 0.295	30	
0.295 to 0.208	50	
0.208 " 0.147	60	
0.147 " 0.104	20	
0.104 " 0.074	16	
Below 0.074	24	
	200	

Each cumulative per cent thus shows what p
coarser than the screen on which it would rest

Plotting screen tests (27). For conveni
may be plotted on coordinate paper, taking c
and corresponding screen openings as abscissa
in 3 ways: (*a*) equal spaces arbitrarily taken
(*b*) geometric scale, the spaces diminishing in
screen openings; (*c*) logarithmic scale of the
has advantages of allowing a wide series of siz
of paper, and also permitting more accurate pl
nate sheets of thin paper ruled according to m
Tyler Co, Cleveland, Ohio, to correspond with
screens. An incidental use of plotted curves is
designed to divide a given crushed ore in s
product will form any desired proportion of th
plotting screen tests for determination of crush

Calculating assay results. The following
how assays are calculated so as to indicate pe
and also the percentage of the total value, in e

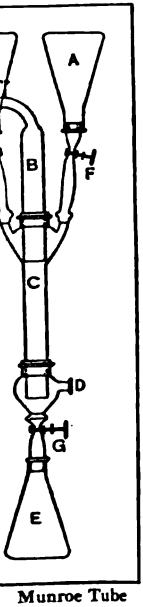
Taken for test, 500 grm. Silver @ 50¢ per

Product	Actual weight	Per cent of total weight	Assay of product per ton of 2 000 lb			
			Gold	Silver	Total	Gol
Mesh	Grams	%	$	Oz	$	%
Over 10	250	50	11.54	0.18	11.63	40
10 to 20	113.5	22.7	14.83	0.18	14.92	23
20 " 60	53	10.6	15.06	0.28	15.20	11
60 " 120	25	5.0	15.66	1.13	16.18	5
120 " 200	14	2.8	16.27	2.76	17.65	3
Below 200	44.5	8.9	24.72	3.02	26.23	15
	500	100	Computed average ass			

4. Classifier Tests

ts. (*a*) To make sorted (classified) products for further tests on sand concentrators. (*b*) To remove slime from sand, for separate treatment. ascertain relative richness of different sorted products (as in screen . For computations and design of commercial classifiers, see Sec 28, 18. Laboratory tests can be made as follows: Beaker, tube, model inted boxes, or spitzlutten.

r classification (16). Fill a tall beaker with water to a depth of Put in 10 grm of the ore to be tested, stir thoroughly and allow to , say, 40 sec; then pour off the water with its suspended solids quickly ully into a pan. Repeat this operation until the liquid to be decanted A grain which will not settle in 40 sec obviously has a falling ve- less than 2.5 mm per sec. Retaining residue of same ore, fill beaker the mark, stir and allow ore to settle for 20 sec; pour off the liquid econd pan and repeat until all ore settles within the specified time. peration for periods of 10, 5, and 2.5 sec. Allow each decanted portion pour off clear water, and dry and weigh solid matter; also that which n 2.5 sec. The settling velocities of the products made in above man- be respectively 2.5, 5, 10, 20, 40, and above 40 mm per sec; other ve- bviously can be attained by varying depth of water or settling time.



Munroe Tube Classifier

Tube classification (16). A tube classifier, made of glass, is shown in Fig 2. (Designed by H S Munroe, and manufactured by Eimer & Amend, and Emil Greiner, N Y). Joints are made water-tight by short sections of large rubber tubing. To make connection between a rubber tube and neck of a flask, a short section of brass pipe is put inside end of rubber tube and the latter forced firmly into flask. A space of ⅛ in must be left between bottom end of the tube *B* and point of contraction in tube *C*. Nipple *D* is connected by tube to dial cock on water main, and overflow from *B* is conducted by tube to pails or pans. Knowing inside sectional area of column, compute volumes of water necessary to give rising currents of any desired velocities ($q = av$). If computations are in metric system, a liter graduated jar can be used for measuring volumes.

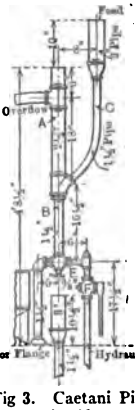


Fig 3. Caetani Pipe Classifier

A weighed sample of ore is divided about equally between the two flasks *A*. Enough water is added to moisten ore thoroughly, with shaking; bottles are then filled with water, and put into position for feeding, the cocks *F* being closed. The rubber tubes must be full of water both above and below *F*, because an air bubble prevents iscbarge of ore. Having adjusted flow of water up the column to desired velocity, are slightly opened to allow ore to drop slowly. Light material will be carried over, e heavier will fall through rising current and into flask *E* at bottom. When all ore upper flasks, a few minutes are allowed for sorting column to clear partially, cur-

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rent is then shut off and a little more time allowed
Cock *G* is then closed, flask *E* removed, and its c
This last operation may be simplified by having at l
when about half the ore has fallen into first flask, it
closing *G* temporarily. Two flasks *E* may then l
flasks *A*. Current is now adjusted for the next
repeated. While working with velocities up to, say
the entire overflow, water and solids, in pails which
of time to permit solids to settle perfectly; with the
to pans in which solids will settle while water ove
each velocity is caught separately, dried, and weig

Caetani pipe classifier (17), in practical use at
fication of Munroe glass classifier, and can readily b

Richards tube classifiers, for laboratory use;
(27), this Sec.

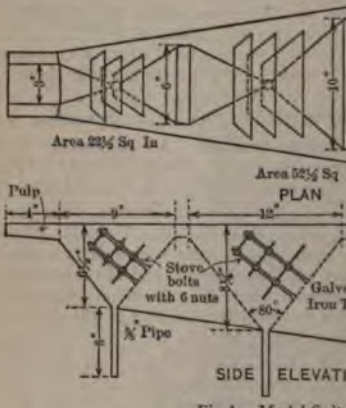

Fig 4: Model Spitz

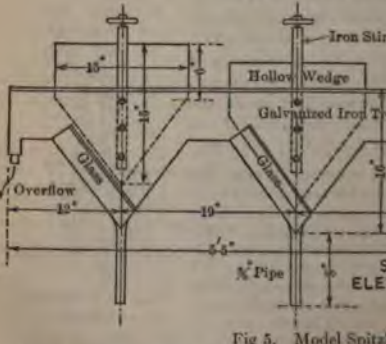

Fig 5. Model Spitz

Model cones, pointed boxes, and spitzlutter machines, and can usually be made by a compete and brass fittings are better than plain iron, to p

Two forms used in the Columbia School of Mines are shown in Fig 4 and 5. In ng rising velocities in inclined channels, measurements of sectional area should horizontally, not normal to walls of channel, since it is the vertical falling velocity s that is under consideration. A sorting tube underneath the apex of cone or box is an essential feature if well classified products are desired. A convenient by which to draw off classified products continuously, while maintaining a rising in sorting tube, is to use a 2-necked Wolff bottle, joining one neck to bottom of tube (being careful to avoid constriction) and connecting other to water cock. elocity in sorting tube should be adjusted to about one-fourth that in free-set- mber above it, to compensate for hindered settling conditions (Sec 28, Art 17).

5. Concentration Tests

d-picking often affords valuable information, even when not considered icable method of treating a given ore under existing conditions. It may ied to ore as fine as pea sizes; for example, to determine proportions of d included mineral in jig middlings. For satisfactory hand-picking, ore be sized between rather close limits, and should be clean; washing also out distinctive color or luster of some minerals (27).

icability of hand-picking on a commercial scale may be tested by attempting the following products, or so many of them as practicable (16): (*a*) RICH MIN- t for market or for metallurgical treatment. These will include not only such as galena, blende, chalcopyrite, etc, nearly or quite pure, but also rich material ossibly can be treated better by some metallurgical process than by mechanical for example copper carbonates, silver chloride, finely disseminated silver ore,) Rich ore, with COARSE DISSEMINATED MINERAL, for coarse crushing and jigging. ntains minerals not easily separated by gravity methods, several classes may be this rich ore. For example, pyritiferous lead ore should be separated from lead aining blende, and ore with a barite gangue from ore with a quartzose or calca- angue. (*c*) POOR ORE, with coarse disseminated mineral, suitable for coarse and jigging, but less rich than the above. (*d*) FINE DISSEMINATED ORE, usu- r, so-called "stamp-rock," in which the useful mineral occurs in such small par- at exceedingly fine crushing will be necessary to liberate it. There are ores in he valuable mineral is so finely disseminated that mechanical treatment is im- ble. (*e*) BARREN WASTE, or material that does not seem to be mineralized. classes may sometimes be made of this, according to gangue minerals or rocks in or when a difference of color, texture, or other characteristic indicates a possible e in richness. Assays of these different classes of barren or seemingly barren will show which may be thrown away, and which should be included with the mechanical treatment. Finally, inspection of the different products may show ne classes are present in small quantity only, and that perhaps several sorts may ined without disadvantage.

d jigging, for testing ore ranging in size from 10 to 2 mm, requires only of water and a few small screens of differing mesh. Before jigging, the ould be sized fairly closely (16).

bout 2 in depth of sized ore into a sieve having mesh fine enough to retain it, and several minutes under surface of water, with a long, slow stroke for coarser sizes of a shorter and quicker stroke for finer sizes. A quick down stroke combined with r up stroke will be found most effective. Care should be taken to keep sieve level avoid any horizontal or overturning movement of the mass of ore. When tailings to be clean, upper layer may be scraped off and replaced with an equal amount and jigging resumed. Middlings and concentrates may be allowed to accumu- sieve, unless these layers become inconveniently thick. Main object of this pre- y jigging is to produce tailings as poor as possible. When whole sample has us treated, accumulated concentrate is first cleaned by careful jigging, aided by cking if necessary. Skimmings from this cleaning of concentrate are added to ddlings first obtained. All middlings are now rejigged and reduced to smallest e bulk, by separating as much concentrate and tailing as possible. It is sometimes e to make two or more grades of middlings, the richness of which may be tested

separately to determine the effect of combining
or with this. Assays of products, and close
different sizes as are tested, will indicate maxi
be used.

Machine jigging. Experimental jigs for tes
are made by the Denver, Colo. and
the are also equipment for testing laborato
using that these machines by using jigging [26]

Panning. The gold pan, or prospector'
of testing gravels, or finely
For [illegible] must small-scale concen
are a which the useful minerals has a hig
panning.

[illegible] sufficient gravel or c
[illegible] say [illegible] In washing wo
[illegible] are [illegible] that the result is
[illegible] [illegible] [illegible] [

heavy mineral, the latter is drawn off through stop-cock. Brögger's apparatus, ng of two bulbs in tandem, with stop-cock between, is better than simple funnel.

ions available for this purpose are: SONSTADT; H_2O sol of KI and $HgCl_2$ (sp gr diluted to any desired sp gr with H_2O. THOULET; 210 gm KI, 260 gm HgI_2, d in 25 cc H_2O (sp gr 3.196); can be diluted with H_2O. KLEIN; saturated H_2O sol ium borotungstate (sp gr 3.281); must be prepared by a chemist, and is expensive; diluted with H_2O. ROHRBACH; 100 gm BaCl, 130 gm HgI_2, dissolved in 20 cc temp of 150° to 200° C on an oil bath (sp gr 3.58); can not be diluted with H_2O, nstable. BRAUN; commercial methylene iodide (sp gr 3.31 at 25° C); can be with benzol. RETGERS; double nitrate of silver and thallium fuses at 75° C to of 4.5 sp gr, which can be diluted with hot water. BIEDERMAN; mercurous uses at 76° C to a liquid of 5.3 sp gr, which can be diluted with hot water. MUTH- cetylene tetrabromide (sp gr 2.95). BROMOFORM (sp gr 2.82) diluted with benzol .88) is cheap and easily manipulated. Saturated sol of ZINC SULPHATE (sp gr 1.5) t some coals. Solutions containing Ag or Hg must be handled with great care; ntaining I or Br are liable to discoloration on exposure to sunlight; those con- nitrates may attack certain minerals, rendering subsequent analysis misleading.

ion necessary to afford a solution of desired sp gr can be computed from: $v_1 = d) \div (d - d_1)$, where v = vol of heavy sol; D = its sp gr; d = desired sp gr; gr of diluting liquid; v_1 = vol of diluting liquid to be added.

ific gravities of both liquids and minerals are most conveniently determined with c-gravity bottle (a thin glass bulb of 25 to 50 cc capacity, having a close fitting pper pierced by a capillary tube so that bulb will always hold a constant volume d, at a given temp). For LIQUIDS, weigh flask empty, clean, and dry (b); then ith distilled water (w); finally, filled with the liquid (W). Sp gr of latter =) $\div$ ($w - b$). For MINERALS, first weigh flask full of water (B), individual weights and water being immaterial; then add weighed amount (M) of dry mineral in agments, allowing displaced water to overflow; and detaching bubbles by stirring, ssary; finally replace capillary stopper and weigh flask and all its contents (A). mineral = $M \div (B + M - A)$.

les for testing sands and slimes can readily be improvised; for example, section of steel channel, or a shallow trough made of planed board or ith linoleum, and resting on adjustable supports, will closely simulate a (27).

re elaborate form (designed by H S Munroe and made by Eimer & Amend, ig 6) consists of a strong metal frame about 5 ft long in which flat plates of pine,

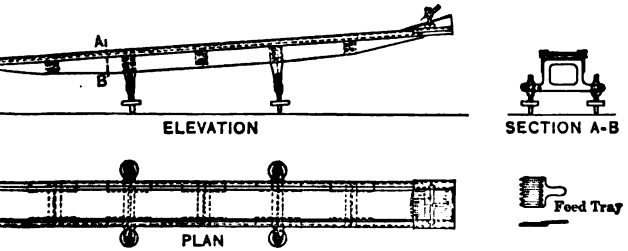

Fig 6. Munroe Laboratory Slime Table

slate, glass or other material may be secured (18). These plates are held by . and the joint between frame and plate is made tight by inserting a strand of wicking. Inclination of table is adjusted by leveling screws, and amount of by dial cock. These adjustments are controlled by testing a small sample of the l to be treated, and giving to the table such inclination and amount of water as fect the desired separation. Data for these adjustments are given in *Trans* E, Vol 27: p 76. It will be found more rapid to adopt somewhat more water and

less inclination of table than recommended in t
washing, reserving the steeper inclinations and re
retreatment of middlings.

A portion of wet ore, preferably classified, is
head of table, care being taken not to feed too fa
rupted and material on the table is washed 2 or 3
off. Tailings pan is then removed, and middli
washed off with a jet of water, into separate pans.
other portion of ore, first replacing tailings pan.
of middlings, clean concentrate and poor tailing
washed by themselves, varying amount of water
lect concentrate, middlings, if any, and tailings; d

Wilfley table on laboratory scale, made by H
facsimile of standard table, including all adjustm
results.

Experimental plant for concentration tests, e
vanners, etc, should preferably be so designed
including tailings, to the same machine in which t
amount of ore and water can be made to operate
time, adjustments being studied and controlled by
and products at regular intervals. This method, a
tion, has these advantages: (*a*) Economy in ore,
information as a car-load. (*b*) Attention can be d
for a sufficiently long period to master details of i
and rate of feed can be maintained uniform for an
completed. This would not be possible if mach
from another machine undergoing a simultaneous t
ore be available, feed conditions are likely to chan
(*d*) Not necessary to clean up and work over, gene
ucts in order to compute results.

Installation of experimental plant involves a
feeder, and an elevator. As installed at Columbi
of any one machine flow to a cone, from which o
consideration) runs to sump from which it can be
ore falls into hopper of feeder, speed of which can
(Centrifugal pump is convenient for this purpose,
charges into feed end of machine, with regulated
used, surplus water required to convey pulp can
cone, at feed end of machine, and proportion of w
tom discharge gate.) Wash water should be und
machine by friction-cone drive, or otherwise.

Computing results of concentration. M
example will be found convenient:

Values in original ore — Au, $30.62; Ag, 8

Product	Weight	Per cent of total weight	Assays		
			Gold, $ per ton	Silver, oz per ton	Copper, %
Concentrate.	71.4	14.28	159.60	27.3	8.45
Tailing.....	284.2	56.84	8.35	4.9	0.72
Slime.....	144.4	28.88	10.68	5.2	1.25
Totals.....	500.0	100.00	....	....	...

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6. Flotation Tests

ability. Thus far, flotation processes have been most successful on ores of zinc and copper; there is no obvious reason, however, why should not be applied to ores containing other sulphides, arsenides, etc, metals (19). Oxide, sulphate, and carbonate minerals have not yet ted successfully, and ores containing over 5% $CaCO_3$ are considered too of acid; in some cases associated sulphides may be floated without acid. on of sulphides from one another by flotation is not yet wholly satisfac- re should be crushed to 60-mesh or finer, but sizing is not necessary; in h processes are found to be improved by presence of slimed sulphides. gests applicability of flotation to finely disseminated ores or tailings. in smelting concentrates in finely divided form is one obstacle to more use of flotation. THOROUGH PRELIMINARY TESTING is particularly advis- re installing flotation process, because the relatively feeble forces of and surface tension, not fully understood, may be adversely influenced unconsidered condition; thus, a porphyry copper ore, suitable for flota- n freshly broken, might prove refractory, due to slight oxidation, after for a time in a stope or on a dump (20). Methods are of 2 general

lm flotation, De Bavay cquisten; froth flotation, Potter-Delprat, Minerals on, Hyde, and Callow.

ethods are now of second- rtance. See Sec 28, Art 28.

methods depend upon adhesion of bubbles on s, sometimes stimulated ute coating of oil (also) on sulphides, and by of acid. Bubbles are d by: (*a*) action of acid onate minerals in ore, or stone added for the pur- no oil being necessary -Delprat); (*b*) liberation lved air, under applica- a vacuum (Elmore); (*c*) ical agitation and aëration (Minerals Sep and Hyde s); (*d*) injection of comp low). To produce mineral th air bubbles alone seems re addition of oil or other agent (21). In "sub- " machine, air enters tically by a pipe to bot- propeller, action of which suction through the pipe.

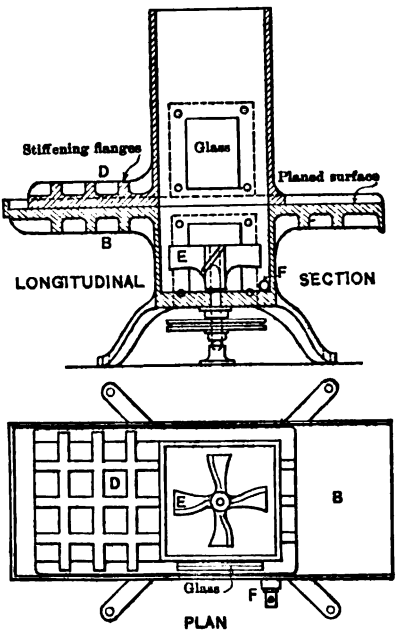

Fig 7. Froth Flotation Testing Apparatus (21)

ing apparatus. Small tests can be made with an ordinary egg beater rovised propeller, operating in a square glass jar, froth being skimmed off spoon. Similar apparatus can be driven by small electric-fan motor, Tif- otor, or water motor; a motor permits more sustained and uniform agita-

tion than hand power. More elaborate dev
is cast in bronze, and sliding surfaces bet
Paddle *E* is driven by belt from motor.
above bottom of upper window; start moto
air through a tube reaching down nearly to
When temp is about 75° C, add desired quar
to give paddle about 1 500 rev per min.
0.5 min for froth to rise, and slide *D* ove
Add water to original depth, repeat agita
Operation may be repeated so long as froth
water and tailings through *F*. Dry, weigh,
If value of ore is known, it is not necessar
Art 5, is considered accurate enough for the
tailings, and compute recovery by difference.
can be disintegrated by boiling in water, or
alcohol.

Variable factors (22) in froth methods
tion of oil to ore; (*c*) proportion of acid
and duration of agitation; (*f*) fineness of o

Nature of oil. Put into a large test-tube 10
0.01 gm of oil to be tested. (To save time in w
should be made as to weight of one drop of each
stand. Repeat with other oils, and compare de
Relative prices of the oils must be considered,
2.5 lb oil per ton of ore. Following oils are av
ANIMAL: oleic acid ("red oil"), whale and fish
numerous sources, blast-furnace oil and tar, cylin
castor (mixed with kerosene), olive-oil residue,
oil, pine-tar oil. Nothing definite can be state
given ore; in general, best results have been at
and pine-tar oils. Selective adhesion is as impor
on some ores, produce an excellent froth, but it n
SOLUBLE FROTHING AGENTS are the following:
essential oils (notably eucalyptus), phenol, cres
tine, pine oil. Advantages of soluble frothing a
sulphides, can be separated directly on tables
agent, whereas the fatty oils must first be bur
does not interfere with subsequent cyaniding of
fatty oils are used.

Proportion of oil. Make series of tests, in n
2 to 5 gm H_2SO_4, at constant speed and temp (7
tion (say 5 min), but vary oil by increments o
weigh, and assay concentrate, and compute per
note at what point weight of concentrate incr
extraction; this condition indicates that further
ful, but would also diminish richness of concentr

Proportion of acid. Make series of tests like
seems best, but varying H_2SO_4 from 0 to 4 gm, in
as above. noting point at which max richness of
centage recovery. Acid is employed: (*a*) to gen
bonates; (*b*) to increase adhesion of oil and bubb
of oil to gangue, and diminish surface tension
gangue minerals to float. It should be understo
its presence, in general, promotes cleanness of c

Temperature. Adopting proportions of oil
series of tests at temp of, say, 20°, 45°, and 90°
viously obtained at 75° C, noting both richness o

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l, higher temp favors both higher extraction and richness of concentrate; it reduce required amounts of acid and oil.

on, speed and duration. Adopting most efficient conditions ascertained as y result of diminished speed of agitation, with a view, if possible, to reducing al energy required in a commercial installation. In general, the more ener- agitation, the more complete is the extraction. In laboratory apparatus having f 4 in diam, a speed of 1 500 rev per min (equivalent to peripheral speed of er min) is recommended. DURATION tests may cover periods of, say, 1 to 8 min, covery and richness of concentrate. In general, longer agitation gives higher but at expense of impure concentrate; shorter agitation requires more oil to sh a desired recovery.

ss of crushing. Make screen analysis of tailings from any preceding test s shown fairly satisfactory recovery, and assay each size. Losses will usually be coarser sizes, and if they are considered excessive, desirability of finer crushing ed. Erroneous results would be given by first sizing the ore and then attempt- at the sizes separately, since commercial methods are always applied to unsized presence of slimed ore seems to be advantageous to froth processes.

7. Amalgamation Tests

amalgamation. A sample of the ore, previously crushed to, say, o-mesh, is ground for a definite length of time in a Wedgwood or an rtar, with addition of mercury and such ingredients as would be used ice; for example, caustic soda, salt, copper sulphate, or sulphuric acid. owing mixture is commonly used: ore, 300 gm; water, 100 cc; caustic gm; mercury, 50 gm. Grind for 1 hr, pan off the mercury, dry and e residue, and calculate extraction.

mall test of this nature, results calculated from DIRECT DETERMINATION of the silver taken up by the mercury are seldom reliable, owing both to the difficulty ing mercury quite free from these metals, and to the likelihood of loss while ff the excess mercury by heat, or dissolving it in acid. If it is desired to make ect determination, either of the following methods is convenient: (*a*) ACID NT. The mercury used in the test (say about 50 gm) is collected in a single by careful panning, and washed free from every particle of adhering matter. heated gently in a casserole or large porcelain crucible with about 20 cc of dilute id (1 vol strong HNO_3 to 2 vol water) so long as reaction continues. From time, the liquid is poured off and replaced by fresh acid, so as to avoid too great tration of dissolved salts, causing crystallization of mercuric nitrate. Finally, ining bead of gold is washed several times with hot water, dried very cautiously, ited and weighed. It is best to wrap it in a small piece of lead foil, and cupel eighing, as it may retain impurities insoluble in nitric acid. (*b*) HEAT TREAT- Put gold-bearing globule of mercury into scorifier. Invert a smaller scorifier, small hole bored through its bottom, over first scorifier, and lute with fire clay. ry carefully in mouth of muffle until no more mercury will condense on cold d over vent in scorifier. Remove top scorifier, add a little test lead and borax orify and cupel lead button. Part and weigh as usual (Sec 30, Art 6–8). Heat has advantage of not requiring such careful coalescing and cleaning of mercury before treatment, since any adhering silica, magnetite, etc, can be slagged off corification.

ening of mercury, through the action of certain minerals or the pres- grease, talc, base metals, etc, in the ore, makes it difficult and tedious to e from the residue, by panning. In such cases the addition of caustic odium amalgam, or sal ammoniac may assist in causing the mercury to n a single globule. SODIUM AMALGAM may be made by adding metallic , in very small pieces, to a quantity of mercury contained in a large . Each fragment of sodium, as it is pressed beneath the surface of the y, produces a slight explosion as it amalgamates. About 2% of sodium e conveniently added in this way.

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Plate amalgamation. A method which f
roll a weighed sample backwards and forw
mercury and dilute caustic soda solution. T
used: ore, 200 gm; solution, 100 cc, contain
A convenient mechanism for this test consists
one revolving loosely and the other driven b
laid between them, and may be kept revolvi
contents are panned to separate the mercu
difference between original and final assays.

Grading analysis may be made on tailings from
finer crushing would be likely to improve recove
purpose, however, it would be best to run a parallel
same proportions, and separate the final residue i
successively through 60 and 100-mesh; each produ
other portion of residue may be concentrated by l
tion, following amalgamation, would be advisable.

Tests on plates, made by passing small quanti
or pan, or by agitating with strips of amalgamate
results, perhaps because the physical condition of
ore differs from that in ore crushed by a stamp ba
be confirmed by TESTS ON A LARGER SCALE, with a
a ton or more of ore in a day's run. In this case,
mounted on an inclined table with raised edges,
imitated as closely as possible. To obtain good
be dressed, previous to test, by rubbing with a su
gam, as a copper plate coated with mercury alone
amalgamation. Samples of tailing should be scr
above, and 3 sizes separately assayed to locate so
finer crushing is suggested; if in the finer, the ba
sliming (Sec 28, Art 7). Microscopic examination
case of "rusty gold" is due only to incomplete cru
ules of amalgam indicates improper amalgamating
water, hard amalgam, or insufficient plate area clo
ish stains on plates indicates only poor amalgamati
possible losses of slimed tellurides and selenides (2

8. Cyaniding Tests (see S

Preliminary tests are often desirable to
at all suitable for cyanide treatment. The
limits to which such treatment could be carr
will cover all possible cases, but the follow
Take 100 gm of ore ground to 200-mesh. A
tinct alkalinity remains permanent on agi
parts of liquid to 1 of ore; add cyanide to g
tate continuously for 72 hr; wash out on
cyanide. Wash residue on the filter thorou
With certain classes of ore it may be necess
by injection of compressed air before or dur
final residue repeatedly by decantation, stirr
decanting, with or without vacuum filtration

Variable conditions in cyanide treatm
suitable for cyanide treatment, the next p
(*a*) fineness to which ore must be crushed;
tion; (*c*) most suitable dilution of pulp;
(*e*) time of treatment necessary; (*f*) whethe

e a portion for percolation treatment; (*g*) whether oxidizers or other
igents are necessary.

agitation. Most of the above points can be ascertained by simple
ests in bottles along the lines of the standard test given above. These
iiently performed by attaching bottles (glass preserve jars are suit-
ıy revolving mechanism, placing the stoppers towards the center of
Bottles may be protected by wrapping in cloth, or may be secured
cases affixed to the wheel, and firmly packed to prevent slipping.
ıld be sufficient to give efficient agitation, but not so rapid that the
ld against the bottom of the bottle by centrifugal force; about 20
n is generally suitable. The following scheme may serve as a model,
d in detail according to circumstances.

s of crushing. Make 6 bottle tests, using in each case: ore, 100 gm;
oo cc; lime, 1 gm; cyanide, 0.5% KCN. Crush an average sample
to 20-mesh and weigh out one charge of 100 gm for test; then crush
to 40-mesh, take 100 gm for test; and so on successively to 60, 80,
200-mesh. Agitate in bottles as above for 24 hr, test strength of
ıd add enough cyanide to restore solution to its original strength of
ntinue agitation another 24 hr, make up strength as before, and
gitation for a total period of 72 hr. Filter, wash, and assay residue.

trength. On indications of previous series, determine fineness to
ore should be crushed. Supposing that nearly identical results are
120 and 200-mesh ore, crush a sufficient quantity to 120-mesh, and
dentical tests, using: ore, 100 gm; solution, 300 cc; cyanide, 0.5%
ld respectively the following weights of lime: 0.02 gm, 0.05 gm, 1 gm,
gm, 3 gm. Agitate simultaneously for 72 hr, and test residue as be-
determine alkalinity and cyanide strength of final solution (see be-

lity of an ore is determined by agitating with an excess of alkali solution of
rth. For example, agitate 100 gm of ore, crushed, say, to 120-mesh, with
% NaOH solution, without cyanide. After 1 hr, titrate solution (without
h standard acid, and calculate consumption of alkali. Sometimes compara-
e desirable to determine whether lime or caustic soda is preferable.

of pulp. Crush a sample of ore to fineness indicated by preceding
l lime or other alkali, also as indicated. Make 6 tests, each on
, with the following volumes of 0.5% KCN solution: 100, 150, 200,
oo cc. Agitate 72 hr, maintaining strength at 0.5%; test final solu-
yanide and alkali, and assay residues after filtering and washing.
ll indicate the most desirable dilution of pulp, assuming that the
treated by agitation, on the all-sliming principle (Sec 33, Art 11).

of solution. Supposing that best results are obtained with a
2 : 1, run a series of 6 identical tests, with 100 gm of ore, 200 cc solu-
n lime (or other amount as previously indicated). Agitate continu-
hr, using solutions respectively of 0.05, 0.10, 0.25, 0.5, 0.75 and 1%
gth. Test final solutions, calculate cyanide consumption, and assay
er washing and drying. In case one or more of these tests show all
ll of the cyanide to have been consumed, it is advisable to repeat
determining the cyanide strength at intervals of a few hours, and
cient solid cyanide, or strong solution, to bring the strength up to the
andard. This is repeated until successive tests show practically
consumption of cyanide. Any solution drawn off for such tests
eplaced by an equal volume of water, so as to maintain a constant
ution to ore. Final extraction, when the solution has been main-

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tained at given strength, will often be muc
test in which most of the cyanide was destro

Example showing method of maintaining KCN solution originally taken; total cyanide agitation with ore, 50 cc taken out, after standi 0.19% KCN; remaining solution, 150 cc at 0.19 added to make up strength = 0.50 − 0.285 = 0 solution containing 1.05% KCN; then 20.5 cc Measure out 20.5 cc of the strong solution, dilute the test. This gives us 200 cc, containing altog 0.25%.

Time of treatment. Assuming previou 0.25% KCN, run tests with (say) 100 gm o tion, 1.5 gm lime, stopping treatment succe 96 hr. Test solutions and assay residues.

It may sometimes be desirable to vary two or m if a strength of 0.25% KCN gives the best result sarily follow that this strength of solution will be agitation. Where facilities permit, a large numb may be made simultaneously, thus shortening tim

Percolation tests. Whenever the results without extremely fine grinding, it become would not be advantageous for at least a po test may be made.

Crush (say) 20 to 100 lb of ore to pass 40-mesh classification, or by repeated stirring with wate sandy portion as dry as possible, mix thoroughly and on an average sample determine moisture an in proportion determined by previous acidity tes tank. This may be an inverted glass bottle with b of a small cask. Filter-bed is required, which m framework of wooden slats, covered with thin can canvas cover. The edge of the filter-frame must or similar material. An outlet for the leaching frame, the flow of liquid being regulated by rubb Only a little space must be left between filter-fra tions are added at intervals, say once or twice a da of ore each time. A good plan is to begin with a ore; then follow with strong solution, draining off a is saturated with it; then add enough strong solut stand, with outlet closed, for at least 24 hr. Afte sively weaker solution may be given, ending with the original moisture of the sand treated. Each s pletely as possible before applying the next, to se mum of liquid. The charge may be sampled at i care to get a correct average from top to bottom, an of the charge. If all the solutions drawn off are alkali, a very fair estimate may be made of the pr working scale. Finally, the entire charge should be and assayed. A portion may also be tested afte filter.

Aeration tests are generally made in sma struction of the Pachuca or other form of a air supply requires careful attention; if ins if too abundant, consumption of chemicals m to compare results from aeration tests with th aeration, but under otherwise similar conditio

ests. A great variety of conditions may occur in different ores, which may an elaborate research into such questions as the necessity or disadvantage ary water-wash, acid-wash, or alkali-wash, oxidizing agents, roasting, concen- algamation, or other process, and the use of additional reagents during cya- ent, such as bromocyanide, lead salts, etc. These tests cannot be detailed he same principles apply as in those above described. Results should be con- larger-scale tests. In general, WATER-WASHING is advisable for partially oxi- containing iron or copper sulphides, and ACID WASHING is also frequently bene- r the same conditions, particularly where basic sulphates insoluble in water . ALKALI WASHES may be advisable with arsenical and antimonial ores; for he preliminary desulphurizing process with caustic alkali and aluminum may useful (28). OXIDIZING AGENTS improve extraction from certain ores contain- idizable sulphides and certain kinds of organic matter; preliminary treatment anganate, with or without dilute sulphuric acid, is sometimes advantageous conditions are present. ROASTING is beneficial and sometimes essential with ning heavy sulphides, arsenides, sulpharsecides, tellurides, etc, but the cost ation, in comparison with the possible increased saving of valuable metals, ays be considered. CONCENTRATION is often advantageous when values are with sulphides or other heavy minerals, provided that the latter do not con- large a percentage of the ore. AMALGAMATION should always be tried in coarse gold and perhaps, also, of native silver. BROMOCYANIDE has been ly successful only with telluride and with mispickel ores.

tation tests may be made by causing a sufficient volume of solution hrough a model zinc-box or, less satisfactorily, through a vertical osely packed with zinc shavings, at a definite rate for a measured f time, varying these conditions in successive tests. Samples of the ntering and leaving the box are analyzed for precious metals, cya- alkali. Tests on ZINC or ALUMINUM DUST precipitation may be made ng (say) 0.5 gm of the former or 0.1 gm of the latter with 100 cc of or silver-bearing solution in a stoppered bottle for 5 min, filtering a vacuum filter, and assaying filtrate. Recovery of silver may be sted by dissolving the precipitate collected on the filter in dilute , and titrating the silver by Volhard's method. Proportions given suitable in many cases, but should be varied to find the most effec- ions.

n tests. At the end of every test, and sometimes at other stages of solutions are tested for free cyanide, alkalinity or acidity, and occa- determination of gold or silver contents is useful as a check on com- f extraction based on original and final assays.

nide. Titrate with standard silver nitrate until a cloudiness or opalescence manent. Many operators add a few drops of strong potassium iodide as an this titration. Standard silver solution commonly used contains 13.04 gm litre, so that 1 cc = 0.01 gm KCN. Some practice is required for determin- finishing point in impure solutions.

y or acidity is determined in the same or another portion of solution, after $AgNO_3$ until distinctly turbid. Add phenolphthalein and titrate with stand- alkali, as the case may be, until the pink color disappears or appears. Alka- culated in terms of NaOH, KOH or CaO; acidity is usually calculated as

y be rapidly estimated by J. Moir's colorimetric method (24). Cyanide is y boiling with sodium peroxide; lead acetate and aluminum dust are then ssively, the precipitate is filtered off, redissolved in aqua regia, and strong loride is added drop by drop until the yellow color disappears. On standing urple color develops and reaches its maximum. The tint is then compared btained from known amounts of gold, in a standard gold chloride solution, the same amount of stannous chloride.

etails of this and other solution tests, see Bibliog (25).

9. Other Wet Extra

Chlorination process is usually applied in which gold is the valuable constituent crushed, is roasted at a moderate temp 100 gm, this may be done in ordinary clay Larger quantities may be spread on floor tests, a small reverberatory furnace lined w

Roasting. Charge is turned over from time t by a piece of stout wire, having one end flatt finish of the roast, when most of the sulphur, ar been expelled, the temp is raised to a moderate withdrawn and allowed to cool. Roasting may not be hurried.

Chlorinating. Roasted charge is now slig over a specially prepared filter bed. For small disk horizontally inside a large funnel, and co glass beads, or similar material on which chlori with flat perforated diaphragm, used for vacuu purpose. A larger test should be made in a s layers of pebbles decreasing in size from below u should be provided. Chlorine gas is then intro diffuse upwards until it begins to issue from ben over the top. The chlorine may be generated parts sulphuric acid and 9 parts manganese di and potassium permanganate; (*c*) chloride of charge is indicated by smell of the issuing gas, o of an ammonia bottle is held near edge of cove from chlorine generator, and cover luted on wit inlet also being closed. Action of the enclose turbed for several hours, in some cases overnigh

Leaching. Charge is then leached with wat through the porous bed, are collected. Washin assayed for gold, or entire gold contents may hydrogen sulphide, collected and assayed. In l for settlement of precipitate (say 48 hr). Wa assayed, when extraction may be checked by di

Acid treatment. Important developm of copper render it necessary to have some to acid treatment. As in cyanide treatm in bottles, varying such conditions as finen tion of pulp, and percentage of sulphuric a be assayed not only for copper but for oth are likely to be extracted; the amount o determined. Nature and amount of the di the causes of acid consumption.

Acid-resisting material becomes importan should be painted internally with some acid-res be avoided so far as possible where exposed to a erally suitable for this purpose; it appears to be hydrocarbon in carbon bisulphide. Various for been introduced as filtering media, and may adv filter for percolation tests. Certain ACID-RES and others, may be useful for pipe-fittings, pu use of metal is unavoidable.

Precipitation. Two methods have been us electrolysis with copper cathodes. Either pro scale tests. To test PRECIPITATION ON IRON, a

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t quantity of iron filings, using various proportions until best conditions are d. For larger quantities, solution may be passed through a launder contain- fragments of hoop iron, etc. ELECTROLYTIC tests require special apparatus, uments for measuring quantity and potential of current. If strong currents ary, the designing of cells composed of, or lined with, some insulating material mportant. Tests on composition of electrolyte, and of products deposited at e and cathode must be made at intervals. Electrolytic processes suggested ent of copper ores generally require an iron anode; ferric sulphate is produced ctrolysis, and its solution is used as solvent for further quantities of copper.

treatment is used chiefly for extraction of antimony and arsenic, lly as a preliminary step, before applying some other solvent for re- precious metals. Methods of investigation are sufficiently indicated sts previously described. Sulphides and polysulphides of sodium or are occasionally used as solvents for ores of this class. In designing s for large-scale tests, the action of caustic alkalies on the material construction must be considered. Silica ware suitable for acid treat- y be found unsuitable in this connection. Iron tanks and fittings rally be used with alkali solutions of ordinary strength.

ulphite treatment is used for ores in which the value is mainly uch ores are generally given a chloridizing roast, and this operation refore be imitated when testing the suitability of an ore for this treat- ometimes it is desirable to give a preliminary oxidizing roast, as de- nder chlorination; in other cases, the required amount of salt may be th the raw ore.

izing roast. Tests may be made in roasting dishes in a muffle, but the kl be carefully regulated and not allowed to rise above dull redness. Since this process depends on a reaction between chlorine, generated from the salt, minerals in the ore, excessive stirring after addition of salt should be avoided; st results are obtained by covering the charge at a certain stage, and allowing o proceed undisturbed. A partial oxidizing roast before adding salt has the nverting iron sulphides into sulphates, which latter react with the salt, forming lphate and liberating chlorine. In some cases it may be advantageous to add lphate, sulphur, or other chemicals, along with the salt. Many experiments ring conditions may sometimes be necessary before best method of roasting is d; thus, proportion of salt, length of time in furnace, stage at which salt is n of final roast, and advisability of preliminary oxidizing, are among the bles. Losses of gold and silver in chloridizing roast are increased by presence elements, such as arsenic, antimony, tellurium, or selenium. When these are sses may be reduced by eliminating such volatile substances as far as possible inary oxidizing roast, before adding salt. Having obtained a satisfactory necessary to determine what loss of weight and of precious metals has occurred, excessive, such losses may counterbalance the advantage gained by a high in subsequent leaching. All charges should, therefore, be carefully weighed, nd assayed, before and after roasting.

lphite leaching is done in much the same manner as cyanide treatment. ases, hot solutions may be advantageous. Strength may be varied from (say) % or more. Sometimes a preliminary water-wash may be given, and occa- dium carbonate is added to precipitate lead in the extract. After the ordinary ite treatment, a wash is sometimes given with an "extra" solution, containing hyposulphite of copper and sodium, which dissolves an additional amount of ompletion of extraction in leaching tests is shown by ceasing of the effluent o react with sodium sulphide, though a reaction with sodium sulphide may in- d, copper, or other metal besides silver. Both solutions and residues should l for silver.

10. Equipment for General Ore-Testing Plant

ing appliances. Pulp sampler (Vezin, Brunton, etc); large and sma[11] mpling floor; mixing cloths; buckboard.

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Crushing machinery. Rock breakers; tube-mills; grinding pans; disk pulverizer

Screens. Standard testing screens; m

Classifiers. Spitzkasten; spitzlutten; low cones; elutriating cones; tube classifie

Concentrators. Frue vanners; Wilfle bateas; magnets.

Amalgamators. 5-stamp battery, wi pan; Berdan pan or Huntington mill.

Cyaniding apparatus. Wheel and rol mechanical agitators; Pachuca tanks; filter; centrifugal pumps; vacuum pumps sories; zinc precipitation box; zinc-dust precipitate press.

Furnaces. Small reverberatory roasti for drying samples; fusion and muffle assa

Assay and analytical apparatus, incl ments, material and reagents.

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SECTION 32

ES ON SELLING, PURCHASING, ND TREATMENT OF ORES

BY

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ction. This section deals briefly with some of the metallurgical problems relat- disposal of ores after they have been mined, and which are of interest to mining To cover the subject thoroughly would require citation and discussion of schedules, which would be out of place here.

n ore is of constant character and value, as is common, for example, in case of inc, fixed and definite terms can be formulated for sale or purchase. But, for complex ores, special terms and quotations are necessary, which are sometimes comprehend. Most non-ferrous reduction works receive heterogeneous ores, more or less complicated schedules and terms of settlement. Moreover, as local in different ore-producing centers often vary widely, schedules applicable to t may require radical modification in another. Rates are seldom intentionally plex, but in certain cases it must be admitted that they could be simplified, for mprehension by small shippers. NOTE. — The prices and rates given below he period antedating the Great War; the present conditions are abnormal.

1. Smelting Ores; Fluxes

pal questions relating to sale and purchase are: (*a*) freight rates; od of sampling; (*c*) composition of the ore; (*d*) fluxing requirements e a satisfactory slag, and available fluxes at the smelter, either in shape material or other classes of ore; (*e*) value of recoverable metals; (*f*) of impurities contained in the ore; (*g*) cost of treatment and refining; l losses occurring in treatment; (*i*) interest on value of metals tied up reatment and commission for selling metals; (*j*) terms of payment; ment charge; (*l*) interest and amortization on plant.

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Freight charges from mine to reductio
(*b*) value of ore; (*c*) tonnage handled.

Small shippers can not always secure as satisfa
custom plants handling large tonnages usually
works, the amount being included under FREIGHT
grade ore always exceeds that for low-grade, beca
in transporting more valuable material, and to
thereby increasing the railroad's business. Ship
smelting returns are on dry ore, and shipper pays

Sampling. For methods, see Sec 25, 28 an
carefully taken, especially in wet or snowy we
time of weighing the ore. It should be dried a
crystallization may be driven off, which is n
present definition and the shipper thereby lo

Final fine-pulp sample is usually divided into
and marked for identification. One sample is us
determine the value of the ore. If these determ
LIMITS, provided by agreement, the average is take
sample is sent to an umpire, the fourth being held
or umpire. The seller or shipper should always
to watch the sampling, and receive duplicate sam

Chemical composition. Determinations
Pb, Zn, SiO_2, Fe, Al_2O_3, CaO, S; sometimes
stances. The analysis reveals the gangue
the ore is self-fluxing, silicious, or basic.

Few ores are self-fluxing; they are usually eith
material is added to the furnace charge; if basic,
the ore mined in many regions is silicious, basic
ore is penalized. The aim is to make a self-fluxing
and the penalty for SiO_2, are such that they offset
culated to produce proper slag. The smelter is
this system of premiums and penalties, though in
his favor.

Fluxes. For proper fluidity of slag, cer
Fe and CaO must be present. In copper sme
proportion to bases than in lead smelting, t
less than formerly. CaO usually costs 6–8
must be accurately known for any given
is sufficient to offset payments for Fe and
shipper of silicious ore may be returned as b

Examples. If limestone containing 52% CaO a
at smelter, and the cost of smelting a prepared or
direct smelting cost will be $3.36, which, divided
(52 units, less 4 units of CaO required to flux the
larly, if an ore containing 59% Fe and 5% SiO_2, v
delivered, $2.50 per ton more than the value of th
cost is $2.50, the total cost of $5, divided by 50
flux SiO_2) gives a cost of 10¢ per unit.

If it be desired to make a slag containing 36%
CaO, the cost for SiO_2 is determined thus: 29 un
7¢ = $1.33, total $4.23, which covers cost of fluxi
per unit of SiO_2 equals 11.75¢, say 12¢. Under
7¢ a unit, Fe at 10¢, and SiO_2 charged for at 12¢

Fe is usually obtained from special ores contain
to such a limited extent in most ores that it must b
quarried as near the smelter as possible.

2. Value of Products; Influence of Impurities

e of recoverable metals is determined by analysis and assay of the (Art 1). The principal metals in smelting ores are Pb, Cu, Au and Ag.

d Cu are reported in percentages. The value of Cu is determined by the standard n per lb; of Pb by the quotation per lb or unit. A UNIT is 1% of a ton, or 20 lb. Ag are reported in Troy oz per ton of 2 000 lb; value of Au is fixed at $20.67 per z. by quotation. In determining the value of Cu, Pb and Ag, it is specified that ation used shall be that of a certain date. The buyer usually pays for the ore making allowance for the time he has to carry the values in the ore. Sometimes ified that the quotation used shall be that of a certain number of days after settle- assay (30, 60, or even 90 days); during which period the metals may be recovered melter and sold at approximately the price that is paid for them.

ence of impurities has an important bearing on the value of an ore on the price paid for it. If the impurities are small in amount, they re- attention. But, if there is possibility that the ore contains impurities tent that will interfere with reduction and recovery of metals, penalties osed.

pal Impurities. SULPHUR in copper ores is not detrimental and is seldom pe- But sulphur in lead ores, if above a certain limit, makes roasting necessary. As eases treatment cost, an ore containing more than say 4% S is penalized on the 15 to 25¢ per unit in excess of that amount, up to a maximum of $2 per ton. pecially as blende, creates irregularities in smelting; part goes into the matte part into the slag, making it pasty, and part is volatilized, causing trouble in the and increasing silver losses by volatilization. The usual limit is 10% Zn; at elters, all in excess of 5% is penalized. The charge for excess Zn is 25 to 50¢ per ARSENIC is usually penalized in lead, and frequently in copper ore. With lead, it EISS (Art 3), a troublesome by-product difficult to retreat; with copper, some of reduced in the bullion, increasing refining cost. The penalty varies greatly, and figures can be given; it is sometimes 20 to 25¢ per unit in excess of 3 to 5%. RY is also troublesome, but is seldom penalized, as it is more easily treated than refining a lead bullion from a Pb-Ag ore, part of the Sb is recovered as antimonial

3. Treatment: Methods and Costs

lting operations comprise: (*a*) roasting the ore; (*b*) furnace treatment; ning the metal or metallic compound from the furnace, and treatment of ucts. The principal by-products are MATTE, an artificial sulphide of r metals, and SPEISS, an artificial arsenide of Fe, or antimonide of Cu. re retreated to recover metals. Under (*c*) are included: refining lead converting copper matte, and refining copper bullion. Fig 1, 2 illustrate plicated processes for producing Pb, Cu, Ag and Au from their ores.

of each treatment step must be considered in fixing the terms for purchasing an these costs vary with local conditions at the smelting plants.

eral basic costs for large plants are: for lead smelting operations, cost of charge, $2.50 to $3.50; for copper smelting, cost per ton charge, $1.50 o; for refining lead, $4 to $6 per ton of bullion; for converting copper about $8 per ton blister copper produced; for refining copper, $11 to $13 bullion. Cost for smelting lead- or copper-bearing material may exceed smelting per ton of charge. Costs at comparatively small plants are igher than for large. Interest and amortization should always be con- in costs.

l losses in smelting and refining. In LEAD ORE, loss of Pb varies from %, depending upon percentage of Pb, and upon whether refractory or The lower limit is for a charge carrying about 40% Pb; the higher, for a carrying about 10–12% Pb. Copper loss in lead smelting sometim[redacted]

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reaches 30%, as the amount of Cu permis
limited. In smelting COPPER ORE, the loss
but the recovery is always much higher t
small, from 2 to 5%, unless conditions ar
small, and is not usually considered.

Interest and commission are relatively small
settlement of assay, interest is allowed on the valu
pending upon how soon they can be recovered. But, if the metals are not paid for until sufficient time is allowed for their recovery, no interest charge should be made. Commission for selling is usually allowed for in the net price received for refined metals.

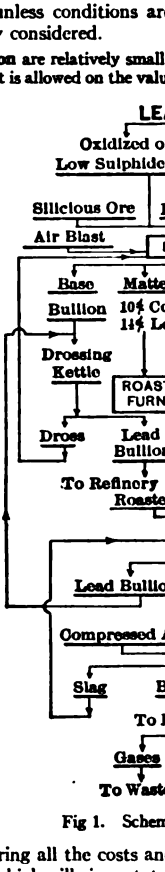

Fig 1. Schem

4. Terms of Payment for Metals

Factors. In purchasing any class of ore all the points stated in the beginning of Art 1 must be considered. From (*e*) is found the value of recoverable metals, and from (*j*) the amount paid for them, the difference being the MARGIN ON METALS. From (*c*) and (*d*) result the premium or penalty for basic or silicious ores, and from (*f*) the penalty for impurities. Cost of treatment (*g*) and losses of metals (*h*) are determined from records and experience in the smelting plant. Treatment charge (*k*) is then fixed by considering all the costs an to this margin a charge which will give a tota desired. Thus the greater the margin on need be, to secure a given profit. (The tre trary figure, acting as a balance wheel.)

The gross value of an ore to the smelter is th constituent arrived at by taking market quotatio freight and refining. The term GROSS MARGIN value of metals present and the amount paid fo losses of metals, and profit (*i*).

Gold is usually paid for on basis of 100% of th but no payment is made when the ore contains

or rich ore; the lower, for poor. As there is little or no loss of Au in ordinary operations, the difference between the value of the Au and the amount paid for an covers the carrying interest and losses, and therefore adds to the margin on the ore.

is usually paid for on basis of 95% of the ore content, at N Y quotation per Troy silver at the time agreed upon. No payment is made if the ore contains less o 1 oz Ag. More Ag is recovered in smelting than is usually paid for, the differng to the margin on metals, as in case of gold (11).

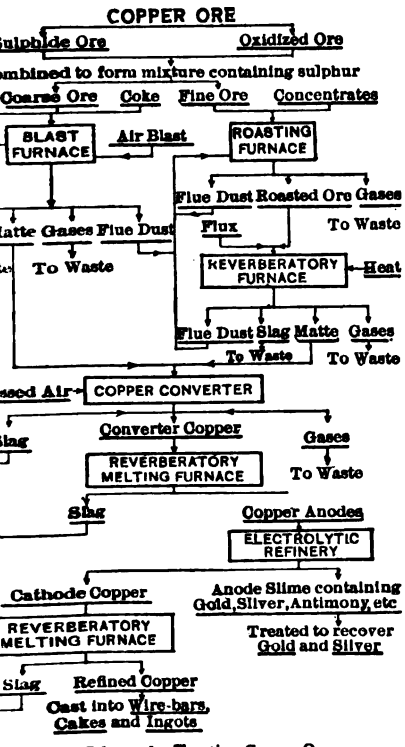

Fig 2. Scheme for Treating Copper Ores

Lead. Many different terms are used in making quotations. The most common is to pay for 90% of the Pb content of the ore, at a certain price per unit. The 10% allowance is supposed to cover smelting loss, and in making it the variation in Pb content does not affect subsequent terms or conditions. The payment is on basis of the quotations, less the cost for freight to refinery, refining lead bullion, and the selling charges. If the Pb content is less than about 5%, it is not paid for. Freight and refining usually cost about $20 per ton, depending upon conditions; hence, if price of Pb is 4¢ in N Y, the price paid in Colo (to allow for freight and refining) would be 4¢ less 1¢, or 3¢ per lb (11). If an ore contains 20% lead, or 400 lb per ton, 90% of this (360 lb) would be paid for on the above basis, at 3¢ per lb (= 60¢ per unit), or a total payment of 18 units × 60¢ = $10.80 per ton of ore. (For local schedules, see Bib 1, 7, 11, 14, 15.) Mode of assay (Sec 30). Wet assay is exact and is preferable for the seller. Dry assay (by fire) is usually lower (1% to 2% lower) assay, and, if it be used, the purchaser pays correspondingly less for the metals. y has more effect on the returns from a low-grade than a high-grade ore. Thus, ontaining 40% Pb might show by dry assay only 38% (difference, 5%); or a might show only 18% (difference, 10%). In the latter case, 90% of 18% Pb paid for (324 lb instead of 360), which, at 60¢ per unit, gives $9.72 instead of Payment is made on basis of 90% of the corrected assay, which equals the wet is a specified allowance of usually 1.5 units.

r was formerly paid for on basis of the dry or Cornish assay, estimated at about less than the amount of Cu in the ore. This difference was supposed to cover u in old methods of treatment. This assay is now seldom used, Cu being paid asis of the wet (electrolytic) assay, less a certain percentage (usually 0.5 to 1

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unit) to cover assumed treatment loss. With pre
1.3 units. After making deduction, remainder of t
tion (¢ per lb), less amount supposed to cover cost
bringing the Cu to merchantable shapes. (For av

These terms seem complicated and the que
all ore be purchased with an outright specifie
that, if the terms of payment for each metal
variations in grade of ore will materially affe
if no allowance were made for Pb loss in qu
would have to be much higher for a rich than
cult to frame a sliding scale to fit all conditio
made for losses, but if proper terms are used th
smelter can name a lower charge than would ot

Treatment charge. After arranging tern
or penalties for character of ore, and penalties
charge is made (Art 3). This charge, with othe
(and roasting, if necessary), treatment of by-
ucts, refining of metals and profits. Treatment
ing cost, in which case margin on metals cover

Problem in ore-purchasing. Assume an ore co
Cu, 2.3%; Fe, 20%; CaO, 3%, S, 16%; SiO_2, 20%
for such an ore are: Au, pay for 100% @ $19 per Tr
say 55¢ per oz; Pb, pay for 90% of wet assay less
4¢ less 1¢ = 3¢ per lb) or 60¢ per unit; Cu, pay
(say 14¢ less 6¢ = 8¢ per lb); Fe, pay for at 10¢
SiO_2 is penalized at 14¢ per unit; Zn, if over 10%
10%. Treatment charge, $2 per ton. Using these

Analysis		Net amount paid
Gold	1 oz.....	100% @ $19.00
Silver	20 oz.....	95% @ 50¢ per oz
Lead	15% wet assay	90% of assay, less 1.
		@ 60¢ pe
Copper	2.3% wet ass'y	less 1 unit @ 8¢ per
Iron	20%.....	@ 10¢ pe
Lime	3%.....	@ 8¢ pe
Sulphur	16%.....	
Less: Silica 20% @ 14¢ per unit.....		
Zinc 12% @ 50¢, excess over 10%.....		
Treatment charge.....		
Gross margin.....		

Outcome: Gross margin.....	
Iron 20% @ 10¢.....	
Lime 3% @ 8¢.....	
Silica 20% @ 14¢.....	
Cost of roasting.....	
Cost of smelting.....	
Lead loss, 10%.....	
Silver loss, 2%.....	
Copper loss, 20%.....	
* Profit to smelter.....	

* In this statement, interest, amortization on p
included, and may considerably affect the appare

tatement shows that the value of the metals at the smeltery is $43.35, while the paid for them is $37.87, a difference of $5.48; which, deducting the Pb, Ag and of $1.84, leaves $3.64 in favor of the purchaser. Though this is not an actual metals, it enables the purchaser to fix a lower treatment charge; in this case, $2 hich is much less than the cost of roasting and smelting. If the margin on metals uced, the treatment charge would have to be correspondingly increased, to ob- profit shown.

asons already given, it is advantageous to allow for losses in quoting on metals, o doing the seller suffers no hardship. The premium paid for Fe and CaO, and levied for SiO_2, are considered in the outcome. The ore, because of its Fe and tent, has an added value equal to the premium paid, and the excess cost, due to which must be fluxed, is considered in treating, this excess being the amount of lty for SiO_2 in the purchase statement. Since the deduction for Zn is considered oss margin, it might seem that it should also appear in the outcome as a cost. is difficult to arrange. Excess Zn increases losses and treatment cost, but the of increase for different classes of ore is not readily determined. For average ore, s and costs are ascertained from the metallurgical results, which furnish the only a (6, 8, 9).

tain localities, as in parts of Colo, ores of uniform character, having their chief Ag and Au, are sold on a FLAT SCHEDULE, no attention being given to ore composi- his schedule is based on a sliding scale, depending on value of ore (1). Treat- arge on low-grade ore is such as to stimulate its production; higher-grade ore d a higher charge. The charge for a Pb or Cu ore frequently decreases as the creases, because: (*a*) an ore of high metal content makes less slag; (*b*) the Pb, a valuable collector for Au and Ag.

ent forms are used for making ore purchase calculations. The table on following ranged by C. S. Witherell) is comprehensive, and includes all the steps required ning ore or bullion value, margin on metals, premiums and penalties, detailed for nt costs and profit.

ores. The purchasing terms are entirely different from those for Ag . Zn ore is mostly sold in open market at a base rate per ton of sulphide ncentrate) containing 60% Zn, a premium of about $1 being paid per unit , and $1 penalty charged per unit under 60. A charge may also be made and Fe. The flat price per ton varies with conditions.

rice usually obtainable for Zn ore or concentrate delivered at smeltery is shown by ula: $(T - 8) \times 20 \times$ price of Zn per lb $- \$16 \pm$ = value at the smelter; in = units of Zn and $16 the treatment charge (W. R. Ingalls). Thus, a concen- taining 59% Zn would be worth, at 5¢ per lb at St Louis, $(59 - 8) \times 20 \times 5¢ -$ 55. In Europe, smelteries generally purchase Zn by the formula,

$$V = 0.95\, P \times \frac{T - 8}{100} - R,$$

V = value, £ per long ton; P = price of ordinary brands of Zn in London (per); T = units of Zn in the ore; and R = treatment charge per long ton (2). The n of 8 is to cover treatment losses, which are quite constant, regardless of the Zn (16).

ore is purchased at flat rate, depending upon the percentage of Fe, with penalty below a certain grade, and a premium if above. For Lake Superior ore the basis 55% Fe. Bessemer ore must contain less than 0.45% P, and commands a higher an non-bessemer, the difference reaching 80¢ per ton. For Lake ore, 1916 prices t 70¢ per ton higher than 1915 prices. Eastern ore is often sold on basis of units Prices in 1915 were 6.5 to 7.75¢ per unit for high-grade ore, delivered in eastern ntracts for 1916 are at 8.25 to 9¢ per unit.

ganese ore. A sliding scale is used (5). The higher the Mn content, the higher e per unit. Thus, if an ore contains 40–43% Mn, payment is made on basis of unit; if from 44–46%, 24¢; 46–49%, 25¢; above 49%, 26¢ per unit. The ore not contain more than a certain percentage of P or SiO_2 (say not over 0.2% P, r 8% SiO_2). Above these limits, penalties are levied (2).

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TERMS OF METALLURGIC

ON QU
SHIPPER
ADDRESS
DESCRIPTION OF MATERIAL
(8 lines)

PROBABLE QUANTITY IN LOTS OF

Assay Quotation Market Value		AU		AG		PB		
Anal	H_2O	INSOL	SiO_2	FE	CaO	ZN	S	
Metal Deductions	METAL	Assay deduction		Pay for %		Quotation Less		
	(5 lines)							

TOTAL PAY FOR METALS
MARGIN ON METALS
TREATMENT

Premiums	ELEMENT	ABOVE	PER UNIT	REM
	(3 lines)			
Penalties	(5 lines)			

TOTAL MARGIN
SHIPPER RECEIVES

Treatment Costs	STAGE OF PROCESS	OPERATING COST			METALLU		
		BULK	RATE	AMOUNT	%AU	%	
	(8 lines)						
	TOTALS						

CARRY	METAL	Value Extracted		Days tied up	Rate
	(5 lines)				
	Total Carry				

TOTAL COST
PROFIT % OF MARKET VALUE % OF T

5. Milling Or

Terms used in purchasing ore to be treate
(Sec 33) are comparatively simple, the operat
in smelting. The gangue composition is far l
tion of making a proper slag, and treatment

ge of recovery of Au and Ag is less than in smelting, but the treatment so much lower (especially in cyanidation) that many ores can be more orily milled than smelted.

Au is paid for at $20 per oz, and Ag at N Y quotation on day of settlement. amount of Au is sometimes paid for, the loss being taken into consideration in ng charge. Recovery of Ag is n that of Au, and an allowance or this in the percentage paid for. certain percentage of Au and Ag r, the charge is lower. The mill- e usually varies with the ore value, by the accompanying table.

Milling Rates on Cripple Creek Gold Ores, Including Freight (1)

(Full assay value in gold paid for at $20 per oz)

Value per ton	Charge per ton	Value per ton	Charge per ton
Up to $10	$4.00	$25 to $30	$6.50
$10 to $15	5.00	$30 to $40	7.00
$15 to $20	6.00	Over $40	8.00
$20 to $25	6.25	.....	.....

silver ore, as that from Cobalt anada, which has a large amount ties, is purchased on special terms us, for 1 000 to 2 000-oz ore, 93 f the Ag is paid for, the milling eing $8 to $10 per ton. The n Ag which, for this high-grade ore, would be very large, is made necessary by sive milling loss. Most Cobalt ores are now (1915) treated in local mills (2).

6. Ore Contracts

acts for selling and purchasing should be in legal form, and supervised eers or metallurgists having full knowledge of the technical conditions.

ing points should be covered: 1, duration of contract, in years; 2, approx amount be delivered; 3, rate at which the ore is to be delivered (if known); 4, point at livery is to be made from miner to purchaser; 5, arrangement respecting freight method for weighing the ore; 7, determination of moisture; 8, arrangement for tation of both seller and buyer; 9, manner in which the ore is to be sampled, and paid for; 10, number of final samples to be taken, and disposition of same; 11, d analytical methods to be used in determining the ore constituents; 12, settle- splitting limits (Art 1); 13, mode of settlement when determinations by seller er are within the splitting limits; 14, arrangement for umpire, if difference ex- e splitting limits; 15, mode of settlement in case samples are sent to umpire; s of payment for metals; 17, the effect, if any, which fluctuations in market price e on terms of payment; 18, the effect, if any, which the value or composition of will have on these terms; 19, dates on which metals are to be paid for; 20, state- payments to be made and penalties to be levied for certain ore constituents; tment charge on which settlements will be based; 22, amount which will be ad- if desired) by purchaser to seller, at any time after settlement has been agreed d before final payment is made; 23, possible conditions which shall be considered reasons for failure to deliver or receive ore under the contract (Sec 29).

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[illegible] H. S. Joseph, Amer Min
[illegible] L. S. Austin. (*E*)
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[illegible] contracts, and summa
[illegible] received by producer is incr
[illegible] cost of treatment a
[illegible] the Smelter's Standpo
[illegible] producer from ores of
[illegible] costs and losses
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SECTION 33

GOLD AMALGAMATION AND CYANIDATION

BY

EDWARD L. DUFOURCQ

CONSULTING ENGINEER, NEW YORK CITY

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GOLD AMALGAMATION

1. Milling Practice

lgamation of gold ores in modern plants is confined almost exclusively to AMALGAMATION, in connection with crushing by stamp battery. PAN AMATION (formerly common), and certain special processes now employed unusual circumstances, will not here be described.

ning and grinding. Ores must be reduced to such fineness that the gold or silver ently freed from gangue to be attacked by mercury. In most mills, after amalga- the pulp is ground still finer, for further treatment by cyanidation or otherwise. in mills, plate amalgamation has been used after tube-milling in preparing ores idation. Crushing is usually by jaw-crusher or gyratory crusher, followed by attery, for details of which see Sec 28, Art 4 to 7. For grizzlies, screens, feeders, mpling devices, see Sec 28, Art 15, 31, and 32; for variations in practice as to size pe of mortars, wt of stamps, and order, height, and speed of drop, see Sec 28, Art 7.

eral principles to be observed in designing and operating the stamp-mill atory to plate amalgamation: (*a*) systematic and accurate sampling of ; (*b*) maintenance of uniform feed, as to amount, size of ore, and (so far ible) as to average values; (*c*) close control of feed water; (*d*) maintenance nstant height of discharge above level of the stamp dies; (*e*) care of bat- reens.

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...fected, the cost of ar... are reduced by ... limited by size of sc... given ram and height of drop... and height of discharge from t... in which it must be crushed ... and height of discharge; the fine ... length of time a given particle of or... with mercury in the mortar. Als...

...types of practice have develope... character of ore.

...mills, stamps weigh 850 to 1 050 lb, drop 6 ... of discharge of 4 to 6 in. Some later mill ... are usually 30 to 40 mesh. In Colo, stam... in, at 90 to 40 drops per min and a 12 to 1... within the mortar) is still practiced, the o... small capacity per stamp. In So Dakota, ... with 8½ to 9.5-in drop, 85 to 90 drops per m... In So Africa, stamps of 1 750 and even 2 000 lb ... are run to 1 000-lb stamps. In New Zealand ... sometimes as low as 3.5 in.

Modern tendency is to abandon entirel... coarse, and to use heavy stamps with lo... is that the development of the cyanide proces... plates, makes possible greater ultimate reco... obtainable by stage-crushing, since for cyani... ground than can be economically effected in ...

...ovement of inside amalgamation has been p... crushing, which causes rapid scouring of the inside ... amalgamation has been subordinated to proper prepar... upon fine grinding. The decreasing output of pre... by production figures given under Cyanide Proces...

2. Inside Amalg...

Requisites: mortar must be wider than for o... between stamps and back of mortar; for high bat... low discharge is necessary, with which inside plate...

Inside plates are of 2 kinds, front and back. B... of copper, not silver-plated, generally 8 to 12 in w... They are set 3 to 4 in above top of the dies, in a rec... They are difficult to remove or to clean in place. ... upon the chuck-block for the full length of mortar o... may be either flat or curved, and are inclined about ... were down, the chuck-block with its attached front ... wood about 2 in thick, originally placed under it. ... against scouring by placing over them, and about o... OUTSIDE PLATES (Art 3) are always used in connec...

Mercury is fed to the battery at rate of about ... Appearance and condition of the outside plates det... too bright, with streaks or drops of Hg on the surf... surface becomes hard and dry, more Hg is required ... the mortar settles in the sands between dies and col... ore is very rich the battery need not be cleaned up ... plates must usually be cleaned and dressed oftener; ... a strip is removed from under the chuck-block, to ... worn dies.

3. Outside Amalgamation

amalgam collects inside the mortar, but less than with inside amalga-
Art 2), since the discharge is lower and the mortar narrower.

le plates comprise: splash plates, deflecting plates, lip plates, apron
d sluice plates. The first two are to regulate the flow onto the lip and
ites, but they are often omitted. As the amalgam caught on SPLASH
hard and does not easily break away, these are preferable to inside plates.
ES are the full length of mortar opening and 6 to 8 in wide; the wider
r. They rest below the bottom of the screen and project over the head
on plate, sometimes feeding the pulp down to the apron plate through a
and distributing box. APRON PLATES are of the full width of mortar,
nnealed copper, and almost always silver-plated. They rest on heavy
ables, which, though adjustable, should be held rigidly and be non-
It has been claimed that by connecting apron plates with the mortar,
g aids the gold to settle on the amalgamated surface, but best practice
is because the amalgam formed is too hard. Amount of silver used in
aries from 1 to 2 oz per sq ft; in some districts as high as 5 oz. Plates
ered after 1 to 3 years' use. In some mills replating is not practiced;
s are put in after several years and old plates are melted down. SLUICE
e rarely used in recent mills. They are about 0.5 the width of apron
d placed in 2 or more sluices or launders, taking the tailings from the
tes. Being narrower and steeper than the aprons, the flow of pulp is
that they catch but little gold.

f plates varies from 1.5 sq ft per ton milled per day (So Africa) to 10
t (western U S). Referred to number of stamps, the plate area may be
10 sq ft per stamp (one 4 by 12 ft plate per battery), the other extreme
sq ft of plate per stamp, arranged in 4 successive rows.

rage for 1 000 stamps in Homestake mills, So Dakota, is 48 sq ft per stamp;
ve mills have 10 sq ft in the first row, 14 in the second, 15 or 16 in the third and
the fourth row. Generally a succession of plates is used with drop between
ough or mercury trap being sometimes placed below the bottom edge of each
ercury traps are advisable, as apron plates may then be kept softer with less
losing Hg or amalgam.

ry traps may be simply transverse troughs, a few inches deep, with
ip discharging onto the next plate, the bottom of trough being provided
ve for discharging collected Hg. A V-shaped, conical or hemispherical
mmoner, receiving the pulp through a pipe reaching nearly to the bottom
verflow going to the distributing box for following plate.

of apron plates varies from 1.5 to 3 in per ft; best practice, about 2 in.
requires more water. Tables for supporting plates may be adjustable,
g the slope.

carrying much heavy base mineral, slope should be increased to avoid too
of pulp on the plates. If total length of plates is short, for the quantity of ore
r battery, and plates are set flat to insure slower flow, the amalgam coating
contain enough Hg to become soft, and thereby increase losses.

ng and care of plates, with maintenance of maximum battery output,
principal factors in successful amalgamation. Hard, dry plates may
rse gold, but fine particles will be lost. If plates are too soft, both Hg
gam will break away from the surface and be lost unless caught in the
n general, unless the gold is very coarse, too much Hg on the plates is
an too little. If, in SCRAPING AND DRESSING, the amalgam is too thor-
moved, exposing the plated copper surface, gold is not so effectively
when a little amalgam has been left.

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As hard amalgam forms on LIP AND SPLASH PL/
oftener than once in 10 days or 2 weeks, unless
moved for dressing, and replaced by reserve fres
up of the battery. These plates require little at
are usually dressed twice a day. On very low
but on high-grade ores, or ores containing subs
may be required every 3 hr. In some mills, ha
tar, a diverting launder throws all the feed to o
cleaned up. Single plates may be divided int
and one-half dressed at a time. This avoids h
tery water must meanwhile be reduced. A c
plates at the monthly clean-up no more amalg

Details of dressing plates. Hg is first spri
amount also on lower end, to soften the amalg
the amalgam is scraped up from lower end to
scraper; or a rag may be used when amalgar
scraping is done with an amalgam knife or chis
plates. After the amalgam is removed, fresh H
used on upper end, as lower end should be kep
small iron bottle of gas pipe, the mouth of which
dealt out daily to the amalgamator, should b
weak cyanide solution on the surface, rather th
To avoid loss or fouling, Hg should never be left
freshly wet with Hg, the plate is well rubbed u
on plates should be cleaned with weak cyanide s
rubbing with NH_4Cl solution, or weak acid. A
ute the mercury over the plate, riffling the su
wise. Some millmen rub down all the riffles af
too much base metal is retained by them, resu
plete dressing of plates may take place only at
the ore), they must be inspected at least every

Condition of mercury. The Hg fed t
free from impurities, which SICKEN the ME
that will not amalgamate the gold.

The sickened condition is due to fine particle
manganese or iron sulphates, arsenides, antim
or grease must be carefully kept away from
through chamois, canvas, or muslin bags will
sickens it must be cleaned with weak acid, lye,
using again. Addition of a little sodium amal

Flouring. When mercury is held too
or when too much Hg is fed, FLOURING wil
ing up of the Hg, in which condition it will
reunite, as the surface tension is very grea

Clean-up, which may be made monthl
ore), consists in scraping and dressing the

The battery is first cleaned out, by cutting
sands with water only, until the stamps just
then hung up, being raised higher than ordin
more easily accessible. The upper loose sands,
and put aside for returning to the battery afte
the dies are dug out and taken to the clean-
washed and panned over the head end of apr
injure the surface of the plates. Kettles shoul
them with sands to be removed, to avoid loss o
then removed, and all amalgam scraped from
plates are used (Art 2), they are removed to th
and freshly dressed ones put in their place bef

leaned of mercury and amalgam. Dies are then replaced, reversing their posi-evenly worn, or are replaced by new dies when necessary. Skillfully handled, a eed not be hung up more than 40 min to 1 hr.

ain mills the amalgam on the first apron plates is first softened by STEAMING. ne by covering the plates with a wooden cover, held a few inches above them, tting steam under the cover. Workmen are in danger of salivation, if they stand lates while being steamed. After the apron plates have been scraped, the traps d of the collected sands, and Hg and amalgam are drawn off through cocks in f traps. Fresh Hg is sometimes put into the traps before starting up again. cted amalgam is removed to clean-up room, which should be provided with onstructed that Hg can be drained from them without risk of loss. Iron kettles or carrying mercury or sands to be treated.

NDS may be treated with additional Hg and water in a clean-up barrel or pan, panned down over the sink to separate the amalgam, the washed sands being to battery. The amalgam recovered here, and that scraped from the splash, side plates, is also washed with addition of enough Hg to make it liquid. Par-on are removed with a magnet. In large mills, the amalgam is worked over in the arrel or pan, with excess Hg; in small mills, it is ground in a wedgewood mortar.

d amalgam is then put in a conical funnel of canvas or heavy unbleached muslin, which the excess Hg drains off; the remaining amalgam is squeezed by hand hamois or cloth, or in large mills by a hand or power press. Chamois is expen-e as it wears out rapidly. Skimmings and dross from surface of the Hg strained ining amalgam are saved for separate treatment. When the ore contains much antimonide or arsenide, the mud settling out of the washings, and the dross from malgam, may be saved and shipped to smelters. Amalgam, squeezed quite dry contains 25 to 40% bullion (the lower proportion when the silver is relatively he squeezed amalgam is finally retorted for distilling off the Hg (Art 4).

4. Retorting and Melting

ts, when of large size, are horiz C-I cylinders built into a brick furnace nd with a pipe from the rear end to an open Hg receiver. This pipe is ed by a larger pipe or jacket, in which cold water is circulated for con-he Hg. The end of the pipe is covered with wet cloth or flannel, dipping receiving pot which contains water.

el of the condensed Hg or water in the pot must never be allowed to rise high cover the end of the pipe; otherwise, if the temp of the retort falls, water may back into the hot retort, causing an explosion. Small retorts are vertical C-I condenser pipe connected to the cover. Retorts are lined with chalk or wood re charging, and the lumps of amalgam are often wrapped in paper, to keep n from adhering to the iron. Vertical pot retorts are filled not over ⅔ full of to avoid having particles of amalgam thrown up into the pipe in case the Hg to overheating. This would clog the pipe, and might cause an explosion.

ging retort. Amalgam is charged in horiz retorts in semi-cylindrical s (usually 4 in number), which are slid into place. The retort cover is th a mixture of fire clay and wood ashes, or clay with a small admixture slimes. Or, gaskets can be used, cut from thin asbestos sheets, heavy first applied to the joint faces. The cover is clamped on securely with nd steel wedges.

g. The retort is gradually heated by wood or coal fire or by gasolene ng to design of the furnace), water running continuously through the er jacket, until Hg begins to pass freely into the receiver. Heat is then d, but should be maintained sufficiently to distil the Hg continuously.

distillation ceases, the heat is raised to a dull red and so maintained for about 0.5 which the fire is allowed to die out, and the retort to cool before it is opened. he final heating the condenser pipe should be occasionally tapped to dislodge the s of Hg. Even with large retorts, distillation should be complete in 5 to 6 hr. ll pots, in say 3 hr.

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Melting. The retorted mass (RETORT ME in a coke or coal-fired (or gasolene) furnace, control of temperature.

The crucible is gently heated before use to driv ported in the furnace to avoid tilting. After raisi is first charged and melted, after which the retort care being taken not to drop them in with a spla charged about 0.75 full. FLUX generally used is amount of sodium bicarb and a little niter. If t stirring with a heated iron rod will cause format usually contains enough particles of metallic iron niter must be used to oxidize base metals, but an In presence of much metallic iron, SiO_2 may be ad a viscous slag, which may carry shots of the bulli or more flux to produce a clean bar. After being be well stirred with an iron rod, to insure a homo if any) are skimmed off, or the pouring may be c

Molds are of C I, well heated, and co mixed with heavy oil; or oil alone, melted

After cooling sufficiently the bars are plunged in H_2SO_4 or HNO_3. The slag (and matte, if any) a numbered, weighed, and drilled for assay sample. side (top and bottom) of the bar and about 1 in d ings being rejected. The matte is saved for separ enough to ship to smelter. Slags are returned to t free from graphite and returned to the battery. should be from 30 to 40% by wt, according to amo is slight, if amalgam has been properly cleaned.

5. Salivati

Mercurial poisoning may result from a from breathing Hg vapor, or swallowing sol

General symptoms are: soreness of gums, t saliva, swollen, inflamed gums and foul breath, la ing of face, with fever and depression. ACUTE M tremors of arms and hands, irregular heart action a ally in loss of teeth, necrosis of the jaw bone, and

Salivation of the workmen in an amalgamating be worn when cleaning amalgam. Care should when plates are being steamed (Art 3), when Hg leaky retorts. Men who may be exposed should and kidneys, and pores of the skin should work principal means of eliminating Hg from the sys poisoning, where mercurial salts have been swall tered, or the stomach may be washed with albur

CYANIDE P

Introduction. The process is based on the sol cyanide of potassium or sodium. The first comm 1887–8 (McArthur-Forrest process); treatment o mills were built: 1889, at Crown mine, New Zeal Validity of the McArthur patent was set aside, F

Later improvements. CHEMICAL: precipitat electricity; use of bromo-cyanogen for dissolvin bellious ores. MECHANICAL: finer grinding; aera in classification, and thickening or dewatering pu

tinuous cyanidation. Since 1903, the development of the process has been improved mechanical appliances; and to the end of 1916, practice has tended vards coarse initial grinding or stamping; very fine tube-mill grinding of the entire air agitation and filter-pressing of pulp; use of zinc or aluminum dust for pre- . The superiority of continuous cyanidation, using counter currents, remains onstrated.

ing importance of cyanidation, as compared to amalgamation, is shown by following tement of oz of total output (Mineral Resources of U S, 1914):

mation	1911	1912	1913	1914
ld	1 120 344	1 003 470	927 572	924 253
ver	941 155	795 755	428 844	271 417
ation				
ld	1 259 400	1 386 526	1 348 322	1 384 934
ver	8 781 552	11 728 730	14 090 407	15 399 836

luction from other sources, the output from placers, chlorination, and smelting . 50% of the gold, 84% of the silver; 1912, 46% gold, 81% silver.

6. Adaptability of Ores

all gold and silver ores can be treated by cyanidation. In case of gold irides may cause interference, but can be decomposed by preliminary Solution of Au in tellurides by bromo-cyanogen has been successful.

ering conditions. OXIDIZED CU ORES cause high consumption of cyie to ready solubility of oxides, carbonates, and hydrates of Cu. SULf Cu, Ni, Co, Pb, Zn, and Fe react but slightly with cyanide solutions. fere by mechanical inclusion of the Au or Ag, preventing direct attack de, unless very finely divided. Some sulphide combinations of Ag with Pb re only slightly attacked by cyanide, even when finely ground. SOLUBLE :S (especially alkaline sulphides) interfere seriously, both by combina- cyanide and by re-precipitation of the Ag and Au. ARSENICAL AND IAL ORES often cause high cyanide consumption, and sometimes form comaving a strong reducing action in the working solutions. Preliminary may favorably prepare such ores for cyanide treatment. MANGANESE l with Ag often prevents successful cyanidation; a preliminary roast ease extraction, but the cost of this treatment has sometimes been pro- or the results have not been satisfactory. METALLIC MERCURY (often n tailings from amalgamation) is not readily dissolved by cyanide, and, in antities, may assist in precipitating soluble sulphides; present in larger s, Hg causes waste of Zn in precipitation boxes. An ore containing Au and Ag in COARSE PARTICLES, not pulverized by fine grinding, would ılt to cyanide with financial success, because of the length of time re- dissolve the metals; such ores can generally be treated by amalgamaowed by cyanidation of the tailings. At the Nipissing mine, Cobalt, H-GRADE SILVER ORES are amalgamated in a closed tube-mill, in cyanide a similar combination process has proved applicable to other ores also. ACEOUS MATTER, if ground with the ore, increases cyanide consumption ts a reducing action, whereby precious metals are re-precipitated before reach the precipitation boxes.

coming adverse conditions. The refractory nature of an ore can often ome by fine grinding. As yet, chemical analysis alone can not deter- adaptability of an ore.

tory experiments must be conducted to determine: possibility of leaching Au om coarser sands; necessary fineness of grinding: acidity of pulp and amount of ired to neutralize it; amount of soluble sulphides in the pulp and quantity of ate or oxide to be added; amount of cyanide consumed in treating the ore for

time required to effect satisfactory extraction. An ore not a
ment, owing to presence of much sulphide (either mech
metals or forming excessive quantity of soluble sulphides in
by removing part of the sulphides before fine grinding.

Working test, in addition to laboratory experime
to determine whether the extraction by cyanide is e
other process, to warrant the extra cost of prelimina
cost of chemicals, possible high consumption of cyani
and cost of plant. (See Sec 31, Art 8.)

7. Chemistry of the Proc

Cyanogen (CN or Cy), a compound radical, is a color
metals. In manufactured forms it is available as KCy or
all metals to form a single cyanide, and with many metals

Chemical reactions occuring during solution of
process have not yet been precisely determined. T
the ore, presence of base metals, and conditions unc
Following fundamental reactions are generally acce

$$2\,Au + KCy + O + H_2O = 2\,AuKCy_2 + 2\,KO$$
$$2\,Ag + KCy + O + H_2O = 2\,AgKCy_2 + 2\,KO$$

Similar equations apply when NaCy is used. They all
with metallic Au and Ag, assuming the necessary presence
an intermediate reaction takes place before formation of
viz: $2\,Au + 2\,KCy + O + H_2O = 2\,AuCy + 2\,KOH$,
yields $AuCy + KCy = KAuCy_2$.

When silver sulphide is attacked, the reaction may be
$2\,KAgCy_2 + KCyS + 2\,KOH$, or $Ag_2S + 4\,KCy = 2\,K$
cyanate (sulphocyanide) is often found in working soluti
mer of these two equations.

Variations in reactions are caused by presence
or other salts, and by soluble salts of alkaline earth

Oxygen is absolutely necessary for the reaction, and
supplied chemically by addition of $KMnO_4$, MnO_2, o
generally costs more than to bring O into contact with th
by successive leachings and drainings. All processes ar
Cy for Au or Ag than for K or Na, in effecting solution
affinity for Zn or Al than for Au or Ag, in effecting su

Relative solubilities of certain metals in cyanide sol
the same as their electrochemical activity in such solut
Zn, Cu, Au, Ag, Hg, Pb, Fe, Pt; hence any metal would
the above order, and would precipitate it from a cyani

NaCy *vs* KCy. NaCy is generally preferred to K
Cy per unit of wt, 53% against 40%; the dissolving p
greater than that of KCy. Commercially, the strengt
of KCy; the strength of working solutions being also

Oxidizing agents. A number of oxygen-bearin
solutions, increase the rate of solution of precious
increased the total extraction: among these are K
Na_2O_2, and MnO_2. Their beneficial effect appear
of interfering substances, and rapid supply of n
use may result in destruction of cyanide.

Nascent Cy. K_3FeCy_6 and BrCy react, without prese
solution, and nascent Cy has greater affinity for precious
as KCy or NaCy. Liberation of Cy by these salts is

$$K_3FeCy_6 + KCy = K_4FeCy_6 + Cy,\ \text{and Br}$$

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tter equation represents the supposed basis of the BrCy process, in which the n presence of Au is stated to be: $BrCy + 3\,KCy + 2\,Au = 2\,KAuCy_2 + KBr$. ions in this process are not thoroughly understood; probably the greater affinity t Cy for Au makes it possible to dissolve Au in such telluride ores as are not at- simple solutions of KCy or NaCy. So far as determined, no bromide of Au is y the use of BrCy.

errocyanides are present in working solutions, $HgCl_2$ may be used to disintegrate formation of double mercuric cyanides, which dissolve precious metals without l O. The cost of $HgCl_2$ limits its use. Certain patented modifications of the rocess claim that an electric current in the agitation tanks aids solution of pre- ials, especially tellurides; the efficiency of such processes is not proved.

gth of solution. The max solvent action of alkaline cyanide solutions to take place with solutions containing between 2 and 5 lb of KCy per a slightly less amount of NaCy. With O present in excess, weaker solu- apparently more efficient. A stronger solution is therefore necessary lation than in agitation treatment, because in the latter air reaches the re quickly and effectively.

ive solubility. It has been held that cyanide solutions dissolve Au and Ag faster or other base metals; it is practically certain that this is only apparent, approxi- ne same amounts of precious and of base metals being dissolved in a given period, ne conditions. Thus, in an ore assaying 20 oz Ag per ton and 0.2% Cu, if all the n equal amount of Cu were dissolved in a given time, there would be 100% ex- of Ag, but only 3.4% of the Cu; this would occur if both metals were in an equally e of division and in chemical combinations attacked with equal ease by the solu- he apparent selective action of cyanide for precious metals is accentuated by the mechanical conditions of precious and base metals are never the same in an ore; als are almost always in coarser particles, and, as their sulphides are less easily . less base metal is dissolved in a given time.

been supposed that presence of certain metals retards solution of Au and Ag, but of such interference the cause is apt to be elsewhere. Thus, in treating certain n New South Wales, in which the Au was contained in Sb_2S_3, it was found that aused no consumption of cyanide except when the solution was too alkaline; in a olution, the only difficulty lay in disintegrating the Sb_2S_3 to reach the fine Au.

ed solution frequently yields higher extractions in laboratory tests; commercially erally found that the increase is not appreciable, and excessive heating of solutions troy part of the cyanide, while also increasing its solvent action on base metals, ailed losses of cyanide. Heating of solutions also expels oxygen, thereby necessi- ncreased aeration of pulp.

mistry of precipitation. Recovery of precious metals from cyanide ns is based, theoretically, on the reaction: $2\,KAuCy_2 + Zn = K_2ZnCy_4 +$ Practically, precipitation takes place only in presence of free cyanide, $KAuCy_2 + 2\,KCy + Zn + H_2O = K_2ZnCy_4 + Au + H + KOH$.

rfering conditions. In actual operation there is always an excess of Zn, s also dissolved by the KCy as K_2ZnCy_4, without necessarily precipitating . In the precipitation boxes, other metals are generally present, forming couples with Zn, which electrolyze the water, liberating O and forming H_2. This combines with free cyanide, forming a soluble double cyanide ly an alkaline zincate). Unless excess of cyanide be present, precipitation $OH)_2$, ZnO, or perhaps $ZnCy_2$, will take place, covering the Zn shavings white deposit, and impeding precipitation of precious metals.

alkali, or excess of cyanide, prevents formation of this white deposit by redissolving precipitate; hence a solution to be precipitated should always contain excess or be strongly alkaline But, the double ferrocyanide of Zn and K, which frequently on the Zn shavings, is but slightly soluble in alkaline or cyanide solutions, and rm enough deposit to interfere with precipitation of the precious metals before the rings have been completely consumed, thus necessitating an earlier clean-up than ise required.

To a slight extent, Al_2O_3, SiO_2, CaO, and Fe precipitation of Au and Ag. Cu interferes still ings. If the shavings in the first precipitation nearly all the Cu can be precipitated before th which the Au and Ag are to be collected. Zn m solution of Pb $(C_2H_3O_2)_2$ (lead acetate); Pb shav first compartments, to furnish a Zn-Pb couple. the Zn; while often causing trouble in subseque generally interfere with the precipitation reactio

Continued re-use of solutions ultim salts, and even if proper strength of free cya become foul and lose part of its dissolving

It has been advanced that the fouling of soluti cyanides of Zn with other alkaline earths beside base metals and alkalies. It has been proposed t cyanides, and then some hydrate to precipitate al cyanide.

Aluminum dust has been successfully especially when the ore contains considerabl is probably due to the fact that Al does n solution by forming a Cy compound; hence As is avoided.

The reaction is possibly: $6\ NaAgCy_2 +$ $2\ Al(OH)_2$. The $Al(OH)_3$ is redissolved in the reaction, with evolution of H, is: $2\ NaAgCy_2$ $Na_2Al_2O_4 + 4\ H$.

Charcoal, especially soft-wood charcoal, has at time of precipitation is not yet understood, bu are decomposed by reaction with gases occluded or CO, which act as reducing agents. There see soluble double cyanides.

Amalgams of Na and other metals, and amal fications of the cyanide process, to collect precip such processes is the passage of a low-voltage el solution, either before or after separation from t are the Gilmore-Young, Pelatan-Clerici, Riecke trolytic precipitation throws down many of the ba the application of electric precipitation to solution filtering, this advantage is more than offset by th

Cyanicides; loss of cyanide. A cyan either combines with Cy or decomposes a loses its dissolving power. Some loss of Cy without presence of cyanicides. KCy an slowly decomposed by moisture.

Loss by HYDROLYSIS is represented by the rea Cy escaping by volatilization. Loss by ABSORP $4\ H_2O = K_2CO_3 + (NH_4)_2CO_3$. In presence of $KCy + CO_2 + H_2O = HCy + KHCO_3$. In pra above equations are very small.

Losses due to cyanicides in the ore ma chiefly by soluble sulphides, sulphates, and are insoluble in water but react with cyanide in oxidized or semi-oxidized ores, are apt t cyanides of Cu or Fe and K or Na. Free aci obviated by maintaining sufficient alkalinit

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ores, K_2S or Na_2S is formed; if then the cyanide solution be allowed to ak, these alkaline sulphides may accumulate sufficiently to reverse the and re-precipitate Ag as sulphide; or the alkaline sulphides may oxidize hates, or may form thiocyanates, thus consuming either the available O y, and decreasing the dissolving power of the solution.

Fe_2SO_4 and $CaSO_4$ decompose alkaline cyanide solutions; arsenates of Fe and als also react with Cy solutions and destroy their dissolving power. While FeS_2 react with cyanide quickly, nor to an appreciable extent, ferrous sulphide and nd ferric sulphates combine actively. Sulphides of Cu and Zn are practically ed by cyanide solutions, but their carbonates, oxides, and sulphates react rapidly.

8. Outline of the Process

ery of Au and Ag by cyanidation consists broadly of 4 operations: (*a*) ion of the ore; (*b*) solution of precious metals; (*c*) separation of dis- netals; (*d*) precipitation and refining of product. After preparing an pulp may be classified into coarse and fine products, SANDS AND SLIMES, ay be treated separately; or, all the ore may be ground very fine and together by the ALL-SLIMING process. Variations in preliminary treat- ll be described in general outline, but many ores present problems re- special modifications.

s, when treated separately, are usually charged into vats or tanks where submitted to washes of cyanide solutions, which are then separated from ed pulp by filtration, and the precious metals in solution are precipitated.

es (pulp passing a 200-mesh screen, or, in some cases, 80% of which will -mesh) can be treated by cyanide solutions, followed by settling and tion; or can be agitated with a cyanide solution, mechanically or by ed air. The pregnant solutions can be separated by decantation after or by passing through a filter-press, their subsequent treatment being e as for those resulting from sand treatment.

ipitation. After solutions have passed through a filter-press (or have anted), it is sometimes necessary to clarify them further before allowing pass to precipitate. In most cyanide plants clarified solutions are ver Zn shavings to precipitate the metals.

e, use of Zn dust for precipitation is increasing, and quite recently, of aluminum e barren solution then passes through a filter-press to separate it from the precipi- or Ag. Other precipitation methods are by electrolytic action, and by charcoal. te is then dried, melted with fluxes, refined, and cast into bars.

al processes and modifications. COMPOSITE METHODS. Many attempts have de to dissolve and precipitate in the same vat, chiefly by using low-voltage electric ; none has been entirely successful. Variations occur principally in the type of which must be so selected as to present a sufficiently large surface, to which the d metals will not adhere too strongly, and will not produce too base a bullion. on, tinned iron, Al, amalgamated Zn or Cu plates, Hg and amalgams have been A fundamental objection to such processes is that deposition of other than the metals on the cathodes occurs, thereby preventing further precipitation. The process, used in Australia, substituted a weak solution of HCy for cyanide, being formed, to the extent of 0.05 to 0.1%, by action of H_2SO_4 on KCy. After of the metals, limewater was added to neutralize before precipitation by Zn. cess was found to be inapplicable to ores containing $CaCO_3$.

lgamation with cyanidation. When Ag and Au occur as metallic par- hese are sometimes too large to be dissolved by cyanide in the allotted In a tube-mill, however, the particles are so flattened that they can be ed in relatively short time. One of the chief disadvantages of plate mation, after battery crushing and before tube-milling, is that the fine

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CONCENTRATION BEFORE TUBE-MILLING, bes
in Art 8, also removes heavy cyanicides.

Tube-mills and conical mills are gener
28, Art 10). For long tube-mills, roller b
greater care to maintain in alinement. Lengt
diam, 4 to 6 ft. Modern tendency is tow
Claim is made for longer mills that, as the
grinding results. If rate of feed for a given
quent slow discharge, a shorter mill will p
Generally, for relatively coarse feed, the dia

Fig 1. El Oro Tube-mill Lining

Total load of pe
terior volume of the
rev per min; higher
sumption and usel
without compensat
new tube-mill, the
sist of 50% No 3 Danish pebbles (or equiva
No 2, and about 17% No 1 size. Subsequent
largest or No 3 size. If the pulp is fed
very coarse, the proportion of coarse
pebbles may be increased.

Feed should be dewatered so as to leave 30 to 38% moisture in the pulp, at which consistency the best grinding is secured. Tube-milling can generally be so regulated that all the pulp discharged can go direct to the agitation or other cyanide treatment; but in large mills it may be advantageous to classify after the first tube-milling and return the oversize for regrinding in the same or other tube-mills.

Hardinge conical mill operates like the tube-mill, but occupies less floor space and some economy in power is claimed (Sec 28, Art 10). The theory of its increased effic is that the pebbles of larger diam seek the largest diam of the cone, and there grind the coarsest ore, while the smallest pebbles do the finer grinding near the discharge.

Substitutes for pebbles. Where hard quartz is available, about the size of coarsest pebbles, it may replace grinding pebbles wholly or in part. This is advantageous in saving the cost of pebbles, and because the quartz itself generally carries recoverable values in Au or Ag. Consumption of quartz fragments from the mine is about 3 to 5 times that of Danish pebbles.

Lining of tube-mills may be of SILEX BLOCKS set in cement, or of steel ribs which hold part of the pebbles ag upon which grinding is done. This is know

Fig

A modification is the KOMATA LINING (Fig 2), about 18 in, with C-I or manganese-steel liner

bars and grind on the surface of the plates. El Oro lining is the cheapest to t and maintain, and the quickest to replace.

harge from tube-mills is through an opening in the trunnion, or through discharge receiving the ground pulp through slots in the cylinder head-Trunnion discharge generally results in finer grinding of the heaviest particles than when the discharge is through slots. Within certain decrease of moisture in the charge increases the mill capacity.

mills resemble tube-mills; they use steel balls in a revolving cylinder, but e through slots between the lining plates and then through an outer They are driven more slowly than tube-mills, admit of coarser feed, use ver, and the wear is less, but they do not grind so fine.

of mills. A tube-mill 4.5 ft diam by 12 ft long requires about 45 hp ; for a mill 5 by 22 ft or 5.5 by 18 ft, as much as 90 hp must be on to start. After attaining full speed the power decreases 30 to 40%. o 5 ball mill about 25 hp is required for running; No 8 mill, about 55 hp. Hardinge mill takes 15 to 20 hp; 8-ft mill, 30 hp or more. The above vary widely with rate of feed and speed of revolution.

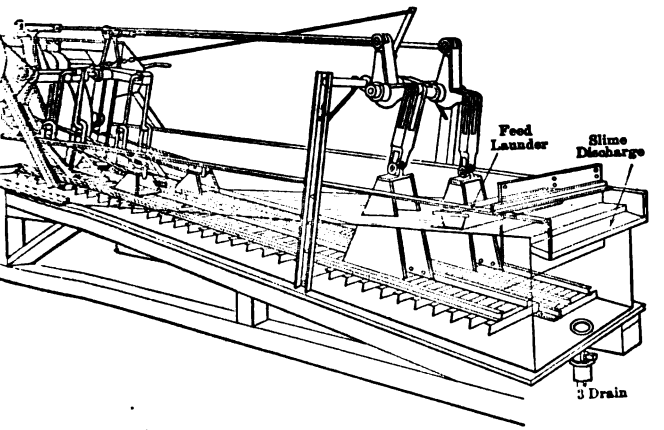

Fig 3. Dorr Duplex Classifier (Model C)

t of tube-milling varies greatly, according to cost of power, labor, and s, and is dependent also on character of ore, size of feed, and fineness of ig. Many mills show costs between 9 and 18¢ per ton. Under favorable ions, 12¢ is an aver figure, where power represents about 25% of total cost, 20%, and pebbles and repairs 55%. (For details of fine-grinding mills, c 28, Art 8–11.)

ssifying. The separation of coarser sizes from the finer, as preliminary king 2 sizes for separate sand and slime treatment, or for removing the ntly fine slimes from the material to be ground in the tube-mill or to be ed as oversize, can not well be done by screens, and classifiers must be used.

zkasten, cone, and hydraulic classifiers generally cause the introduction of too much when they size effectively, and modern tendency is to use mechanical classifiers. classifiers have large capacity when the proportion of slimes is great, but do not

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effect so positive a removal of the coarser particle
vent clogging and to maintain adjustment. Of th
appears to be better adapted to coarser feeds. D

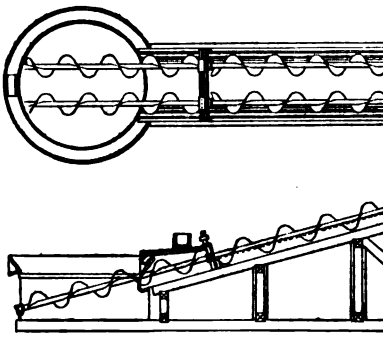

Fig 4. Ovoca Classifier

closely related to the classification of sands a
to produce sorting or sizing, but to rid the sa
ide solution of proper strength can be used.
justed to the work of dewatering, or a simp
masonry can be made of large diam relative

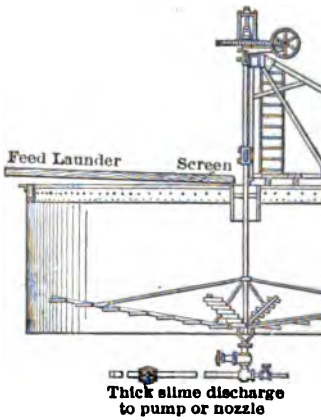

Fig 5. Dorr Continuo

water will overflow into a launder at the upper
should contain only enough moisture to enab
the percolation tanks (for sands), or to the agi

Dorr thickener (Fig 5) is successfully used fo
going to agitation or decantation tanks. It may b
having a circular overflow launder and a central di
ing slimes and admitting water or solution under p

es 4 radial arms with short rakes, which in revolving slowly move the settled ards the central discharge. Bottom of tank may be slightly conical, conformnation of the radial arms, or, if the tank be flat, the accumulated slimes will soon nical bottom surface. Feed enters the tank through a short central pipe, dislow the overflow level, to avoid surface currents. About 0.25 hp is required to This machine has also been used partially to decant very rich solutions from nes that are sticky and would filter slowly (see Sec 28, Art 18).

10. Sand Treatment

. **Sands,** after separation from slimes, are charged into tanks fitted -bottoms, consisting of a grating overlaid by cocoa matting and canvas. s a layer of sand is charged on the filter proper, and over this is spread to limit shoveling of the tailings when the tanks are discharged.

grating is arranged to insure free circulation of the filtered cyanide solution on ottom towards the solution discharge pipe; there should be always enough origh the supporting frame of the bottom grating to allow all parts of the tank drain freely towards the discharge. The canvas, which is placed over the cocoa s calked into position around the edge with rope; it is also calked around the ate when the tank is discharged through the bottom by shoveling, by sluicing, chanical excavator.

of tank depends on fineness of sand and amount of slime carried. With clean d, tanks may be deeper; in practice, depth is 6 to 10 ft. Depth is decided upon rimenting with the ore to be treated, to determine the RATE OF PERCOLATION. should be between 1 and 3 in or more per hr for satisfactory results and the pulp so classified that the resulting sands should have at least this rate of percolation. **of treatment** being determined, and rate of percolation and depth of tank de- **area of tank** is made such that one or a number of tanks will hold approxie day's mill run.

tion. The tank is generally filled with wet pulp from the classifiers, stributed by some automatic device. This usually consists of revolving s with perforations, or modification of this device. After filling, the drained, and, if crushing has been done in solution, the solution is stored until it is high enough in precious metal content to be effectively pre-

After the first draining, the first and generally the strongest mill solun into the tank to cover the charge. Practice varies as to admitting on at top or bottom of the tank.

ents determine the strength and number of the successive cyanide solutions used hing with clear water. A variation of practice is to use downward percolation; ly drawing off the cyanide solution from the bottom, and adding fresh solution until the desired extraction has taken place. But this practice does not aerate well as intermittent flooding and draining of the charge, which generally gives traction.

ary precautions for percolation of sands: charge should be evenly distributed s and free from slimes which pack and prevent passage of solution.

ions of practice in percolation of Au ores show following range: drainter from wet charged tank, 8 to 18 hr (aver, 13 to 14 hr); leaching, 12 ver, 15 hr); washing, 16 to 22 hr (aver, 20 hr). Most mills use 1 strong ak-solution treatments, and 1 with wash water. Some mills have intraction by forcing air up through the filter bottom before each appli- solution. Pressure used is only sufficient to aerate slowly without hannels in the sands.

rging the tanks by shoveling costs most, and is used in small plants er is scarce. Sluicing is cheapest, and is used where water is plentiful. al excavators (radial arms, carrying circular plows that are lowered ands and move them to the central discharge door) are high in first

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cost and are adopted only for large plan
between that of sluicing and shoveling.

Size of material treated, AMOUNT and ST
TREATMENT, must be determined experimentall
to sand treatment, due to its porosity, freedo
Au, may be successfully leached by percolation
ceous ores have given good extraction when cr
sand treatment is that of the Wasp mill, Black
ing $2.20 gold (1911) is profitably leached, cr
continuous leaching with 0.25% cyanide solutio
with 0.12% cyanide solution and 1 wash with cl
tons per 100 tons ore; total extraction claimed

11. Slime Tr

Agitation of the thickened slimes may b
by centrifugal pumps, or by compressed a

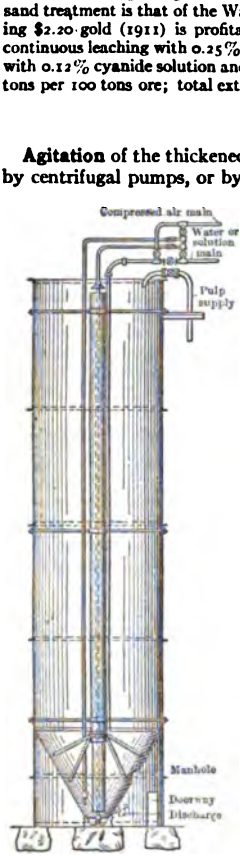

Fig 6. Pachuca Tank

Simplest m
2 or 4 blades
this is the H
conical botto
propellers lea
over an apron
AGITATOR co
with use of
from top of
forated pipes
AGITATOR co
ing vertical s
ing plows sin
plows draw
where a jet o
der. From
tributed over
revolving lau
carries a num
volve rapidly
It is said to
down, the se
having prope

Agitation
the BROWN
high steel t
tube, throug
liberated at
to the top,

There are
is used in a
form being a
sides of the
central tube
Air is used in
a velocity of

Some claim that a high tank will agitate double
height can treat, with the same number of cu f
compress the same amount of free air to twic
It is said that in high tanks there is less loss of
of the excessively high tank is much against it.
tall building also increases cost of plant. At

ıtion of this tank, in which the central tube extends only to ⅔ the height of ; it is stated that there is increased agitation in the upper third.

gh Pachuca tanks are more easily started up when the air is turned on after a ın than are mechanical agitators, trouble often occurs when a tank must be closed ithout discharging completely, as the slimes may pack tightly in bottom of the This is obviated by the TOWNE AGITATOR, in which the central tube is perforated of 4 or more holes spaced at intervals of 12 or 18 in for the complete height of the After a shut-down, agitation begins through the topmost holes in the tube, in the rt of tank where the charge is loose, and the moving pulp gradually stirs up the ands until all the charge is again in circulation. A further advantage of this de- at agitation is started during filling, taking place through the holes near the sur- . similarly, agitation is maintained while a tank is discharging, circulation taking r the falling level of the pulp. BUEL'S MODIFICATION of the central pipe consists able air lift, which can be raised when sands pack near the bottom. PARRAL TANK)onald) has a flat bottom, and uses 4 air lifts, discharging from the side near the e pipes, which imparts a whirling motion near surface of charge. PATTERSON consists of an injector which introduces clear solution in the bottom of the cen- . BANKS MODIFICATION, operated at Waihi mill, New Zealand, uses an air pocket n a Pachuca tank without a central pipe, by which compressed air is released at m with a pulsating action.

Dry slime in pulp, %	Parts of sol to dry slime	Sp gr of pulp	Wt of 1 cu ft of pulp, lb	Cu ft pulp for 1 ton dry slime
16.67	5 : 1	1.111	69.5	172.9
20.00	4 : 1	1.136	71.1	140.8
25.00	3 : 1	1.176	73.4	108.8
28.57	2.5 : 1	1.206	75.3	93.4
33.00	2 : 1	1.246	77.8	77.7
50.00	1 : 1	1.429	89.2	44.8

ion of pulp. To the d pulp charged in the ; tanks enough stock solution is added to ıe pulp to the required ıcy, at the same time ing the proper strength de in the charge. Dilu- the pulp depends upon r of the slime; thick ime requires greater dilution for prompt extraction. Practice varies from lution to 1 part dry slime by wt, to 4 or 5 parts solution to 1 part dry Accompanying table is useful in dealing with the sp gr of pulps; it is a sp gr of 2.5 for dry slime.

rliate values in the table may be interpolated. In practice, it is usual to take f the thoroughly agitated pulp in a weighed flask of known capacity, and from t of the pulp calculate the sp gr, the other data being obtained from the table.

gth of solution. The amount of cyanide in the solution which forms the he agitation tanks is determined by experiments on the ore, and varies n different places. The strength may be expressed in terms of lb or per KCy, it being usual to figure back to the basis of KCy even if NaCy be n general, for Au ores, from 0.05 to 0.25% KCy covers the range of of solution. For Ag ores, the range is from 0.25 to 0.50%.

battery crushing or tube-mill grinding has been done in solution, the strength of ed is generally much lower than that named above, and, to bring the pulp up to percentage of cyanide, a stronger solution is added in the agitation tanks to the pulp; or, if the pulp reaches the agitation tanks already sufficiently dilute, anide are dissolved in the charged tank when agitation commences. At the same proper amount of Pb $(C_2H_3O_2)_2$ is added, which is generally done by suspend- burlap bag in the charge, containing Pb $(C_2H_3O_2)_2$ corresponding in amount to ed for the treatment of the number of tons of dry slimes in the tank.

arge should also be tested for protective alkalinity, though the proper amount insure this is generally crushed with the ore in the battery, or milk of lime is the charge in the tube-mill.

rouble has been experienced in air agitation, due to excessive use of lubricating compressor cylinders. Some plants use soapsuds or graphite lubricant for ers, and claim that extraction is thereby increased.

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Quantity of air required. A. Groth
states that violence of agitation does not
constant agitation is needed which will mo
the inlet of the air lift, so as to prevent accu

Grothe says, "To obtain this perfect circulatio
for a tank 15 ft diam by 45 ft high, holding a cha
mesh, and 75 to 80% to 200 mesh, with a propor
another matter when some peculiarity of the or
overcome, in which case it may be judicious to c
below the top of the air-lift, making use of the
might otherwise accumulate and cause trouble."

Time of agitation varies greatly with
testing the solution shows a satisfactory ex

In a number of plants, the charge is agitated fo
hr, the clear liquid decanted and fresh solution
repeated for from 4 to 8 times. With other o
necessary, followed by 12 hr or more for settling,
tant clear solution is decanted and the settled p
plants use a Dorr thickener (Art 9) to take th
tanks without allowing any time in the latter fo

Temperature of solutions. There is no
tions, because with the increased solubility
solubility of the base metals is also increas
advisable, and in cold climates provision mu
of solutions, or of very decided drop in tem

Decantation. Simple decantation is rapidly
rating pregnant solution from treated pulp; it w
until the advent of modern filters. It was usual
certain settling period, were repeatedly refilled wi
commonly used to aid the settling of flocculent sli
before the last settling and discharge of the slime
here refers to a process distinct from the partial s
agitation treatment, which presupposes a final t

13. Continuous C

Object is to avoid the loss of time cons
agitation tanks. There are 2 methods: co
rent decantation.

Continuous agitation. A series of tanks
ally by inclined 6 or 8-in pipes. The pulp fe
and is delivered somewhat below the top of
(even as far down as half the height of tan
through the series, until the completely tre
agitation tank to the storage tank or settle

The above system might be modified by usi
box with a series of compartments having an u
ments being of equal length. In flowing through
air passing through the bottom, made of JUST SIE
ing bottom, with air introduced beneath. Wit
lessened for any particles of pulp to get less tha

Countercurrent decantation uses a series of
tling tanks connected with each. Solution is in
with the mill pulp. Water is added to the last
it the dissolved values and cyanide solution. Thi
thickeners of the series, and the solution passes

umber 2, from which it is delivered to the mill solution. The solution from overflow is sent to the precipitation plant, from which the barren solution is urned to the next to last thickener of the series. The discharged pulp from the ner (where it has been treated with wash water) may with certain ores be so dissolved values as to be rejected as final tailing, but in some cases a final filter- e necessary to recover all the dissolved values.

13. Filtration

ion by gravity (Art 10) is employed for percolation of sands.

pressures or other appliances are used to separate solution from pulp in nent of slimes. They are classified as VACUUM FILTERS and PRESSURE In the former, the solution is drawn from the pulp by vacuum; in the solution is forced through the pulp, by pump pressure or by In general, vacuum filters are better for ores containing much flocculent colloids; pressure filters, for treating granular or crystalline slimes.

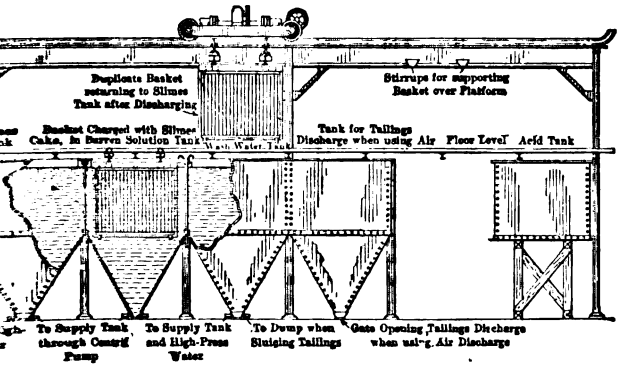
Duplicate Basket returning to Slimes Tank after Discharging
Stirrups for supporting Basket over Platform
Basket Charged with Slimes
Cake, in Barren Solution Tank
Wash Water Tank
Tank for Tailings Discharge when using Air
Floor Level
Acid Tank
To Supply Tank through Centrif Pump
To Supply Tank and High-Press Water
To Dump when Sluicing Tailings
Gate Opening Tailings Discharge when using Air Discharge

Fig 7. Moore Filter

n filters. One of the commonest is the MOORE FILTER (Fig 7), con- a pipe frame enclosing wood strips which support heavy canvas. The drawn through the filter surface into the perforated pipe frame by a ump during 2 to 5 hr. This leaves a cake, 1 or 2 in thick, on the outer the canvas, after which the filter sections, or leaves, with the cakes, ed for washing with solution and with water. The washed cake is charged from the leaves by air press, applied from inside of the pipes. ting cost is low and capacity large, but the washing is often imperfect.

filter resembles the Moore, using cocoa matting inside the canvas, but the d discharge of cake are effected in the tank where the cake is formed. Cost ion can be made somewhat lower than that of the Moore, omitting royalty Washing of the cake has generally shown no advantage over that attained in OLIVER FILTER, also of the vacuum type, is continuous in action (Fig 8). It a revolving drum partly submerged in a tank about half full of thickened slimes. t of the revolution of the drum, the surface of which carries the filter, the solu- n off by a vacuum pump, connected through the trunnion of the drum, through c. The cake thus formed is sprayed through a succeeding arc with wash water, awn off to weak solution storage by another valve in the trunnion. During ter of the revolution air pressure is applied to inside of the filter through a third e trunnion, discharging the cake, which is finally removed by a scraper held wire windings outside of the canvas surface. ADVANTAGES are: low cos

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continuity of operation, economy of space o
repair cost. One of its DRAWBACKS is the lir
to washing. The cake formed is not over c

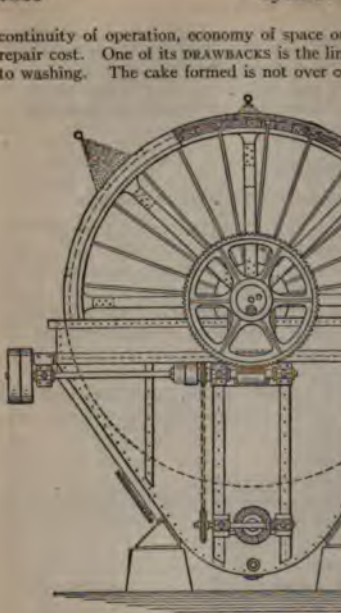

Fig 8. Oliver Continuous Fi

used for small plants. Pressure is ge
plates being covered with duck or filter-c
in 2 to 3 hr. It is washed by pumping w
and discharged by hand. Cost of oper
labor and small capacity.

Merrill filter, probably the most widely u
the ordinary plate filter-press; it is used gen
by interior sluicing. There are 2 types: one
on granular slimes), the other making a cake
mer the cake is formed solid and the wash so
latter the wash is forced through the halves
between them. Filling is done at 25 to 30 l
can be obtained by vacuum filters. Capacit
installation is high. Cycle of treatment, incl
4 hr for the center-washing type to 8 or 9 h

Burt stationary filter has pipe-frame lea
suspended in a diagonal cylinder (45°), into
a cake of 1 to 1.5 in is formed. Wash water i
solution and dissolved values, after which th
pressure of air in the pipe-frames forces the
slides out of the cyl. A small stream of wate
ating costs are low, but constant attention is
KELLY FILTER operates like the Burt, but th
all the leaves is run out of it on a wheeled f
operation is probably slightly higher than fo

Burt revolving filter is also a pressure fil
the cake forming on the interior surface a

n is not continuous like the Oliver. Wash water is forced through, after which the cake is sluiced out. A thick cake (5 to 6 in) is formed, and the operating cost is almost as low as for the Burt stationary filter, since power consumption is low. are many modifications of these filters, as the Blaisdell and the Sweetland, ificial pressure, and the Portland, a variation of the Oliver continuous vacuum Cost of operation varies considerably, as low as 4¢ per ton being claimed for the d Burt. An aver of 14¢ may be assumed for the vacuum non-continuous types. y run to 90¢ or more for small installations of the ordinary plate-and-frame type. he high operating cost of filters requiring a long cycle, is the more effective wash- ke and more complete elimination of dissolved values.

formation of the cake, the solution leaving the filter represents only about ¾ he dissolved values, the remainder being in the cake and in the moisture retained ter leaves. Good practice requires that at least 95% of the dissolved values re- should be removed from the cake and filter by washes of weak solution and of If the cakes are not uniform, or if they have channeled during formation or during the wash will not effectively reach and displace all of the retained values and very results. Pressure gages are attached to the pipe system, to indicate breaks tering medium.

ne of wash solution should be at least 1 to 1.5 times the volume of held in the cake and filter leaves. This is followed by wash water nt to the volume of water held in the cakes.

mps forcing the pulp into the pressure filters, or for handling wash solutions, may centrifugal or plunger pumps. The latter are provided with circulating water ffing boxes to avoid cutting away by action of the gritty pulp. Pumps and pipe e made entirely of iron; no brass valves, linings, glands, or plungers are admissible, nide readily attacks them.

ying of solution. Filtered pregnant solutions are often cloudy, con- ne slimes or other matter in suspension. Best practice is to clarify utions before running them to precipitation tanks. The solutions are ravity to a redwood or steel tank, with a false bottom of slats covered as and a thin bed of sharp, clean, fine sand, care being taken to main- and constant flow through the clarifying tanks; hence the layer of sand watched to detect channels that may be formed in it.

t complete settling, rapid currents must be avoided in all parts of the bed, and should be of relatively large diam and shallow depth. In some plants solutions

rd by running
ter presses with
area and loose-
ilter cloths.

ipitation and fining

itation on vings is the thod. The olution passes of boxes con- he shavings. ion overflows box, rises he bottom of and so on to Each box bottom mber to

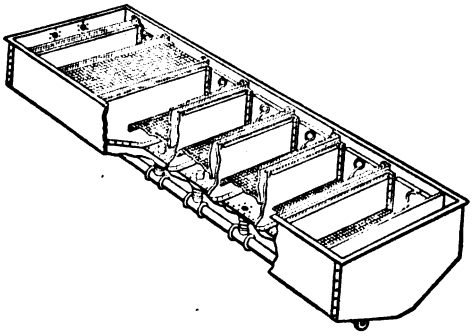

Fig 9. Steel Zinc Precipitation Box

ontained in a longitudinal tank. The boxes, 6 in or more in width and height, with a total re of wood or sheet steel (Fig 9).

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The Zn shavings rest on a screen bottom, a
¼ in wide and extremely thin, generally not
packed uniformly in each compartment to insu
Amount of Zn is calculated at about 1 cu ft o
treated per day. The Zn is sometimes dippe
Pb couple, making the precipitation more effe
the Zn, the strength of the cyanide solution run
more; the Zn is then washed or shaken in a s
is washed down into the bottom of the troug
clean Zn is moved up from one compartmen
fresh Zn being added to the lower compart
"short" zinc as may have fallen with it, is t

Precipitation by zinc dust. In usi
feeder into the pregnant solution, so as
Sometimes a little cyanide solution is mix
stream of pregnant solution drawn from
provement in use of Zn dust by slightly
remove any coating of ZnO from the met
to react with the cyanide solution.

The pregnant solution mixed with the Zn d
press. Precipitation takes place almost imm
the filter leaves, while the precipitated Au and
leaves. When enough cake has been formed
gage connected with intake side of filter), pum
water is forced through; then air is forced int
the precipitate, which should leave the press
then opened and the caked precipitate remov

Size of press is figured by allowing for eac
2 tons solution to be filtered per 24 hr, when A
being treated; for Ag solutions, about 40% 1
of space for forming cake in the filter should b
24 hr, depending upon amount of metal whi
ATURE OF SOLUTION to be precipitated should

Zn dust *vs* shavings. With Zn du
conducted there is less loss of Zn and less

Thin Zn sheets, cut into strips about 1.5 by
and will be completely consumed in precipitati
ing loss of Zn "shorts" in the precipitate. T
more surface for contact with solution than
become coated even in presence of considerab
precipitated by 1 lb Zn dust, in case of low-gr
cess alkali. With strong solutions, as much as

Cost of precipitation by Zn dust in
of solution; other mills state their precip
treated. Cost with zinc shavings, is con

The interfering agencies which cause troubl
dotes, have been referred to in Art 7.

Precipitation by aluminum is similar
as effective and cheap, and does not fou

Siemens-Halske electrolytic precipitatio
with iron anodes and lead cathodes. Its gre
Au solutions. Butters modified this by using
precipitate is likely to contain many base met
to melt and flux. The apparatus is costly a

Precipitation by charcoal has been used w
Precipitating boxes are charged with soft-woo

is allowed to flow very slowly through them. After a short time the precipitating f the charcoal is greatly diminished. Principal advantage is the low cost of instal- d of attendance.

ing and refining the dried precipitate is done in graphite crucibles, without some preliminary preparation. Certain plants dissolve out the Zn with a 10% mixture of H_2SO_4, and dry the cleaned precipitate for melting. HCl is also used.

imes a preliminary roast is given the precipitate before melting, in which case a r may be mixed with the charge before roasting. Melting in crucibles is done in l, or gasolene-fired furnaces. Flux is generally mixed thoroughly with the dried te, except in mills treating very large quantities of bullion, where large tilting are used and the precipitate cakes and flux are charged in the melting pots in Precipitate may also be melted with litharge and flux, and the lead bullion cupelled.

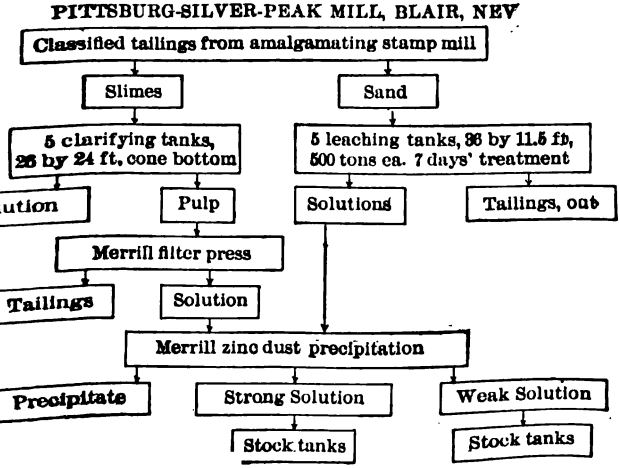
position of fluxes varies considerably. An aver flux is 30% sodium 10% clean SiO_2, and 60% borax glass. Wt of flux is 18 to 35% of wt of te; more, in case the precipitate is high in Zn. A little $NaNO_3$ may be o 10% of wt of flux), but an excess results in scouring the crucible rapidly. itate carries much base metal, the proportion of SiO_2 may be larger.

elted bullion is poured into molds, generally with the slag, which is allowed to fill old and is later broken away from the bullion. Sometimes the slag is skimmed pouring, especially if there is much S, and additional niter and flux have to be The melt is always thoroughly stirred with a red-hot iron rod, to settle out shots before pouring. Molds are heated and prepared as described in Art 4. Slag and any) are saved, broken up, and retreated when enough accumulates. This re- s done with a large excess of very liquid slag and sufficient niter to oxidize the d is performed at a very high heat.

15. Flowsheets of Typical Mills

following flowsheets of typical mills, operating on widely varying ores, the opera- indicated only from the point where the cyanide treatment proper begins, generally second stage of crushing. When the preliminary crushing is done in solution, ted. Several of the mills have somewhat modified their treatment at this date nce the data here given were collected (1913).

PITTSBURG-SILVER-PEAK MILL, BLAIR, NEV



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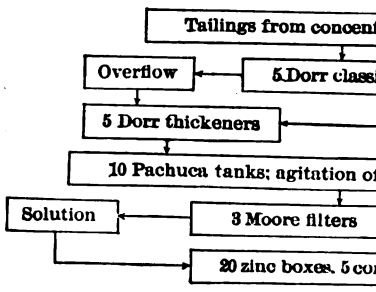
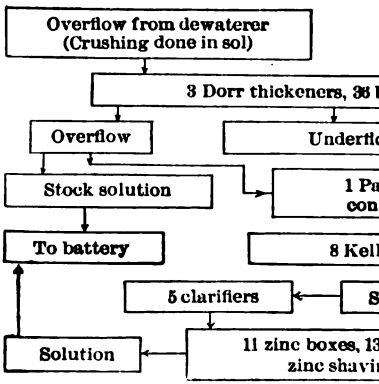
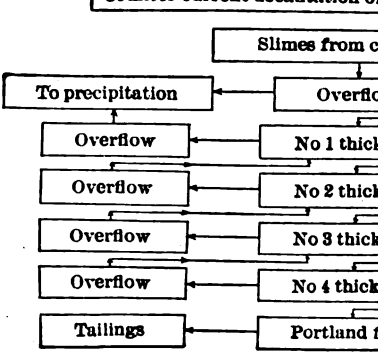
SAN RAFAEL MILL, PA



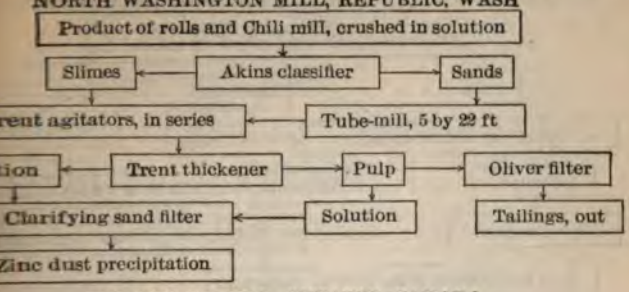
VETA-COLORADO MILL, PA



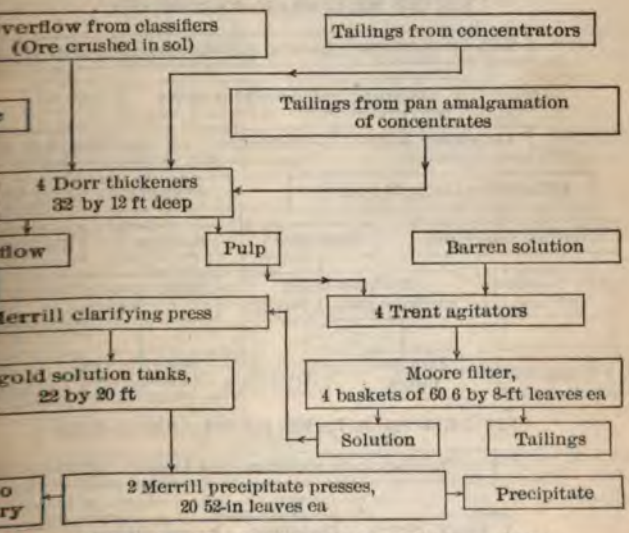
RELIANCE MILL, TROJA



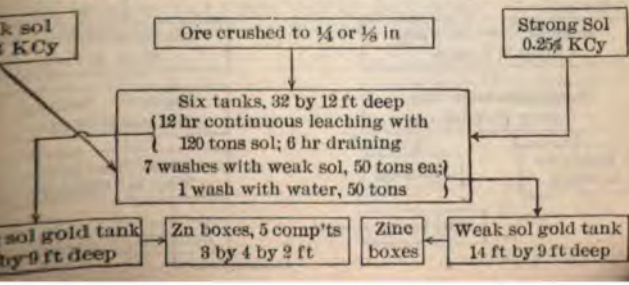
NORTH WASHINGTON MILL, REPUBLIC, WASH



HOLLINGER MILL, ONTARIO, CANADA



WASP NO 2 MILL, BLACK HILLS, SOUTH DAKOTA


k sol
KCy
Ore crushed to ¼ or ⅛ in
Strong Sol
0.25% KCy
Six tanks, 32 by 12 ft deep
12 hr continuous leaching with
120 tons sol; 6 hr draining
7 washes with weak sol, 50 tons ea;
1 wash with water, 50 tons
sol gold tank
by 9 ft deep
Zn boxes, 5 comp'ts
3 by 4 by 2 ft
Zinc
boxes
Weak sol gold tank
14 ft by 9 ft deep

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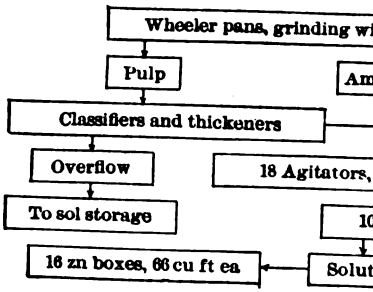
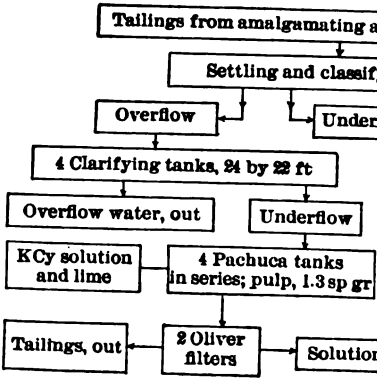
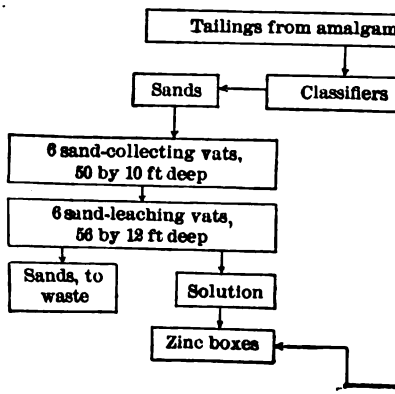
GREAT BOULDER PROP MINES, K



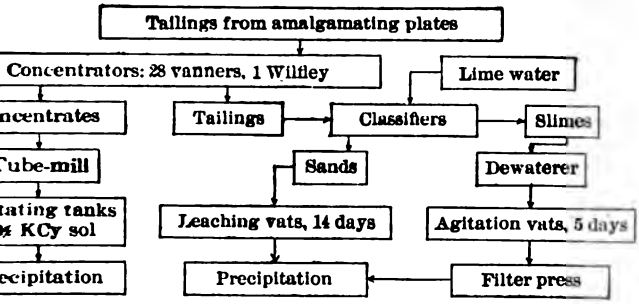
EMPIRE MILL, GRASS



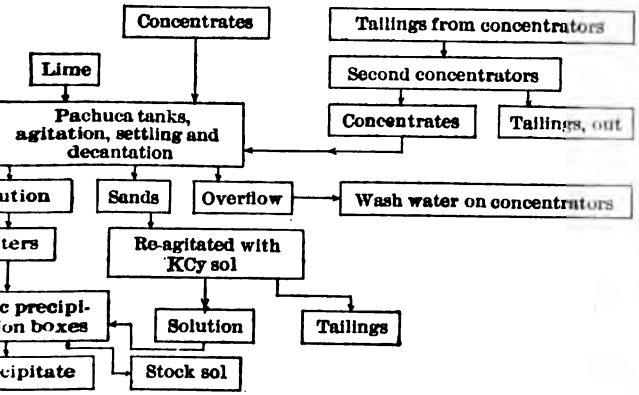
CITY DEEP MILL, RAND, SO



WAIHI MILL, NEW ZEALAND



FREMONT MILL, AMADOR CO, CAL



16. Examples of Costs

of cyaniding materially advanced almost everywhere at the end of 1914, after the ak of the European War; in Mexico, also, because of the disturbed conditions there. increased notably for Zn dust and shavings, pebbles and silex bricks for tube-mills cyanide. The costs here given are for 1913 and previous years.

United States

estake Mining Co, Lead, S Dak. Gold ore, partly oxidized, with some pyrite. amation followed by sand and slime treatment (2 sand plants, 1 slime plant).

	1910	1913
bor	$0.0744 per ton	$0.0740 per ton
wer	0.0203 " "	0.0191 " "
emicals	0.0927 " "	0.0807 " "
pplies and miscellaneous	0.0202 " "	0.0310 " "
Total	$0.2076 " "	$0.2048 " "
consumption of chemicals, all plants:		
dium cyanide	0.289 lb per ton	0.223 lb per ton
me	4.860 " " "	3.750 " " "
nc dust	0.086 " " "	0.089 " " "
Cl (slime plant)	0.500 " " "	....
wer used		2.1 kw-hr

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North Star mines, Grass Valley, Cal, Oct, 1913.
Concentrating (power, labor, and material).
Cyaniding: Labor.....
Supplies.....
Power.....
Total.....

Total, concentrating, and cyanid
Consumption of chemicals per ton treated: cyan
Aver extraction (1912), 97.5%.

Deadwood, S Dak. Gold-silver ore; plant fo

Cost, based on treating 14 100 tons per month

Labor.....	$0.213 per ton	C
Zinc.....	0.033 " "	L
Power.....	0.085 " "	S
Miscellaneous.....	0.136 " "	T

New Portland and Independence mills, Colo.
$1.23 to $1.25. Independence mill only (1911–12)

Blair, Nev (1908). Gold ore with sulphides.
ceded by amalgamation. Extraction, 92%. Ave

Bullfrog, Nev (1907). Gold ore. Amalgamati
leaching and slime agitation. Extraction, 90%.

Goldfield, Nev (1910). Complex sulphide o
followed by all-sliming in Pachuca tanks, Butters
tion. Extraction, 94%; cost, $1.61 per ton.

Goldfield Consol Mining Co, Nev, reports cy

Nipissing Mining Co, Cobalt, Can (1913). C
Crushing, elevating, stamping and tube-milli
Desulphurizing by aluminum and alkali filter
Cyanide treatment and cyanide filter.....
Precipitation and refining.....
Water service.....
Total.....

Hollinger Gold Mines, Ltd, Timmins, Can (J
by stamps, tube-mills, slime concentrators, Dorr t
and precipitation by Zn dust in Merrill presses.
treating concentrates, and 6% in continuous decan
General expense, superintendence, etc .....
Mill power, crushing, and conveying.....
Classification, tube-milling, and concentration
Pumping pulp, thickening, decanting, and filt
Clarifying, precipitating, and refining .....
Cyanide.....
Assaying, clean-up, etc.....
Total.....

Porcupine Crown mine, Can (Jan, 1914). G
current decantation. Extraction, 96%. Cost, $1

Pachuca district, Mex (1913). Aver of a numb
concentration, tube-milling, agitation in Pachuca t
oz Au and 10 to 12 oz Ag per ton. Aver recovery
(data from Hugh Rose).
Crushing and concentrating.....
Classifying and tube-milling.....
Thickening and agitating.....
Filtering .....
Precipitating, melting, and refining.....
Miscellaneous.....
Total.....
Aver for district, $1.50 per ton.

s weigh 1 200 to 1 550 lb, 6.5 to 7.5-in drop, 100 to 105 drops per min. Concentra- vers only about 10% of total. Tube-mills aver 5 by 20 ft, 30 to 32 rev per min. k used instead of pebbles. Pachuca tanks, mostly 15 by 45 ft. Air consumption, per min at 30 lb. Lime costs $5 per ton; 10 to 25 lb used per ton ore. Con- n of NaCy, 1.5 to 3 lb per ton. Pebble consumption, 10 lb; mine rock, 100 lb per dust consumed, 1 lb per ton. Shavings, 1.3 lb per ton. Litharge used instead cetate; consumption, 0.75 lb per ton. Power, $50 per hp per year; cost per ton, t.

y of cost:			
Labor	$0.20 to	$0.35 per ton
Power	0.20	0.35 " "
Supplies	0.77	1.45 " "
Total	$1.17	$2.15 " "

Rafael mill, Pachuca, Mex, reported for 1912, extraction of 91% Ag and 97% Au. 10 per ton.

Francisco mill, Pachuca, 1912. Extraction, 92% Ag and 95% Au. Cost, $1.93

o Mines Ltd, El Oro, Mex, reported extraction of 80% Ag and 95% Au. Cost, ton.

juato Consol mine, Mex (1912). Silver ore treated by sand leaching and slime- ; extraction, 87%; cost, $2.25 per metric ton. Consumption of chemicals per Cy, 3.8 lb; CaO, 102 lb; lead acetate, 0.4 lb; Zn, 1.8 lb.

strellas mill No 1, Michoacan, Mex. Sand and slime treatment. Extraction es, 71% Ag and 92% Au; from sands, 68% Ag, 90% Au. Cost, $1.55 per ton.

tersrand, So Africa. Early practice. Free gold ores, sand treatment (1894). raction, 78%. Milling and cyaniding cost, $2.20 per ton. Simmer Deep (1909). ore, sand percolation and slime decantation. Extraction, 90%; cost, $1.09. d (1910). Sand and slime treatment. Extraction, 92%; cost, $1.33. Gelden- 1). Grinding and cyaniding concentrate, agitation, and continuous decantation. n, 98%; cost, $6.32.

rlie, Australia (1909–10). Free gold and telluride ores. Roasting, amalgama- cyaniding by agitation. Extraction, 95%; cost, $2.40 per ton.

ealand. Treatment of complex sulphide concentrates by fine tube-mill grind- gitation, and filter-pressing. Extraction, 96% Au, 93% Ag; cost, $6.25 per ton.

17. Cyanide Poisoning

s and symptoms. Cyanide poisoning may result from taking cyanides y or from inhaling HCy gas. CHRONIC form of poisoning, due to long to the fumes, is manifested by disturbance of vision, pain in the fore- l in region of the heart, palpitation, difficult respiration, vertigo, cough- attacks of suffocation. Symptoms of ACUTE poisoning, as when taken y, are, besides the above, constriction of throat or general contraction of and dilated pupils. Face becomes pale and bloated, nails blue, some- thing at mouth. Convulsions occur, followed by coma, and death om asphyxia.

ment. If poisoning is caused by breathing fumes, and the case is not elief may be afforded by breathing ammonia fumes and remaining in . If gas has overcome the patient, a number of hypodermic injections % solution of H_2O_2 may be made, or a 10% solution of it given internally.

al respiration may be resorted to; or, if available, O alone may be administered. nics of ether, strychnine or whisky are given to prevent collapse. If the cyanide **swallowed,** an efficacious antidote is $Fe(OH)_3$, which must be freshly prepared.

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gm of powdered MgO. These 3 ingredients are k
To use them, 30 c c of each of the 2 solutions are
diluted with a glassful of water and given imme
moments, the stomach should be pumped out o
antidote, the stomach may be washed with a 0.3
of $Co(NO_3)_2$.

Prevention. Cyanide poisoning can be l
supply of drinking water in the works and g

Workmen in the melting and refining room m
poisoning may be caused by HCy, and H, alone
with acid, will not support life. When As is pres
duced, which is quite as poisonous as HCy. W
been left closed or covered with cyanide solution
off enough gas to cause poisoning. A slight poiso
hands have been cut or scratched; avoided by
handling precipitate.

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SECTIO

PREPARATION AN
ANTHRACIT

BY

PAUL STE

MECHANICAL ENGINEER, LEH

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PREPARA

1. Market Sizes o

Classification. Run-of-mine coal is cl
market, into: lump, steamboat, broken, egg,
wheat (or buck), rice, barley, and buckwhea
is due to the fact that anthracite burns bes
permitting easy passage of air through the

Table 1. Market Sizes of Anthracite Coal

(Screen openings, in inches, over and through which each size is made.)

Size of coal	Punched plate				Woven wire	
	Round		Square			
	Over	Through	Over	Through	Over	Through
.....	6½	.....	.....	.....	.....	.....
boat.....	4½	6½	4	.....	4	.....
n.....	3¼	4½	2¾	4	2¾	4
.....	2⁵⁄₁₆	3¼	2	2¾	2	2¾
.....	1¹ 1⁄₁₆	2⁵⁄₁₆	1½	2	1½	2
nut.....	1⁵⁄₁₆	1¹ 1⁄₁₆	1 3⁄₁₆	1½	¾	1½
.....	⅝	1⁵⁄₁₆	⅜	1 3⁄₁₆	½	¾
wheat.....	7⁄₁₆	⅝	⅜	⅜	5⁄₁₆	½
.....	¼	7⁄₁₆	.....	⅜	5⁄₁₆	5⁄₁₆
.....	3⁄₃₂	¼	.....	.....	.....	5⁄₁₆
wheat No 4.....	3⁄₆₄	3⁄₃₂	.....	.....	.....	.....

f market sizes is about as follows: lump, for locomotives; steamboat, for nace and forge; broken, for domestic furnace and forge; egg, stove, and nut, for furnace and range; pea, for domestic furnace and range, and steam boiler; at, rice, and barley, for steam boiler. Buckwheat No 4 is a recent product, er boilers at some collieries, to replace rice and barley.

pared sizes" is the trade name for: broken, egg, stove, and chestnut, which e about 65% of the total production. General demand for lump and steam- mall and decreasing. Probably these sizes will not be produced in a few years; be "broken down," as is done at present, when there is no demand for them.

m sizes" is the trade name for: pea, buck, rice, and barley. These are the ion resulting from the mining and handling of a brittle or laminated material, ot especially desirable.

f-mine is the unsized mixture of coal and impurities, as dumped into the According to method of loading cars in the mines, it may be divided into:

1. Hand loaded (as in flat workings), containing lumps up to about 150 lb ively free from impurities in coarse lumps.

2. Chute or steam-shovel loaded (as in steeply pitching seams or in strippings), g lumps up to about 2 000 lb, and frequently wet and dirty.

ities in run-of-mine vary with character of seam, and with method of loading. hand loading they may be partly removed, but in chute loading the entire cluding all impurities and foreign matter, as clay, top and bottom rock, and imbers, must be loaded into the car. Impurities in Class 1 usually range from to 25%; in Class 2, about 10 to 55%.

dard of preparation designates the percentage of over or under size mpurities allowed in each size of coal (Table 2). BONE (mixed coal and defined as containing between 40 and 65% carbon.

car yield is a local term, usually indicating number of long tons of marketable pped from the mines for every 100 cu ft of mine car capacity dumped. This ment, for hand-loaded cars, is somewhat misleading. For chute loading, the city is figured at its water level; to this, for hand loading, is added a volume the horizontal area times 6 in. The added height of 6 in, above top of car, is PPING. Should the topping exceed the required 6 in, no account of it is made ating mine car yield, nor is it practicable to do so. The exact method would be mine car yield on the weight of loaded mine car; but this would complicate nt rates paid miners, lead to disputes and labor troubles over amount of allow- urities in each car, and violate the award of the Anthracite Strike Commission of the subsequent agreements between operators and their employes. The present admits a fair comparison between mines operating under similar conditions. A

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"yield" below a previous established record
gated, and, if possible, corrected.

Table 2. Standard
(Percentage of impurities and of over

Size of coal	%
	Slate
Broken	1
Egg	2
Stove	4
Chestnut	5
Pea	10
Buckwheat	15
Rice	15
Barley	15

* If slate is entirely removed, the percenta

Shipments of prepared coal from a mine, p
steam sizes), may vary from 0.75 ton, with v
tons in clean seams, hand loaded.

Loss in prepared sizes underground
the coal seams; kind of explosives used;
steam shovel); shocks in running over poo
during gravity runs or mechanical haul
the breaker is due to: poorly constructed
ning chutes with abrupt turns; poor typ
breakage in or on mechanical cleaners,
elevators; drawing coal under pressure
losses can be entirely eliminated, but the
ment and supervision over mining and
taking design and construction in the br

Breaker products. Table 3 gives av
mine, for Classes 1 and 2, percentage of

Table 3. Sizes of Coal, Per Cent of
from 1 000 Tons R

Size of coal	Price per ton	Class		
		% of		Co
		Coal	Rock	to
Lump Steamboat	$2.83	39.0	4.0	3
Broken	2.71	7.0	1.5	
Egg	3.20	6.0	1.0	
Stove	3.20	5.0	1.0	
Chestnut	3.34	10.0	1.5	1
Pea	2.01	5.5		
Buckwheat	1.45	5.0	5.5	
Rice	0.71	4.0		
Barley	0.42	4.0		
Dirt	....	....	....	..
Totals	....	85.5	14.5	8

the mine cars, with prices at mines, as of Feb, 1913, assuming 1 000 tons erial delivered to breaker. (War prices are not considered.)

le 4 gives percentage of shipments of each size made by preparation of oo tons of run-of-mine in Table 3, and the average and total amounts d, assuming each size to contain the allowable percentage of impurities, able 2. Each size in Table 4 contains no under or over size, and if the um amounts allowed (Table 2) were put into the highest-priced sizes, erage price per ton would be increased. In practice it is impossible to coal without under or over size. Hence, tests are made, and the screen ncreased or diminished, so that the greatest allowable percentages of over er size are carried in the highest-priced sizes; or the screen mesh is ad- to produce maximum amount of sizes required to supply the market. y all the barley coal, and sometimes part of the rice, is required for the boiler fuel, so that the average price per ton does not generally include izes. Fuel burned at the colliery approximates 5 to 10% of total pro-

Table 4. Percentages of Sizes and Value of Breaker Products from 1 000 Tons Run-of-mine Coal

of coal	Price per ton, $	Class 1				Class 2			
		Per cent of shipment	Tons of		Value, $	Per cent of shipment	Tons of		Value, $
			Pure coal, Table 3	Coal and impurities shipped			Pure coal, Table 3	Coal and impurities shipped	
iboat	2.83	10	85.5	85.5	241.97	5	30.05	30.1	85.18
n	2.71	12	102.6	105.7	286.45	7	42.07	43.3	117.34
	3.20	15	128.25	133.4	326.88	14	84.14	87.5	280.00
	3.20	13	111.15	118.9	380.48	14	84.14	90.0	288.00
nut	3.34	28	239.40	263.3	879.42	21	126.21	138.8	463.59
	2.01	6	51.30	61.6	129.36	10	60.10	72.1	144.92
wheat	1.45	6	51.30	59.0	85.55	18	108.18	124.4	180.38
				827.4	2330.11			586.2	1559.41
ge price per ton above rice					2.81	....	....	.....	2.66
	0.71	5	42.75	49.2	34.93	6	36.06	41.5	29.47
	0.42	3	25.65	29.5	12.39	3	18.03	20.7	8.69
	....	2	17.1	.....	.....	2	12.02	.....	.....
al	....	100	855	906.1	2377.43	100	601	648.4	1597.57
ge price, all sizes					2.62	....	....	.....	2.46

. In Table 4, Class 2 shows a loss of $779.86, compared with Class 1, efly to difference in amount of impurities in the run-of-mine (Table 3). ould also be a further financial loss due to increased preparation cost in g and removing the higher percentage of impurities.

king-down large sizes. If the lump, steamboat, broken, and egg m 1 000 tons of Class 1, run-of-mine, Table 4, were broken down to stove, l the steam sizes, the approximate shipments in tons and amount realized e as shown in Table 5. The saving, $75.28, is attributable to the effi- f the slow-speed rolls used. With high-speed rolls, of poor design or ll teeth, the efficiency would usually be low, and the breaking-down ause a loss due to high percentage of pea and smaller sizes made, which pproximates 20 to 30%, as compared with 13.8% obtained in Table 5.

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Table 5. Results of Breaking Down L

Size of coal	Price per ton, $	Total tons shipped, Table 4, Class 1
Lump Steamboat }	2.83	85.5
Broken.....	2.71	105.7
Egg.....	3.20	133.4
Stove.....	3.20	118.9
Chestnut.....	3.34	263.3
Pea.....	2.01	61.6
Buckwheat.....	1.45	59.0
Rice.....	0.71	49.2
Barley.....	0.42	29.5
Dirt.....	.....	.....
	.....	906.1
Average price per ton above rice.....		
Average price, all sizes.....		

Breaking-down losses may be partly offset
purities which the broken-down coal is allo
tons of stove may be increased by approxin
by 4.8 tons, 13.6 tons of pea by 1.5 tons,

Economic principle in preparation of coal
sizes of highest priced coal, each size containing
impurities. Varying percentages of the differ
impossibility of adjusting crusher and re-breakin
tion of any one of the prepared or steam sizes, p
ments. Hence, the problem of economic prepa
each particular breaker, by frequent tests of r
machinery must be adjusted to give results as n
with market demands.

2. General Comparison of

Classification: *A*, dry preparation; *B*,
preparation. The method to be adopted
mined. Fig 1, 2, and 3 show outline flow-s

Class A is employed when the coal seams
impurities, or when the benches of slate in th
be removed during hand loading, the run-o
7 or 8% of rock or slate, which may be r
mechanical separators. Class *A* offers the
low cost of construction, operation, and m
are preferred by the trade, being free from ri
difficulties in unloading.

Class B is employed when run-of-mine c
ties, including rock, slate, and bone. Thou
as 55%, the run-of-mine must contain larg
as a separate product, as in Class *A*. The
wet methods to improve its appearance ar
Class *B* possesses to some extent the adva
higher in first cost, operation, and maintena

s C is adopted when run-of-mine is high in impurities and is discolored, e case near an outcrop; or when entire product comes from wet, dirty

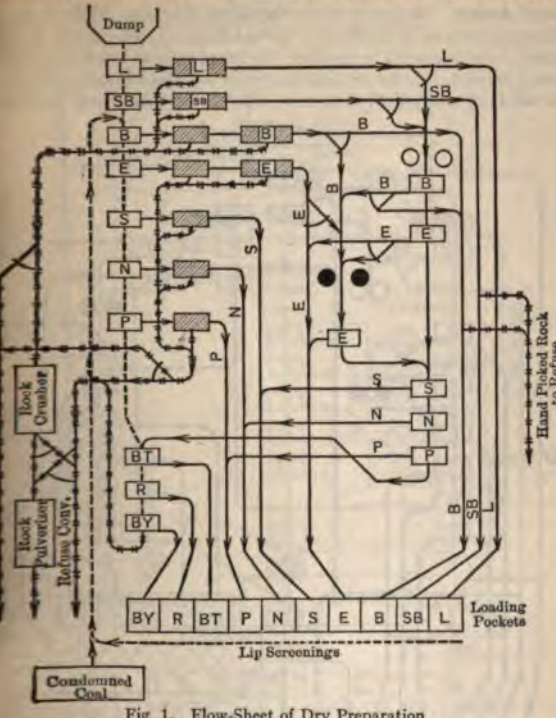

Fig 1. Flow-Sheet of Dry Preparation

Lump	L	Chestnut	N
Steamboat	SB	Pea	P
Broken	B	Buckwheat	BT
Egg	E	Rice	R
Stove	S	Barley	BY

Pure Coal
Mid Screen Coal
Rock
Screen
Picking Table
Mechanical Cleaner
Rolls: Crushers, Re-breakers
Jig
Over
Through
Switch in Chute

Key to Fig 1, 2, 3

equiring a thorough washing to remove dirt and discoloration; or when r of the seam is such that the run-of-mine contains no large lumps.

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Class *C* permits no dry shipments and is
maintenance than Class *A*.

General design. Referring to the flow
vary the combinations of machinery, using
the product from one point to another, one
to the next smaller, gravity picking chute
different kinds of mechanical cleaners to re

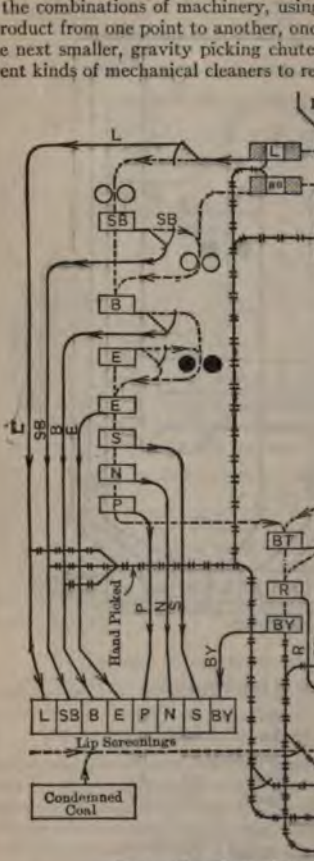

Fig 2. Flow-sheet of Dry

or revolving screens in place of shaking scr
tion are identical, and the diagrams are st

Ideal breaker (Fig 1) includes the fe
mine cars into the breaker receiving hopp
breaker, and feeders to all sizing and crus
that all coal may gravitate from receiving
with automatic devices to control flow of c
aration and in loading from storage pock

. when practicable, and shipment of dry coal; adequate outside facilities box and gondola R R cars quickly and with minimum breakage; sufficient

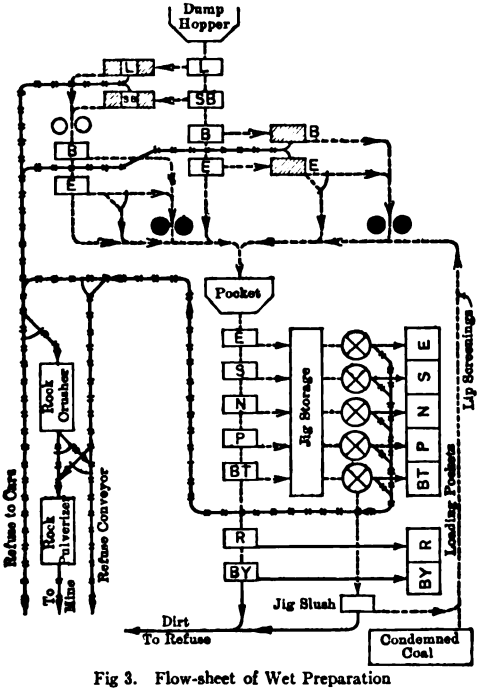

Fig 3. Flow-sheet of Wet Preparation

and loaded car standing track to hold a full day's output, tracks being to operate by gravity.

3. General Method of Preparation

line is the same for the 3 classes of breaker. Coal is dumped into receiv- pper and fed automatically, or by a hand-operated gate, on to the dump s at the top ("head") of breaker. Lump, steamboat, and broken are rom the run-of-mine (always the first 2 sizes, and often the broken also) e cleaned of impurities by hand-picking. This cleaned coal is the "pure- product, and is handled separately, except in a Class *C* breaker. It re- no additional attention except to be broken down, when necessary, sized elivered to loading pockets. The prepared and steam sizes from run-of- including the broken coal, when not handled with the pure coal) are "mud-screen product." This gravitates from dump screens to sizing creens, from which each size flows to its respective mechanical cleaner en runs to storage pockets. In Class *A*, the pure and mud-screen coal xed and go to same pocket. In Class *B*, these sizes are stored in sepa- ockets, when dry coal is desired or practicable; otherwise they are washed r and mixed. In Class *C*, the sizes are always mixed.

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Breaker refuse of slate, bone, or rock, as cleaners in all classes of breakers, is convey central point for final disposition, for which (*a*) Refuse is returned into the mines to fill coal. This method, called SILTING, involves large pieces of refuse, from broken coal and la material, together with all other breaker ref machine, reducing entire product to 0.5 to 0.7 mine through pipes (Sec 10). (*b*) Refuse BANKS, by chain or belt conveyer, or by du haulage, or by hydraulicking all or part of it,

Silting should be given preference and is super to the stability of the remaining mine pillars a surface disposition of refuse is necessary, the hyd topography of the region permits, and there is plen licking is not practicable, the conveying system i jacent to breaker is available. Banks can be can extended in length by added horizontal conveyin around top of bank. To eliminate trouble from refuse is crushed to about egg size. When bre of dump cars is advisable, advantage being that breaking.

4. Class A Br

Details of preparation (see Fig 1). Ship for, with arrangements to break down any si smaller size, or all sizes into stove, nut, and st ing market demands.

Run-of-mine is elevated to a receiving hopper a it is fed, mechanically or by hand, over a "bank" lump size, the remainder going through to the sec and the third removes broken. Material passi the MUD-SCREEN PRODUCT, and is sized into egg lump, steamboat, and broken gravitate to their r or stationary), are inspected, and the pure rock, by hand to a refuse or rock chute. Pieces of roc special tables, where the coal is chipped loose by

PURE-COAL PRODUCT, cleaned coal from pickin or, when there is no demand for lump, it is broke smaller. When there is no sale for steamboat, th as the lump, into broken and smaller. Pure-coal of 3 screens, sizing steamboat, broken, and egg. pocket or to the RE-BREAKER ROLLS, which may b and smaller. The pure broken coal from crushe from picking table and gravitate with it to its p going to the re-breaker rolls and the remainder t go to the re-breaker rolls. The pure egg coal re-breaker rolls, or may be divided in the sam product from re-breaker rolls flows to a broken pocket, while the undersize mixes with the prod above the re-breaker rolls, and the combined ma arranged to size pure egg, stove, nut, and pea coa

MUD-SCREEN PRODUCTS, egg, stove, nut, and p usually), contain impurities in excess of that sta removed, to keep within the allowable limit for from its sizing screen to a mechanical cleaner or s impurities, the coal flowing to its pocket and the sionally the mud-screen buckwheat requires clea similar manner; when such cleaning is unneces

ley are mixed with the corresponding pure-coal sizes and delivered to the buck-ice, and barley screens, where each size is separated and delivered to its pocket, e fine material through the barley screen goes with the refuse. If it is de-make buckwheat No 4, an additional screen is installed. While loading into shipment, the fine dust and screenings made during preparation are removed by the coal over punched steel plates or woven-wire screens, LIP SCREENS. Lip-oduct is elevated to the sizing screens to be re-sized and returned to the pockets. ading, the coal is inspected for size and purity, in conformity with Table 2. s in this inspection, it is condemned, unloaded, and elevated into the breaker and d.

gn in Fig 1 contemplates an ideal breaker. For 1 500 to 1 800 tons apacity, total height above loading track would not exceed 115 ft. Coal through breaker by gravity, thus eliminating conveyors and elevators. nveyor only would be required to handle the lip screenings and condemned Roll tests show highest percentage of prepared sizes when breaking only ne size to the next smaller; that is, steamboat to broken, broken to egg, stove. With this arrangement the breaker would require 4 sets of rolls, creasing the height, unless the re-broken material is re-elevated, but the e in prepared sizes would not warrant this installation. In practice, the oat and broken screens, set immediately below crusher and re-breaker espectively, are omitted; because a sufficient percentage of steamboat lly made from the picking table, in proportion to the other sizes shipped uired for the market, and the crusher rolls will break lump or steamboat into broken without oversize of steamboat, which would condemn the . Sufficient egg coal can be made above the re-breaker rolls, and any e from them will be carried by the stove. Mixing the pure and mud-prepared and smaller sizes is advantageous. When pure coal from the is broken down, each size will usually carry less than the percentage of ties stated in Table 2; hence, the mud-screen coal from the mechanical rs may contain an excess of impurities without causing the resulting re of pure and mud-screen coal to exceed the allowable percentage.

mple. Assume prepared sizes from run-of-mine to be 130 tons egg, 220 tons and 250 tons nut. Percentage of impurities in each size from run-of-mine is 14%. l coal, 45% is pure and 55% is mud-screen coal, all impurities being removed from re coal. Percentage of impurities allowable in mud-screen coal, without causing on of mixed product, is as follows:

130 tons egg × (100 − 14%)	= 111.8 tons clean coal	
111.8 × 0.45 (%)	= 50.31 "	pure coal
111.8 × 0.55 (%)	= 61.49 "	mud-screen coal
50.31 × 0.04 (% of impurities, Table 2)		= 2.0124
61.49 × 0.04 " " "	" "	= 2.4596
Total tons of impurities allowed,		4.4720

2 ÷ 61.49 = 7.3% impurities in mud-screen egg before mixing with pure egg.

same manner, mud-screen stove may contain 12.7% and mud-screen nut may n 18.1%, or in this particular breaker, it would not be necessary, theoretically, to e any impurities from the nut, as the mud-screen product, mixed with the pure would be within the limit of allowable impurities. Impurities in run-of-mine often the average, and to handle the varying product the cleaners are usually installed, hough figures indicate that they are not required.

5. Class B Breaker

tails of preparation (Fig 2). This type provides for separate shipment re and mud-screen coal of all sizes, except buckwheat, rice, and barley. ral method of preparation is practically the same as for Class *A*, except

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that it is not necessary to remove all impur
that portion which would condemn the fi
Class 1, then one mechanical cleaner on eac
pare the product, and jigs are generally used.
the practice is to install a preliminary mech
screen coal, to remove a percentage of the
product gravitates to a second mechanical c
going to pocket. Ordinary mechanical clean
to remove about 50% of impurities from a
stant attention; and where used as a prelim
move this percentage, the remainder being h
cleaner is under supervision of an attendant,
impurities in the product, so that the coal g
Broken coal is usually handled with the pur
with the mud-screen coal for Class 2. If a
larger than 2 cu ft, this lump coal is usually
rolls, another pair being used to break the
the large degradation into steam sizes resu
large lumps direct into broken size.

When dry coal is not desired, nor practica
are mixed after cleaning. The mode of clean
to that of class *A* breaker, respecting perc
screen product, before mixing with pure co
cleaners for each size is entirely dependent o

Example. Assume prepared sizes from run-o
egg, 90 tons stove, and 190 tons nut; percenta
20%, respectively; that 40% is pure coal and 6
purities are removed from the pure coal. Then,
carried by the mud-screen coal, so that mixed
number of cleaners for each size would be determ

With 20 tons of broken containing 32% impu
duce impurities to about 16%. If it is desired
remove remaining impurities by hand picking; b
gravitates direct from mechanical cleaner to re-b
with the mud-screen egg and smaller sizes, and t

90 tons egg × (100 − 35%)	= 58.50 tons c
58.5 × 0.40 (%)	= 23.40 "
58.5 × 0.60 (%)	= 35.10 "
23.4 × 0.04 (% of impurities, Table 2)	= c
35.1 × 0.04 " " " " "	=
Total tons impurities allowed	

Hence, 2.34 ÷ 35.1 = 6.6% in mud-screen egg, b
6.6% = 28.4% are to be removed by mechanic
would be used; first would be adjusted to remo
product; second cleaner would have to remove 17

In same manner, mud-screen stove may contai
two cleaners, unless jigs are used, which produce
nary cleaning. Similarly, mud-screen nut may
like stove. Preliminary cleaners on broken and
ing down, to eliminate part of the slate or rock
smaller sizes, increasing their impurities, and ma

6. Class C B

Details of preparation (Fig 3). This
all sizes smaller than broken, since it is usu
3 larger sizes with arrangements to break

-mine is elevated to a receiving hopper at head of breaker, from which
l mechanically or by hand into a bank of 4 screens which size lump,
oat, broken, and egg. Lump and steamboat are "skinned" on picking
that is, only a part of the rock is removed. After being broken down
zes are mixed with the mud-screen coal and prepared with it. Pure
id steamboat are crushed to broken in crusher rolls and flow to a bank
reens, which size broken and egg. Broken coal gravitates to the re-
rolls, and is reduced to egg or stove and the smaller sizes, going thence
tral storage pocket. Egg coal gravitates direct to central storage pocket,
ixed with the broken going to re-breaker rolls. Mud-screen broken and
usually run over preliminary mechanical cleaners. These sizes are
from the cleaners in the same manner as the pure broken and egg, and
n the central storage pocket. The unsized mixture of pure and mud-
oal is fed mechanically to a bank of 5 screens, sizing egg, stove, nut,
l buckwheat, each size gravitating to its respective JIG STORAGE pocket.
ocket is a reservoir for the jigs, accumulating coal during rush periods;
essary because jigs are operated at a fixed capacity. The jig removes
t impurities to make marketable coal, and from it each size flows to
pockets. When the jig is emptied, the hutch product, containing nut
l smaller, is washed over a JIG-SLUSH SCREEN, of a size to remove only
l smaller than barley, the oversize being re-elevated to the central
pocket. Rice and barley falling through buckwheat screen are handled
ss *A* or *B*. Sized coal is loaded and inspected as in Class *A*; lip screen-
condemned coal are re-elevated with the jig-slush-screen product.

ntage of bone coal in run-of-mine is usually small, and the sized products will
the bone in a seam, if the rock is eliminated (Table 2). This mode of handling
recommended above all others. When the bone percentage exceeds total allow-
e and slate, bone is handled separately, and the preparation here is identical
ss *A* or *B* breakers. A three-part separation is made of the run-of-mine on the
tables: coal, bone and impurities. Re-breaker rolls reduce this material into
tove and smaller, the broken-down product being mixed with the pure coal sizes
roportions which each will carry. Prepared-coal mechanical cleaners also make a
rt separation; bone from egg and stove is broken into nut and smaller, or the
ve, and nut into pea and smaller, and then screened and mixed with the sized
ng to pockets. Modern methods usually disregard bone coal, and make no
rovision to handle it, except to consider very poor or heavy bone as an impurity
d pick it into refuse from the large sizes.

coal, characteristic of certain seams, as compared with the more usual cubical
when occuring in sufficient quantity in mud-screen prepared sizes, is removed by a
picker before going to mechanical cleaner, thus avoiding the difficulty in removing
es from a mixture of cubical and flat pieces in a mechanical cleaner. Flat coal
its own mechanical cleaner, and the clean product is mixed with cubical coal
oing to pockets. The quantity of flat coal is often sufficient to spoil the appear-
prepared sizes, when loaded. When this occurs, part of the flat coal is removed
ken into smaller sizes.

7. Breaker Structures

es: (*a*) all timber; (*b*) all structural steel; (*c*) all reinforced concrete;
el base, timber superstructure; (*e*) reinforced-concrete base, timber super-
ure; (*f*) reinforced-concrete base, steel superstructure. To determine best
adopt, divide estimated cost by total estimated tonnage, and to quo-
dd estimated preparation cost plus yearly maintenance charges per ton
l.

reciation. TIMBER STRUCTURES. For dry breakers, depreciation depends
kind and quality of timber used, usually varying from 4% per year for

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Finger or fixed bars are occasionally use
size or dust from prepared sizes. They are n
spaced round W I or C I bars, or angle iron w
45°. Opening between bars has a ratio of le
50 : 1. Specially constructed bars are used t

Punched-plate or woven-wire screens
plngs from prepared sizes, especially for lip s
wire is used for dry and punched plate for w
water jet is directed on the screen to remov

ADVANTAGES OF FIXED SCREENS: low first cos
capacity when used as lip screens. DISADVANTA
pitch, they increase breaker height and cost; requ
adjustable bars involve a drop at lower end, and

Cylinder or revolving screens, usually
were formerly used for all sizes of coal; th
screens. They may be single or double jack

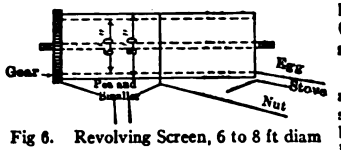

Fig 6. Revolving Screen, 6 to 8 ft diam

prod
(Fig
shou

AD
act s
slow
blind
high

with shakers under similar conditions; small cap
about one-eighth of the screening surface at one
screens; greater breakage in the screen and in the e
more difficult to inspect and repair than shakers; ex

Shaker screen (see also Sec 35) is usuall
Sides are of steel plate, angle iron or wood,
the punched-plate segments are bolted. It i
with pin-connected end, or 1-in oak or hick

Screen is driven by a pair of eccentrics connec
arms with fixed ends, and reduced in section to
or by forged crank arms, with metal connecting r
throw is 3 in, speed 150 rev per min. Ratio of
hangers, 3 to 6 : 1; for wood, 10 to 15 : 1. Han
of the screen, so that the vertical resultant actin
lift the coal out of the mesh and prevent blindin
to 3 in per ft, according to size of coal and len
built in 1, 2 or 3 decks, which should be hung in p
trics on the same shaft, 180° apart, to balance vib
due to: angularity of eccentric rods, the fact tha
varying quantities of material on the screens.
the breaker structure, which should be braced a
VANTAGES: exact sizing, low first cost, accessibili
of construction, reliability in operation, saving i
ADVANTAGE: vibration imparted to breaker struc

Side-shaking screen, now being developed
the end shaker. Design is similar to the above.
ft; speed, 250 rev per min, with 1.25-in stroke.
of motion, and down the pitch. Experiments sh
as compared with Table 6, which should result in
Better sizing is important, especially for egg coal;
size of stove in the egg is also broken down, ca
tests (Nov, 1914), on the side shaker, show the

ed with 8 to 20% with the present screen; also that a side shaker 4 ft wide by gives better sizing than a 4 by 15-ft end shaker, and is at least equal in capacity.

ting screen (*Trans* A I M E, xix, p 398) is now used only in old breakers and is eplaced by shakers. ADVANTAGES: exact sizing of prepared sizes, large capacity crease in breaker height. DISADVANTAGES: high first cost, high repairs and ance, inaccessibility for inspection and repairs, poor sizing of steam sizes, im- bility of proper balancing, and excessive vibration of breaker structure.

lating or movable-bar screens (*Loc cit*) are seldom installed in new breakers; are perseded by shakers. Their field is limited to dump screens, for sizing lump n-of-mine. ADVANTAGES: heavy construction, enabling them to handle large pieces rial; their action as feeders to the machinery following; saving in breaker height; ed; no vibration to structure. DISADVANTAGES: poor sizing; fixed space between th no adjustment; high first cost; when not hand fed from a hopper, the mixed run- often rushes over onto picking table.

Table 6. Comparison of Shaker and Revolving Screens

(Sq ft of screen area required to screen one ton of coal in 10 hours)

f coal	Shaking screen		Revolving screen	Size of coal	Shaking screen		Revolving screen
	Dry	Wet			Dry	Wet	
boat..	.....	1.50	.....	Pea.....	0.20	0.69	1.75
n.....	0.60	1.20	0.75	Buckwheat ..	0.61	0.53	2.00
.....	1.20	1.10	1.00	Rice.....	0.50	0.65	2.25
.....	0.25	0.35	1.00	Barley.....	0.67	0.67	3.00
nut....	0.20	0.27	1.5				

For sizes of openings, see Table 1.

lel revolving roller screen consists of a number of parallel rollers equally spaced. rollers (2 to 4 in) and opening between them vary with size of coal handled, open- g varied by adjusting the roller cen- creen is self-contained, and is usually d in a chute at a pitch down which the gravitate. Rollers revolve in same . and at speed of about 350 rev per This is not a sizing screen; it is used o remove flat coal or slate, and is led a "slate-picker." It is occasion- l as a lip screen.

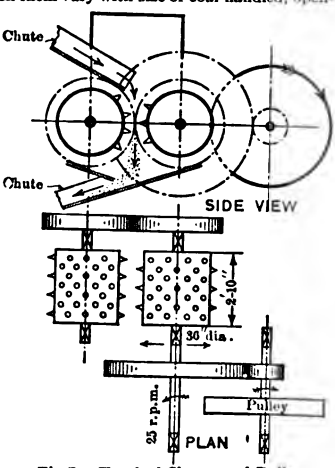

Fig 7. Toothed Slow-speed Rolls

10. Breaker Rolls

sification of rolls for breaking m a larger to a smaller size: R or No 1 rolls (for breaking steamboat or broken); MER- or No 2 (steamboat to broken RE-BREAKERS, or No 3 (broken or stove); BONEY or No 6 (egg stove, stove bone to nut or pea, e to pea). Rolls are also clas- y their peripheral speed: high- 00 ft per min) and slow-speed per min).

ation. Slow-speed rolls, with pointed teeth (Fig 7) give a higher ge of prepared sizes than the high-speed, under same conditions. Coal broken by points of the teeth, and *not crushed* between roll bodies.

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Hence, jaw and gyratory crushers should break the coal, thereby increasing degrad design, rolls are divided into DRIVEN-TOO first, hardened-steel teeth are driven into or drum; in the second, cast-iron or st drum or cast spiders. Pedestals suppo driven-roll pedestals being adjustable, i limited by the length of gear teeth. If g tainable by the gears in use, these are chan cushion springs, which compress when an

Table 7. Size of Rolls, Size and Spacing of

Roll name and number	Size of drum, in	Numbe per rov
Crusher No 1.....	51 × 41	8
" 1.....	33½× 46	8
" 1.....	35 × 34	6
Merchant 2.....	30 × 36	10
" 2.....	35 × 34	8
Re-breakers 3.....	30 × 36	14
" 3.....	35 × 34	13
Bone 6.....	24 × 32	21
" 6.....	24 × 32	21

Table 8. Percentages of Prepared Siz a Large

Roll number	Size of drum, in	Peripheral speed, ft per min	Size of coal broken	Stear boat
1	35×34	250	Lump	27
1	36×36	942	"	29
2	35×34	230	Steamboat	....
2	30×36	900	"	....
3	35×34	250	Broken	....
3	30×36	900	"	....
3	35×34	250	Egg	....
3	30×36	900	"	....

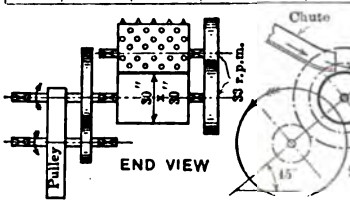

Fig 8. Smooth and Toothed Inclin

upper roll. Discharge chute is at cente coal without drop. Recent tests on the 12% degradation into steam sizes.

11. Mechanical Cleaners

sification. There are 3 groups, the operation of which depends on: ference in specific gravity; (*b*) difference in coefficient of friction; (*c*) ce in shape of fracture.

Plunger and pan jigs. Plunger types are generally preferred. A jig an overflow discharge breaks less coal than one equipped to elevate m jig-tank mechanically. The latter are largely used when jigs are to ed low down in the breaker, to save height. Advantage of a jig is that consistently treat a mixture high in impurities, yielding a product to market requirements, and discarding less than 4% of coal in the refuse. ful operation depends largely on jig tender's skill, and on the size of coal utomatic jigs are now being introduced. Breakage loss may vary up according to size of coal treated, and jig construction. Average maxi- rcentage of impurities to be successfully eliminated by a jig should not for egg, 25%; stove, 30%; nut, 35%. Capacity per hour: egg, 4 to stove, 4 to 12 tons; nut, 5 to 12 tons; pea, 6 to 15 tons. Usual sizes of rforation: for egg, ⅞ in; stove, ⅞ in; nut, ⅝ in; pea ⅜ in; buckwheat, (For further description and illustrations of coal jigs, see Sec 35, Art also, for jigging of ores, Sec 28, Art 20 and 21.)

ble 9. Loss by Breakage in Jigs, due to Mode of Discharge, and Abrasion of Coal in Jig-Tank

ize of coal	% loss into			Approx loss in value, ¢ per ton	Method of discharge
	Stove	Nut	Pea and smaller		
n.....	0.42	0.21	1.78	5.0	Overflow
.....	0.40	0.22	1.68	5.0	"
n }	0.98	0.56	4.09	10.0	Flight conveyer
n }	0.10	0.42	3.18	8.0	Belt "
.....	.....	.....	1.43	4.0	Flight "

Fixed and movable cleaners (friction). FIXED type may be designed a 2-part (slate and coal) or 3-part (coal, bone, and slate) separation. sually consist of a series of inclined planes, separated by adjustable cross Mixed material gravitates from top to first slot, where the slow-sliding d bone (due to their greater coefficient of friction and flatter shape) fall slot, the coal jumping over; this is repeated until the coal is suffi- cleaned. Refuse slate and bone falling through the slots may be simi- ated in another cleaner: the first slot removing slate, the bone jumping d so on. These cleaners are usually adjustable, as to slot opening and planes. They require constant attention, and hand pickers for the SPIRAL CLEANER consists of a central column, with a series of spiral nclined toward center, down which coal slides (see Sec 38, Fig 15). aintains a helical path around the column so long as friction balances gal force. As the velocity and centrifugal force increase, friction is over- d, and coal jumps over outer edge of spiral and falls into a chute. Slate e adjust themselves on the spiral, according to coefficients of friction, and arge end they may be separated and run to their respective destinations. require some attention and adjustment, since a change in character or n of the run-of-mine (as from wet to dry) will vary the coefficient of

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friction. Efficiency is affected also by dr
(see Sec 35, Art 9). MOVABLE CLEANER
adjusted to give a pitch in 2 directions: a
that moving band travels up the pitch.
corner; coal slides obliquely across the ta
coefficient of friction, is carried up the pitch

(c) **Fixed and movable cleaners** (fract
specially made bars, placed in a chute, forn
cubical pieces of coal slide, while flat slate an
slotted steel plates may be set in the chute,
holes or slots. This type is usually attached
Some forms have a slight advantage over otl
of flat slate than of coal. MOVABLE type
Art 9). Slate from cleaners of this type us
or (*b*) cleaner, for final preparation.

Comparison of mechanical cleaners.
when properly operated, plunger jig being su
jigs should be so installed that refuse and
convenient inspection. Jigs are high in fi
and use considerable water; but these disa
their good results. SLOT PICKERS are recom
ers. Breakage is usually high, and if adjus
likely to contain an excessive amount of coa
SPIRAL may be used as a final cleaner on pro
impurities, according to the size of coal; or
impurities exceed the above. When refus
easy inspection, one attendant can inspect
Breakage rarely exceeds 1.5%. MOVABLE
dance, as well as pickers for refuse and c
power to operate, and is high in first cost.

12. Conveyers, Chutes, Feeders

Conveyers are classified as: (*a*) single-str
(including scraping and carrying conveyers)
veyers with single strand may be used for al
double strand for larger.

Usual sizes of flights: 4 by 10 in to 6 by 12 in
for all other conveyers (condemned coal, refuse,
speed is 125 ft per min, which may be increase
conveyers working in acid water are to be avoid
changes the pitch and causes trouble. Cast-iro
water better than steel, and should be used in per
from 2 to 4%, according to method of feed and di
action of acid mine water deteriorate rapidly. Ne
up of 3 or 4 equal pieces of different kinds of bel

Elevators should be double-strand and o
breakage at discharge end is less than for
is from 2 to 5%, according to method of fee

Chains for elevators and conveyers workin
contain few parts, with safety factor of 6 to 10.
not be used. Rivetless chains having few parts a
power be required. To reduce extra parts carri
chains at any one colliery should be of same ty

ined chutes for conveying coal by gravity, if badly constructed, cause **adation loss exceeding all other losses combined.** This loss is attributo **irregularities** in chute bottom; striking of one piece of coal against r; **drop at any point,** especially at right-angle turns; and the blow a piece of coal receives at such a turn when it strikes side of chute. on which coal will slide varies with quality of coal, and on chute lining. coal will slide on flatter pitch than a mixture of coal and impurities. A n section of inclined chute can often be remedied by replacing steel

ble 10. Pitch and Width of Chute, for Coal Sliding on Steel Plates

of coal	Pitch, in per ft	Width of chute, in	Size of coal	Pitch, in per ft	Width of chute, in
.....	2½ to 2¾	48	Pea.....	4 to 5	12 to 18
boat.....	2¼ to 3	36	Buckwheat.....	4½ to 6	12
n.....	2½ to 3	30	Rice.....	5½ to 7	12
.....	2¾ to 3¼	24	Barley.....	7 to 8	12
.....	2¾ to 3½	18 to 24	Dirt.....	8 and over	12
nut.....	3 to 4	18 to 24	Rock.....	5 and over	18 to 60

for bottom of dump hopper, not less than 5 in per ft; for storage pockets, not 7.5 in per ft.

ith bronze, if impracticable to increase pitch. Chutes for dry coal are 2-in plank lined with sheet steel. For wet coal, use 2-in plank, lined ply smooth coat prepared roofing, covered with ⅞-in tongue and groove , and lined lastly with sheet steel or bronze. Right-angle or 180° turns es should be built of laminated wood strips, ½ by 2 in, nailed together.

cal chutes, "telegraphs," may be designed merely for lowering coal, Fig 9*a*, owering and depositing to variable depth in a pocket, Fig 9*b*. In the latter nen top of pile reaches and blocks lowermost opening, coal then issues from gher opening and deposits on summit of pile; breakage due to subsequent

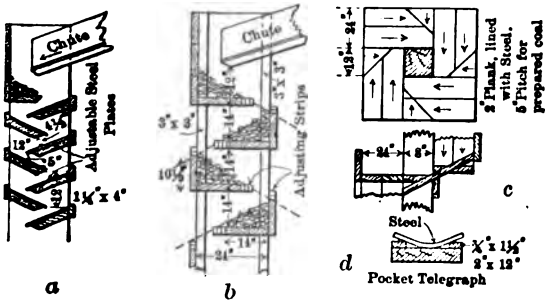

Fig 9. Vertical Chutes or "Telegraphs"

hing down slope of pile is negligible. Capacity of such telegraph, 24 in is 150 tons per hr; 48 in square, 300 tons per hr; it is used for all sizes of coal g down, and may be of any height up to 65 ft. Breakage in a well-built h is only about 1%. Spiral telegraph is shown in Fig 9*c*. Automatic mechanism, Fig 10, is designed to keep telegraph full of coal, by discharg- as fast as fed, thus reducing breakage. Coal entering the top presses u gainst pan *a*, thereby opening gate *b* at bottom, through rods and leve

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Automatic feeders are installed betwe
screens, and in front of rolls. Feeders m

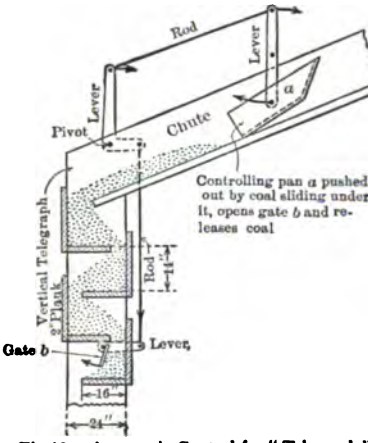

Fig 10. Automatic Control for " Telegraph "

shaker screens, of 60 to 90 sq ft, 2 to 2.5 in; j
1.5 to 2 in. A fan-tail spray nozzle directs
and against the flow of coal. Gate valves a
plug valves at sprays on shaker, quick openi
line arranged to operate from the loading p
for cleaning.

Engines. Breaker machinery is usually driv
stalled in a separate building free from dust.
main line shaft, from which power is distributed
chinery to which they are tributary are often dr
clutches from main line shaft. All drives from
clutches, and the drives combined so that mud-s
demned-coal and rock-handling machinery, may
operated breakers have advantage of unit drives

Table 11. Horse Power for Oper

Machine	
Revolving screens, 6 to 8 ft diam by 24 f	
Shaking screens, single deck, 60 to 90	
Oscillating bars, 60 sq ft	
No 1 crusher rolls, 36 by 36 in	
No 3 re-breaker rolls, 30 by 36 in	
Horizontal conveyers 100 ft long at 10	
Size of flight	Capacity,
6×12 in	
6×18 in	
8×24 in	1

ble 11 (*Continued*).—**Horse-power for Operating Breaker Machinery**

Jigs				
Size of jig	Size of coal	Capacity, tons per hr	H p	R p m
4×8 ft	Egg, stove or nut	8 to 15	2.3 to 4.5	90 to 95

Elevator			
Size of bucket	Height	Ft per min	H p
22 by 24 in	80 ft	100	13.4

able 12. **Total Breaker H p, with Number and Kind of Machines**

er ty,	I H P		Machines, number and kind					
	Light	Loaded	Rolls	Shakers	Picking tables	Conveyers, No and length	Elevators, No and length	Jigs
	216	280	7	33	3	2–185 ft	3–184 ft	.....
	342	512	4	34	.....	2–135 "	4–210 "	32
	354	471	4	35	.....	3–225 "	4–240 "	.....
	274	336	3	36	1	3–220 "	1– 80 "	30

13. Breaker Costs. Number of Employes

of breakers varies with location, type of construction (Art 7), and of preparation (Art 2 to 6).

Table 13. Cost of Breakers, Cents per Cu Ft

Construction	Foundation	Breaker structure	Machinery	Labor	Total cost per cu ft	Volumetric capacity, cu ft
er.....	$0.014	$0.10	$0.071	$0.036	$0.221	140 000
.....	0.006	0.051	0.052	0.0248	0.1338	664 000
.....	0.0045	0.0076	0.0705	0.0368	0.1194	754 000
ete and steel.....	0.0056	0.0085	0.0865	0.077	0.1776	869 000
ete and timber..	0.0083	0.0041	0.0368	0.023	0.0722	1 337 500

ght of structural steel breaker approximates 1.5 lb per cu ft volume; not exceed 1.75 lb.

ble 14. Cost of Preparation and Maintenance per Ton, and Daily Capacity of Breakers

d breaker (Art 2 to 6)	Preparation cost	Maintenance cost		Total cost	Tons per day
		Labor	Material		
A.....	$0.069	$0.014	$0.018	$0.101	1 650
B (dry).....	0.083	0.013	0.019	0.115	1 790
B (wet).....	0.098	0.043	0.049	0.190	2 030
C.....	0.053	0.017	0.026	0.096	1 120

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Cost of breakers is divided in about the fo
given:

Excavations	0.37%
Foundations	6.23
Floors	1.20
Structure	34.60
Lumber	8.25

Table 15. Average Number

	Class A. Run-of-mine, 14% refuse
Boss	2
Ticket taker	1
Dumpers	2
Platemen and table tenders	14
Pickers on pure coal	8
Pickers on jig coal	.....
Table refuse, pickers	.....
Jig refuse, pickers	.....
Mech. cleaner, refuse picker	4
Mech. cleaner, tenders	2
Mach'y attendants	3
Breaker oiler	2
Ropeman	1
Breaker cleaner	2
Engineer	1
Loaders	12
Total force	54
Tons per day	1 650

STORA

14. Location and Requiren

Irregularities of market for all sizes of an
with continuous operation of the mines, have le
direct control of the mining companies. Stora
venient for traffic. They comprise the whol
amounting to approximately 6 000 000 long t
retail storage yards, in cities and towns throug
usually vary from 1 000 to 3 000 long tons,
sumers' coal bins are the great storage units
not under control of the producers, customers
reduction of price, on April 1st of each year, an
after added each month until the normal is rea

Seaboard plants are situated at or
deliveries to consumers, and less interfe
labor difficulties. Disadvantage: capit
charges.

Local plants are situated in or near
short haul from the mines and quick rele
shortage); low freight charges on coal sto

t. **Disadvantage:** unavailability during labor troubles or interrupted portation.

erior plants are on or near the Great Lakes, especially at Buffalo as a ship- oint, and at receiving points in Minnesota, Wisconsin, Illinois, Canada, few railroad points at important junctions west of the Alleghanies.

l storage plant should fulfil the following conditions: (*a*) Production of mini- reakage in stocking and reloading; (*b*) separate storage of each size and in varying ties; (*c*) rapid handling in storing and reloading (docks, on the Great Lakes, ng coal by steamers, should have large unloading and stocking capacity, as quick h of boats is desirable; (*d*) screen house to resize all coal from storage; (*e*) sep- storage for screenings and facilities for resizing them; (*f*) arrangements for re- and screening more than one size at a time; (*g*) loading facilities for both box ondola cars; (*h*) covered storage when in regions subject to extreme cold and snows; (*i*) facilities to handle frozen coal, especially at plants receiving wet coal he mines or at uncovered plants; (*j*) ample railroad classification yards; (*k*) am- ckage through plant, operated by a car haulage system; (*l*) loaded and empty les; (*m*) minimum danger from fire; (*n*) design permitting enlargement; (*o*) low st per ton of capacity, and minimum operating cost.

zen coal is most efficiently thawed by hot water; steam jets are of little use. should be ample provision to handle the drainage, which contains much fine dirt.

eration of storage plants is very irregular. Night and day rushes, in ding or reloading, are often followed by long periods of inactivity, with ant either full or empty. Low operating cost, when the plant is working, e offset by high fixed charges during idle periods.

15. Classification of Storage Plants

h of the two general classes, wholesale and retail, is sub-divided into nechanical and mechanical.

nechanical	
ocking on surface	Reloading by hand, yard reloaders, or steam shovel
ocking on surface from trestle	Reloading by hand or yard reloaders
ocking on surface from trestle	Reloading by tunnel, with cars or conveyers to a screen house, with or without yard scrapers
ocking from trestle into bins	Reloading from bins into cars or wagons over lip screens
ocking from trestle on hillside	Reloading by hand, dock scrapers or by hydraulicking
anical	
ocking by elevator and conveyers into bins	Reloading by tunnel as in *C*, or direct into cars or wagons, as in *D*
ocking by cable railway and dump cars into bins or on surface	Reloading by hand, tunnel, or from bins, as in *C* and *D*
dge system. Stocking by truss trimmers in conical piles	Reloading by swing conveyers, or pivotal reloaders with inclined conveyers to screening towers
ocking on hillside by traveling cantilever trimmer	Reloading by gravity and hydraulicking
ocking on surface by traveling trimmer	Reloading by longitudinal and cross tunnel conveyers, fed by gravity and traveling reloaders, delivering to screen house
ocking by surface cable cars, operating on traveling tramways	Reloading by tunnel conveyers to screen house
ocking by grab buckets, operating on traveling or fixed tramways	Reloading by grab bucket and conveyers to screen house

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16. Application and Outline of the Diffe

A. Limited to temporary storage of steam sizes level surface, laying tracks on the accumulating sto the storage grows. Reloading with steam shovel o edge of pile, or by hand. Only one size can be sto is excessive, no screening is possible, and operating almost prohibitory.

B. Commonly used for small retail yards, when Consists of a trestle, the height of proposed pile, Sizes are separated by partitions. Reloading by Only hand screening is possible. All sizes may be to high drops in stocking. Operating cost, appro advantage.

C. Occasionally used for large storage, wh more expensive plant. Stocking is done by t tunnel under the storage pile. Reloading ga to a belt or scraping conveyer, which transf point or screen house. Lip screens at loading ing when loading cars. Screenings are collect transferred by hand to a screening storage. A mately 60% may be reclaimed by gravity, t hand or yard scrapers, with a partial screeni is excessive, due to high drops in stocking a in reloading. High first cost, imperfect sizin mating 10¢ per ton are the disadvantages of t

D. For large retail storage plants, for whol interior plants loading boats or steamers. C road tracks, from which each size is dumped i is direct into cars or wagons, which pass und gates in each bin feed the coal over lip screen may be collected in dust boxes under the s conveyers to screenings storage. ADVANTAG quantities, and reloaded by gravity; can unl at the same time; partial screening. DISA due to high drops in stocking and drawing co imperfect sizing; operating cost approximat approximates 5¢ per ton handled.

E. For large storage, of pea coal and small slope of 25° to 30°, at least 300 ft wide, with a this slope to a level surface for tracks; reason the storage hill and down again. Coal is sto hill, and avalanches down the hillside until a a level space at bottom. Reloading is throug layer retained by the wall and lying above gravity to reloading gates. It is usual to rel scrapers, swing conveyers, or by hydraulick Fixed screens in front of loading gates rem which may be conveyed by cars or conveye house. Lip screens not used on sizes small sizes are separated by partitions, made mova ties. Entire hillside should be protected b intermixture of dirt. Operating cost approxi

F. For small retail storage, where location prob an easy grade. May be developed for large storag

R cars into a receiving hopper, which feed into an elevating and scraping or carry-nveyer, transferring coal to storage bins. Automatic shovel unloads box cars into ng hopper. Reloading is direct into cars or wagons, through gates with lip screens delivery chutes, the screenings collecting in a box underneath. Lowering chutes graphs in each pocket reduce high drops when stocking. Stocking conveyer may e storage building, the lip screenings gravitating from the storage box to the lower the conveyer; then re-elevated and discharged into a screening bin. The space en under side of pockets and the surface may be used for storage by stocking through in bin bottom. This stock coal is chuted to the lower run of the conveyer and ed to pockets.

For large storage plants or retail yards; especially to transfer water-ed coal to storage or direct to screen house. Consists of a cable or gravity n car, traversing bins or a trestle over a surface storage floor, and dumping desired points. Reloading is by hand, or tunnel, with conveyer. Method eening is similar to that of *C* and *F*, or when well-sized coal is desired, it be re-screened in a separate screen house, as with *K*. Low in first cost peration, readily adaptable to extension and for covered storage. Break-nto pea and smaller, may be as low as 8%, but is generally higher.

Dodge system, used for large storage, consists of storing on the surface nical piles by a trimmer truss supporting a flight conveyer. A unit has mmer trusses with track hopper, one central reloader, and a screening , with loading tracks and scales. Loaded cars are dumped into a receiv-opper and fed into the trimmer conveyer, which is built on a catenary from under the hopper to meet the incline of trimmer truss. Coal is arged at apex of the growing pile, through bottom of conveyer trough. om of trough is a flexible steel ribbon, which is pulled up to vary the dis-e point as height of pile increases. Reloading is by a pivotal reloader ed between the two conical piles), which carries a flight conveyer on its Reloader travels on a curved track, and is drawn against the edge of ile by wire cables winding on drums placed on the reloader. Coal is drawn e pivotal point of reloader, where the conveyer is inclined, and is carried reening tower. Shaking screens in this tower size the coal and deliver it t into cars. Screenings are collected in a hopper and transferred to a n house for sizing. Driving engines or motors are located near reloading r. Trimmer conveyers are usually rope-driven, to minimize weight on . This type may be readily enlarged by adding additional piles. OPER-G COST, which averages 5¢ per ton handled each way (total 10¢), is gov-d by the activity of the plant. BREAKAGE in the prepared sizes varies approximately 6% for egg to 2.5% for nut.

is type requires large area in proportion to its capacity. A 120 000-ton plant covers 00 sq ft, as compared with 105 000 sq ft for a modified Dodge plant, not including s. Practically all the coal is reclaimed by machinery. A modified Dodge plant lly has fixed trimmers in a storage building, with bulkheads between piles. Reload-s by a tunnel conveyer under each pile, delivering each size to its respective screen e, and thence to a storage pocket to be loaded into cars.

For large storage plants, especially for steam sizes. Consists of a side e, as described under *E*, and single-track trestle with continuous bin chutes fficient height to permit feeding into a traveling cantilever trimmer, by ch the coal is elevated and discharged onto storage floor. Cantilever trim-comprises a platform, mounted on trucks, traveling on a broad-gage k, parallel to dumping trestle. It carries a cantilever truss, and a scraping eyer with a movable trough bottom, which permits discharge at any de-d point. To increase capacity and prevent stored coal from flowing back rimmer track, a bulkhead may be erected between storage floor and track, ading as in *E*.

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When surface contour will not permit an ap
grades, a loaded car plane can be installed. This
which they should be operated by gravity to the
the cars are lowered to the empty stand track.
tom into a pit, should be used on each plane. Pl
tracks and loaded-car stand tracks, operated by
and shipping. Hot water should be supplied for
Operating cost is close to 3.5¢ per ton, depending
when handling steam sizes; no exact figures avail

J is a modified Dodge plant, consisting of a s
chutes, on which cars are handled by a rope h
desired. Hand-controlled gates feed coal from
supported on a traveling truss. This traverses t
wedge-shaped pile with rounded ends. Lower
moving on a depressed track, upper end being su
an elevated single track, which rests on steel t
Each column is stayed by guy ropes, anchored
by tunnel conveyers, placed under the line of colu
which takes the coal beyond edge of pile and ele
is sized over shakers and delivered into loading
is tributary by gravity to reloading conveyers,
remainder is delivered to the gates by a traver
truss, mounted on wheels and carrying on its edg
against the face of storage pile by steel cables,
and passing around propelling sheaves mounte
for operating trimmer and propelling trimmer
allel to storage floor, and through collectors on t
supplies power to reloader from plug stations o
screen house may be operated by steam, but p
include extensive railroad yards for receiving a
mates 2.3¢ per ton, during average active perio
type *H*.

K. For large storage plants and occasio
by fixed or traveling tramways, spanning
railways. This type is readily adapted to
or barges, especially when storage floor is
or dock. Steeple unloading towers, at dock
elevate coal to a receiving hopper, under w
releasing a gate in the hopper which allows
charged into passing car. Cars traverse a
it, automatically stocking the coal at any
engages with a tripper. Reloading may be
ferring product to a screen house.

Quantity of coal reclaimed by gravity depends
remainder may be handled by dock scrapers. Ca
ferring coal direct from unloading towers to scre
storage, in cable cars operating in connection w
type permits stocking of separate sizes in varying
between sizes, or by leaving valleys between them
Increase in speed of stocking is accomplished by
is especially adapted to irregular areas, using fi
covered storage. Operating cost approximates
sizes varies from 8 to 15%, due to high drops in

L. For large storage, especially at interior
which receive coal by water. Grab buckets
used for stocking. Storage floor parallels th
or more bridges, which may be propelled a
one end of floor to the other. Bucket is sus

across the bridge on a track hung from the lower chord; a hinged boom, tracks matching those on the bridge, extends over the vessel so that the t may unload direct from boat into storage. Reloading may be done by l conveyers, or a conveyer may be placed at rear of storage floor, to re- coal from the grab bucket, either direct from vessel or picked up from stock. conveyer transfers the coal to a screen house.

COVERED STORAGE, this type may be modified by placing fixed trolley s in the storage building, which match tracks on a traveling unloading , so that one tower may serve several tracks.

versing trolley carrying the bucket may be operated by one of two methods, as "rope" or "man trolley." In the former, the bucket-hoisting and trolley- ing engine or motor is built into the bridge and connected to the bucket and trolley re cables. The operator, from a convenient point, controls the bucket and trolley. ontrolling stations are often installed, one overlooking the boat, for unloading, and her arranged for best supervision of reloading. In the electrically-operated man- . the hoisting and traversing motors are mounted on the trolley, with the control- vers in a cab suspended from it. Usual bucket capacities are 1.5 to 3 tons for the r and 5 to 7.5 tons for the latter. Rope trolley is cheaper in first cost, due to light raction required as compared with man trolley; but is higher in breakage, since the rolley operator is always in view of his work, and can better lower the bucket onto e pile when stocking, thus decreasing the drop and degradation. It also saves by ling and reloading in large units. Fixed bridges may be arranged to span the house, thus eliminating reloading conveyers; but these are limited to small plants equire a bridge and screen house for each size stored.

s type is adapted to stocking separate sizes in varying quantities, with or without eads, as under *K*, but does not lend itself to enlargement. It is usually rapid in ng, the speed depending upon skill of operator; but speed of reloading is limited by reen house capacity. Hoisting speed is about 500 ft per min; trolley speed, up to ft per min. Operating cost is close to 3¢ per ton; degradation, approximately 10%.

17. Ideal Screen House

sential elements. 1. Screens to remove undersize from each size reloaded.
Storage pockets for each size handled, with capacity to hold all the undersize made the size being reloaded, unless sales orders can be so arranged as to permit loading rsize as it is made; otherwise the undersize must be returned to storage when its ing pocket is filled.
Lip screens to remove pocket breakage.
Separate screenings storage floor, with transfer conveyers from screen house to ge, with arrangements to return them to the screens for re-sizing.
Arrangements to load either gondola or box cars.
Minimum breakage.
Minimum force of men, and low operating and first cost.

esign should include a receiving hopper, with feeder to shaker screens, for each size handled, arranged one above the other and directly over the kets. Each size of coal should gravitate directly from its shaker to its et, using telegraphs for reducing breakage. Loading into cars may be as at mines, with preference for belt conveyer system (Art 8). All sizes ve pea should pass over lip screens, the screenings being transferred to nge. Local and interior plants receiving box cars should be equipped with hanical BOX CAR LOADERS.

lting or gravity loader is superior to all others, and has a capacity of 12 box cars per hr. loader, without foundations, costs $10 000; foundations vary from $4 000 in clay, 10 000 in soft water bearing soils requiring piling and a waterproof pit. Less expen- loaders cost from $2 500 to $3 500, but their capacity is less by half than the gravity , and the breakage is higher. COST of a screen plant approximates $10 per ton of ling capacity, not including foundations.

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18. Miscell

Mechanical car unloaders for gondola
points at seaboard or on the Great Lakes,
Car dumpers operate by elevating the car to
gravitate into the vessel, at which point ca
run out. Cost is approximately $80 000;
unloaders or dumpers may be constructed lil
first of these to be erected was at Buffalo,
in 1915. Their cost is approximately $15 oc
hr. Average rate for unloading box cars b

Breakage of coal. This always occurs
from 5 to 25%, according to method of ha
coal, drawing coal under pressure from sto
structed chutes, and handling by convey
BREAKAGE FROM DROPPING varies with diffe
impossible to avoid it entirely, it may be re
or on itself; tests show that when sized cos
angle of repose by avalanching in large mass

Table 16. Breakage of Coal Du

Size	Breakage in smaller pre pared sizes
Broken	3%+0.43 *L*
Egg	4%+0.43 *L*
Stove	2%+0.33 *L*
Nut	
Pea	
Buckwheat	

Note. — *D* is drop, in feet.

COAL UNDER PRESSURE from deep bins or
breakage, though often neglected. When re
breakage is reduced by opening first those
the pile. CHUTES, poorly constructed, cau
constructed in same manner as in breakers (
CONVEYERS cause breakage of from 2 to 5%
latter. BELT CONVEYERS cause no breakag
discharge is often high unless discharge ch
GRAB BUCKETS cause some breakage, the pe
to the length of cutting edge per unit of c
cubic capacity of the bucket the less the deg

19. Cost of Storage Plants

ost varies with type of construction.
llowing data are sufficiently accurate for making estimates.

ype	Capacity, tons	Cost		
		Total	Per ton	
A	.....	.....	.....	Only cost is for tracks, at $1.20 per ft, plus cost of a steam shovel
B	2 788	$2 346	$0.83	Standard timber R R trestle, 12 ft high by 204 ft long, foundations and rails not included
B	2 280	727	0.32	Cheapest kind of wood trestle for industrial cars. Coal elevated to cars in tubs, handled by steam-operated boom derrick. Cost is for trestle only, including sills on surface for foundation
C	2 788	4 080	1.46	Standard timber R R trestle, same as *B*, with 6 by 6-ft concrete tunnel for cars
D	3 000	30 900	10.30	Two-track, covered wood structure, 96 by 44 by 30 ft to top of rails; 16 pockets, with gates delivering into wagons; two driveways under bins; corrugated roof and sides. Cost includes foundations and trestle approach, 96 ft long
F	1 000	7 000	7.00	Covered wood structure, with elevated pockets and outside driveway. Elevating and scraping conveyer encircling pockets, with surplus storage under bins, reclaimed by stocking machinery. Cost includes structure, foundations, machinery, with gas engine
nd L	2 000	15 800	7.90	Covered wood structure, with elevated pockets and two driveways under bins. Encircling conveyer and elevator. R R cars unloaded on separate trestle, with continuous bin chutes feeding to stocking machinery. Cost includes plant above foundations
G	25 000	35 000	1.40	Open steel trestle, 216 by 72 by 35 ft high, with columns 24 ft c-c, traversed by a cable railway with 20 3-ton dump cars. Reloading by hand. Cost includes foundations. Length of haulage, 2 500 ft, including a timber trestle 1 000 ft long
H	120 000	79 500	0.6625	2 units, 333 ft diam by 85 ft high } Cost does not include foundations, track hoppers, power plant, screen house, nor tracks
	100 000	72 000	0.72	" 313 " " " 80 " "
	80 000	65 000	0.8125	" 293 " " " 74 " "
	60 000	59 800	0.995	" 263 " " " 67 " "
	30 000	46 200	1.54	" 208 " " " 53 " "
and L	100 000	298 370	2.98	Modified, covered Dodge plant, 480 by 230 by 90 ft high at center, to store 3 sizes; 3 trimmers, 3 tunnel reloading conveyers, separate screen house for each size, 3 transfer conveyers, to handle direct from railroad cars to screen house, bulkheads and machinery and boiler plant..... $200 000 grading and foundations.... 41 000 eleven miles of track..... 57 370

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Type	Capacity, tons	Cost		
		Total	Per ton	
I	225 000	.....	1.25	Covers c sloping ing w and en ering t erator, house, plant a
J	383 000	.....	1.45	Covers c traveli suppor track c ers, lon nel an house, house, ity bo railroa founda
K	75 000	311 379	4.15	Complete storage iron; c operate steel t plant a with 1 veyer t box-car founda A retail st screen h long, co veyer
L	125 000	275 906	2.10	Complete founda buildin and 6 t 2 travel ($43 75 house a box-car lighting

Table 17. Operating and Total Cost, in Cents per Ton per Year

Operating cost			Repair cost		Overhead cost		Tons handled per year		Average number employes	Degradation cost, cts per ton	Total cost
Unloading	Reloading	Engineers, fireman, fuel and light	Buildings	Machinery	Superint'd and clerks	Office, taxes and insurance	Stocked	Reloaded			
0.11		....	0.075	....	0.033	0.047	18400	18100	6	0.03	0.295
0.17		....	0.049		...	0.068	9400	8900	2	0.039	0.326
031	0.105	0.07	0.021	0.04	0.057	0.072	25100	35000	10	0.10	0.496
031	0.047	0.015	0.004	0.018	0.013	0.011	177600	186300	32	0.029	0.168
015	0.018	0.032	0.0029	0.024	0.016	0.022	166000	126000	25	0.11	0.2399
0045	0.0078	0.027	0.006	0.004	0.009	0.011	134700	174600	15	.....	0.0693
019	0.035	0.017	0.019	0.0052	0.013	0.0086	119600	133900	27	0.105	0.2218
015	0.045	0.022	0.007	0.024	0.019	0.030	154400	129000	24	0.25	0.412

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SECTIO

PREPARATION AN
BITUMINO

BY

H. McKEAN
MINING ENG

Art	DRY SCREENING	Page
1.	Gravity or Bar-screen Tipples	1874
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12.	Coal Analyses and Commercial Sizes	1891

Note. — Numbers in parentheses in text ref

Introduction. Preparation of bituminous c
that of anthracite. Coal is delivered from m
wood, steel or concrete. For the purer coals,
by screens. Impure coal is hand-picked for
say 2.5-in and under.

General considerations. Sizing is by gra
screens. Most tipples are equipped with first
Connellsville, Pa, coke region, and are being
eastern Kentucky. Shaker-screen tipples repl
fines. A tipple may readily be changed from
struction as a whole being no greater than if c

Early in the industry the only preparation
of-mine (r o m) coal, while loading into R R ca
more uniform sizing promotes efficiency. Loa
1.25 in) is practiced at many small mines.

DRY SCR

1. Gravity or Bar

General design. Coal is dumped fro
(For details of several types of dump, s
over dead plate *B* (Fig 1) to screen bars

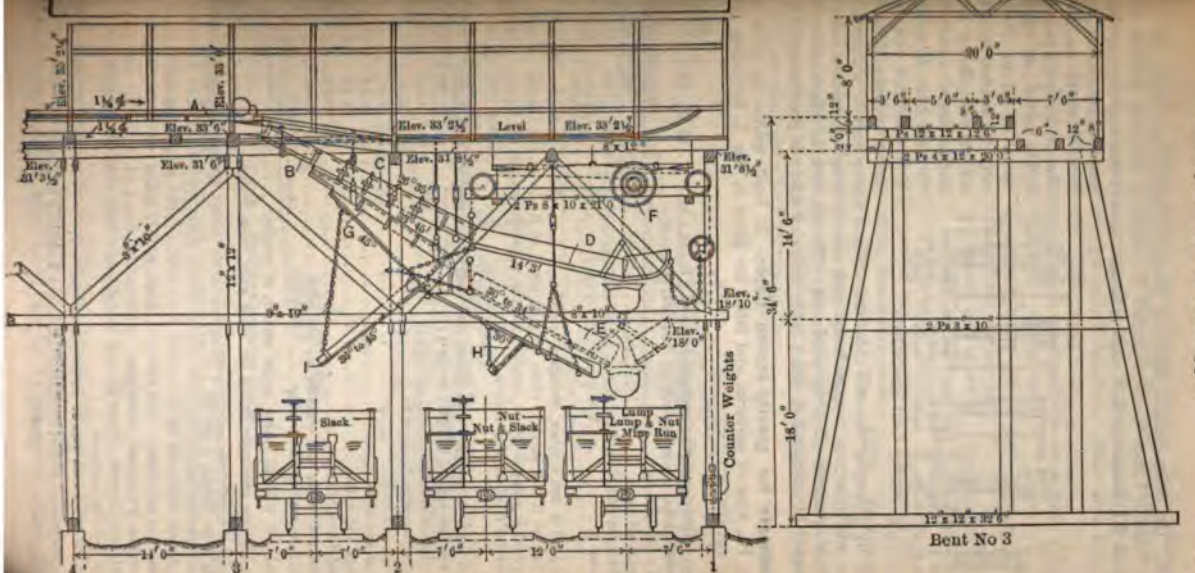

Fig 1. Gravity Wooden Tipple

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pitch. (In other designs shaker or revol
size ("lump") goes to weigh-basket *D* (
lines at *E*), thence to R R cars. Loade

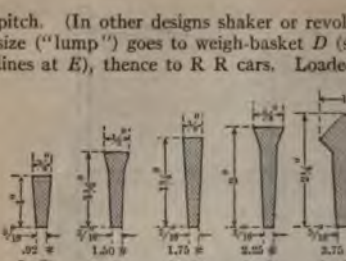


Fig 2. Coal Screen Bars

made by changing the movable bars. I
With gravity-screen tipples lump coal o
basis of quantity of lump produced. T
indicated by calculation, but a large fact

Estimate for gravity-screen tipple burg, Pa, cost of material and labor for (Jan, 1914):

Excavation (assuming average depth of 3 ft)	
Foundations, 44 yd (concrete mixture, 1 :	
45 bbl cement	
15.8 yd sand	
24.4 yd stone	
Labor	
3 679 bd ft oak lumber, standard lengths,	
15 555 bd ft oak lumber, extra lengths (ove	
3 372 bd ft 3-in oak plank, random widths, st	
1 700 bd ft sheathing boards	
3 000 sq ft corrugated iron siding and roofir	
20 window sashes, 6 lights 10 by 12	
10 window frames	
150 boat spikes 3½ by 8 in	
2 kegs 40d nails	
1 keg 20d nails	
2 kegs 10d nails	
1 keg 8d nails	
Labor unloading and erecting 24 306 bd ft	
Labor on roofing and siding	
Total cost without dump, scale, screens or t	
One Phillips crossover dump for 2-ton cars, f o b Pittsburg	
Freight on dump, 3 500 lb	
Labor erecting dump	
Freight on scale, 500 lb	
Labor erecting scale	
Screen rig, with scale, complete	
Freight on screens, 21 000 lb	
Labor erecting screens	
Mine car tracks on top of tipple, complete	
Total estimated cost of complete tipple, not i ing tracks nor grading	

k-back dump. Oldest style is horn dump, Fig 3*a*. It has been im- by addition of springs to assist returning of car after dumping.

lips automatic crossover dump (Fig 3*b*). Standard design is for 2 to ars, 42-in gauge; heavier design built if required. Plate *abcd* is 0.25-in late, or thicker if necessary. Riveted to it are kers. Treadle *A*, generally in right-hand rail, nected to horns *BB'* by chain *C*. Horns are d to shafts *DD'*, set in journal boxes *EE'*. r as shown is empty. A loaded car coming in nto treadle *A*, pressing it forward and down- thereby, through chain *C* and connecting-rods rcing the horns outward. This releases empty hich, on being bumped by loaded car, runs for- ver the crossover rails, while the spring pulls he horns in time to catch next loaded car. The s then released by hand lever *G*, and, being over- d by the weight of coal, tips forward. It is righted by counterweight *H*. ake control is on friction bar *I*. Weight of dump shown, 3 500 lb; capacity, cars per min; price f o b Pittsburg, $250.

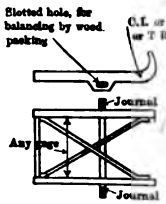
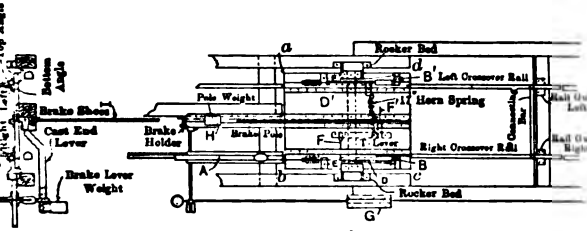


Fig 3*a*. Kick-back Car Dump



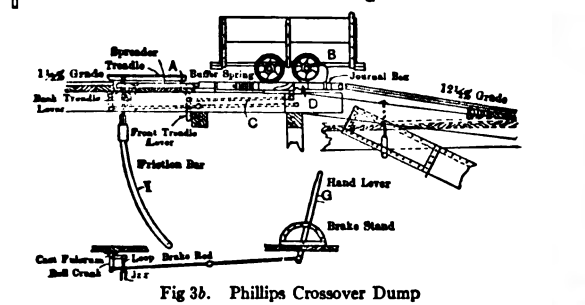


Fig 3*b*. Phillips Crossover Dump

rey crossover dump holds car in dumping position on a steeply inclined ection of rail continuous with tipple track, by a pair of horns which rock ds under control of a band brake on a drum keyed to same shaft; horns unterweighted to bring them back to starting position. Maximum ty, 8 cars (of 3 500 to 5 000 lb) per min; weight of dump, 3 500 lb; price, olumbus, $345. This, and 2 preceding dumps, require end gates on cars.

ary dumps may operate either as crossover or as pull-back; power- ed or automatic. Advantages: no end gates required, and car bodies us be made tight; as many as 18 cars can be dumped at once. Pow

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ACTUATED TYPE, Fig 3*c*, is a drum, of a
and longitudinal members. In addition

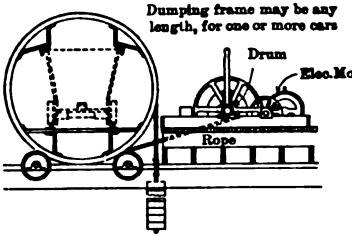


Fig 3*c*. Power-actuated Rotary Dump

uncouple cars before dumping. Power is
and friction drive. Breakage of coal is
of drum where coal drops. AUTOMATIC T

	Diam
No 1 Mine	13'0"
Northfield Mine	13'4"
Reserve Mine	14'10"

Fig 3*d*. Automati

in Alabama and the Northwest. It has 3
for 2 or 3 cars end to end. Weight of loa
to overbalance empty cars, and cause rota

2. Shaker-scr

Shaker screens, built in several design
ing round, square or oblong holes. Thi
gauge. Weight after being perforated is
4 to 5¢ per lb, varying with base price f
of holes.

ish shaker screen (Fig 4) (Wilmot Eng'g Co, Hazelton, Pa) is cheap, light and . The wooden hangers *a* are bolted rigidly to overhead timbers, giving spring-tion to screen, which keeps perforations clear of coal. Driven by eccentrics *c*, flexible wooden rods *b*, bolted rigidly to screen frame and eccentric. Having no s to get loose, the screen runs smoothly and quietly. A 4-deck screen, as in Fig ride by 15 ft long, has capacity of 2 000 tons per 9 hr, making 4 sizes. Weight, c, 4 000 lb; cost (May, 1914) $173, f o b Hazelton.

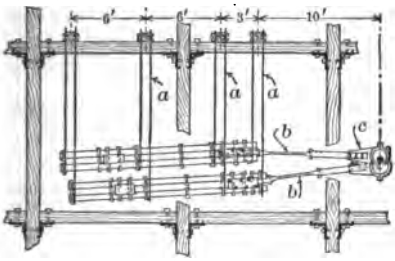


Fig 4. Parrish Shaker Screen

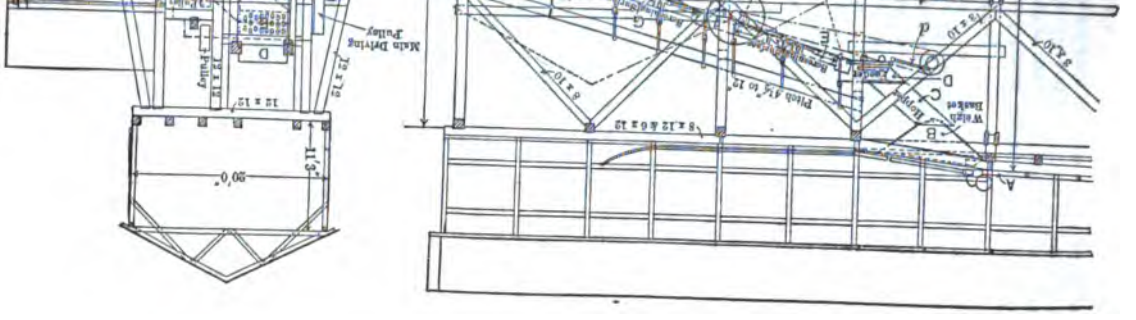
eral design. Fig 5 shows same frame as Fig 1, but with shaker screens ner differences in design. From crossover dump *A*, the coal drops into basket *B*; thence into hopper *C*, kept nearly full to minimize breakage . Below the hopper is a reciprocating feeder *D*, the feed plate of which ated by connecting rod *d*, from driving shaft. About a wheelbarrow of dropped on screen *E* at each stroke of feeder, but this, with number of , may be varied. Screen *E* is 6 by 15.5 ft, with staggered, oblong holes, 2 in (sometimes 2-in round holes). The screen plates are stepped, to more efficiently the nut and slack, which drop into bin *F*. Oversize d lump (sometimes called round, lump and block) go to screen *G*, hav-r 4-in round holes. Egg through *G* drops on picking and loading table ch is a pan conveyer mounted on a movable boom (3). This is pivoted at 5, end elev), so that loading end may be lowered close to bottom of car, reatly reducing breakage. Slate pickers may stand on a platform along-boom (4). Oversize lump from *G* goes to a similar loading boom *I*. boom usually has its own motor *J*, but may be connected with main (20 hp) by belting and clutches.

retarders (3). Three of these, built by Fairmont Mining Mach'y Co, nt, West Va, complete the equipment. They consist of a drum, with rweight and brake. A wire rope from drum passes to the R R car to be the car brakes being slacked. The apparatus is controlled from any bout the tipple, by a small rope led around guide pulleys. Pulling this fts a weighted lever attached to brake. This releases the drum, and al-ar to run by gravity to desired point, where it is spotted by letting the drop, thus braking the drum. The loaded car runs out by gravity, and er rope is detached and rewound on drum by the counterweight. Re-works best with 2% track grades.

mate for shaker-screen, wood-frame tipple, shown by Fig 5 (Jan, Competition in domestic market compels installations almost as com-is this in West Va, eastern Ky, Ill, and Jellico field of Tenn, but they hardly be built in Penn. The estimate assumes a plant within 50 miles bin, Ky.

Dry S

1880


Main Driving Pulley
Main Driving Pulley
Winch Barrel
Winch Barrel
Winch Barrel

vation (assuming av depth of 3 ft) 57 yd	@	$ .40		$22.80
dations, 53 yd (concrete, 1 : 3 : 5) 53 bbl cement	@	1.60	$84.80	
5 yd sand	@	.50	8.80	
4 yd stone	@	.50	14.70	
r, mixing and tamping			50.00	
r, building forms			50.00	
53 yd concrete @ $4.00)				$212.00
2 bd ft oak lumber, standard lengths	@	$20.00	$243.24	
3 bd ft oak, extra lengths, over 16 ft	@	24.00	479.83	
bd ft 3-in oak plank	@	20.00	86.00	
bd ft 1-in sheathing	@	16.00	27.20	
				$806.27
sq ft corrugated iron siding	@	$3.50 per sq		147.00
ndow sashes, 6-light, 10 by 12	@	.56		13.44
ndow frames	@	1.10		13.20
oat spikes 3½ by 8 in	@	2.15		3.22
olts ¾ in, average length 20 in	@	3.00		4.50
s 40d nails	@	1.75		7.00
s 20d nails	@	1.75		3.50
s 10d nails	@	1.85		3.70
8d nails	@	1.85		1.85
r, erecting 38 155 ft lumber	@	10.00		381.55
r, erecting roofing and siding				50.00
l cost framing without dump or machinery				$1 700.13
illips crossover dump for 42-in gauge, 2-ton cars, f o b Pittsburg				$250.00
ght on dump, 3 500 lb	@	$ .88		30.80
le and basket, complete, f o b Columbus, O				375.00
ght on scale and basket, 3 698 lb	@	.41½		19.67
ciprocating feeder, f o b Philadelphia				225.00
ght on feeder, 2 000 lb	@	.88		17.60
ir of shaker screens, complete, f o b Columbus, O				640.00
ght on screens, 8 000 lb	@	.41½		47.20
ading booms, complete with motor, f o b Phila				3 600.00
ght on booms, 7 000 lb	@	.88		60.60
ain motor, 20 hp, delivered				300.00
ain driving shaft, 2 pulleys, 4 bearings, bolts, 1 countershaft, 3 bearings, 1 pulley with feeder				300.00
o lb cast brackets, bolts, etc				20.00
airmont car retarders, f o b Fairmont				450.00
ight on retarders, 5 100 lb	@	.92 cwt		46.92
ight on shafting, pulleys, etc, 5 718 lb	@	.41½		32.73
				$6 415.52

cost, complete for operation, except tracks and grading, $8 115.65. Capacity of plant, 2 000 tons per day maximum. At an average of 1 000 tons, cost per ton eration would be 2.49¢.

aker-screen steel tipple (Fig 6); a plant built in winter of 1910–11, at roy mine, McKell Coal and Coke Co, by Fairbanks, Morse & Co, Chicago ription by Robert Elder).

pacity, 200 tons run-of-mine coal per hour. Cars holding 1.5 ton dump into 4-ton r *A*, over a Phillips dump (Art 1). From *A* coal goes to pan conveyer *C*, by a anks-Morse rake loader *B*. *C* has 3/16-in steel pans, 38 in wide, attached to 2 ds of steel roller chain. *C* delivers to steel shaking screens *D* and *E*, 5 ft wide by long. Lump coal passes over 3.5 by 5.5-in holes, egg over 1.5 by 2.5-in, and nut 0.75 by 1.5-in. Lump is loaded to cars on track 1; egg, track 2; nut, track 3. is collected by chutes at one point, and elevated by *F* (Fig 6, side elev) to steel holding 150 tons; thence to cars on tracks 2 and 3. Design permits loading run-

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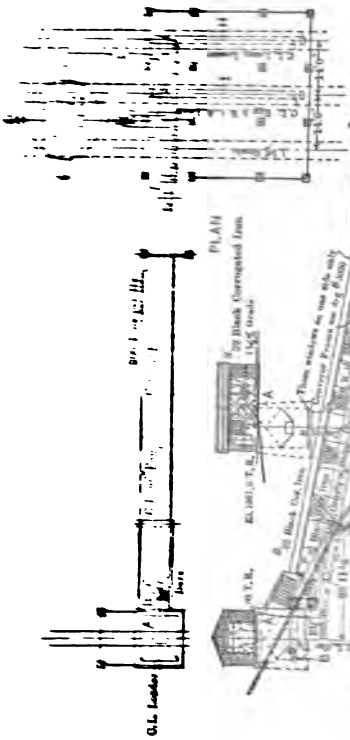
finished. Lump and
ft. of boom is
ft. The adjustable
every end is so adju
A 250-volt el
for stack eleva


PLAN
20 Black Corrugated Iron
Screen Plates on one side only
Screen Plates on both sides
Elevation of Tipple
Elevation of Tipple
Elevation of Tipple

%; **slack, 53%.** Coal handled is from the very soft Sewell seam, New River. **ng cost per day:**

nen:	1 dumper @ $2.50; 2 car spotters @ $2; 1 coupler @	$1.00	$7.50
m men:	1 tipple operator	2.50	
	4 men spotting R R cars and picking slate @ $2	8.00	
	1 weighmaster	2.75	13.25
	Total labor per working day		$20.75
	ated power per day	$1.00	
	ated repairs per day ($110 per yr of 220 days)	.50	
	eciation, on basis of $15 000, 30-yr life. (Insurance 0.00)	2.27	3.77
	Total cost per day		$24.52

age (1913) was about 110 000 tons, but for life of tipple average tonnage can prob- doubled, giving operating cost of 2.45¢ per ton.

aker-screen steel tipple, of 750 tons per hr capacity, was built in 1909 by Jeffrey o. Columbus, O, for Empire Coal and Coke Co, Landgraff, West Va. It was de- to handle the soft Pocahontas No 3 coal with minimum breakage. It has 3 load- ms, and a rotary screen to separate nut from slack. Coal is dumped from 3-ton a 130-ft conveyer. Cost complete, with 2 slack bins, $75 000 (3).

ker-screen concrete tipple (5) As yet, few are in operation. The ated Eng'g Co, of Somerset and Louisville, Ky, controls all rights for in- g reinforced-concrete shaker-screen frames. To April, 1914, this firm

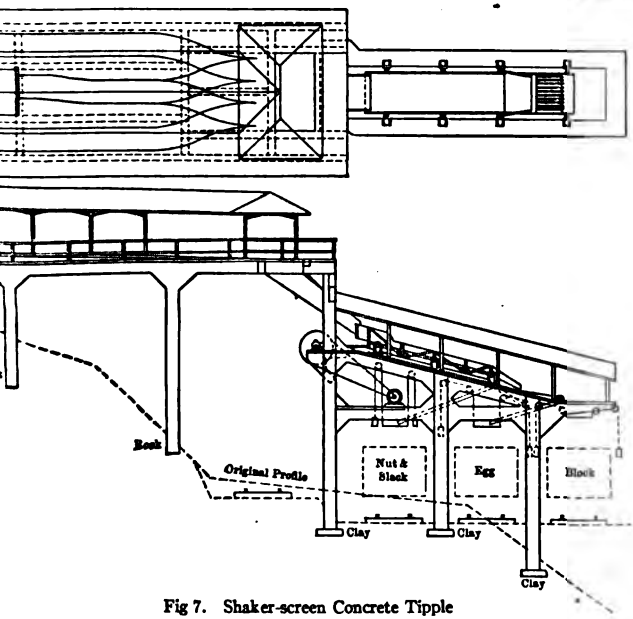


Fig 7. Shaker-screen Concrete Tipple

built and put in operation 4 concrete tipples. Fig 7 shows one of them, at rns, Ky, with rated capacity of 200 tons per hr. It is designed to ship 3 of coal. Another, at North Fork, near Hazard, Ky, completed Ma

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1914, which is almost a duplicate of Fig 6, grading, and mine-car tracks. Estimated c increase this figure to $10 200.

Two problems in successful operation of co O. G. Petersen (5): 1. Periodical strains incid sign of a mode of screen suspension, by which t by vertical ones. Details of solution are given tons per hr, passing over a screen 72 in wide, Petersen's design a power consumption of 6 hp screen suspension. The screens of North Fork full load. In the ordinary design there is an plates; in the others, an arc of 0.5-in vertical

"**Marcus**" **screen** is a new design, combin that of a conveyer. Screen is driven through a quick backward, motion; virtually pulling scr carried forward. It is set horizontally on whe screens, while in motion. A screen of 1 000 and make 4 sizes, would be 5 ft wide by about of tipple and screen, would cost at Chicago $4 ing head alone is stated to cost $1 250.

3. Comparison of above

Capacity: avg, 1 000 tons; m

	Gravity screen, wood frame
Pig No.	1
Construction cost	$3 236.63
Operating cost per day:	
Labor	$18.25
Power	.....
Repairs	0.50
Depreciation	0.74
Insurance	0.85
Operating cost per day	$20.34
Cost per ton	2.1¢
Years for depreciation	20

* The steel-frame, shaker-screen tipple is n them, one man could probably be dispensed wi

† Not including conveyer nor slack bin, whi

COAL WA

4. Introdu

Washing is the cheapest and most effi It is usually applied only to 2.5-in sizes picked. For closely associated or combin washing. Since impurities are nearly alw clean coal, washing is in effect concentrati lowing average specific gravities:

	Sp gr		Sp gr
.....	1.00	Slate.....	2.25 to 2.5
.....	1.25 to 1.5	Coal or slate with pyrites..	3.20 to 3.6
al.....	1.45 to 1.8	Pyrites.....	5.00 to 5.2

erard washer, a plunger jig, was introduced in England, 1851; in France, Penn, 1873 (14). It seems probable, however, that coal washing was y practiced. In all washers the coal is agitated with a plentiful f water.

5. Trough and Barrel Washers

h washer (Fig 8), probably the simplest, is a wooden trough *AA*, from ft long, 3 ft wide and 15 in deep. Raw coal enters at hopper *B*, water pipe *C*. Trough has sufficient grade to carry down coal and water, urities settling above the movable dams *D*, while the coal passes over n to screen *E*, where water drains off and washed coal goes to car *F*. is stopped when spaces be- ms are filled with refuse; dams except lowermost are nd refuse is run through to car *H*. The efficiency largely on care in conduct- ation (8).

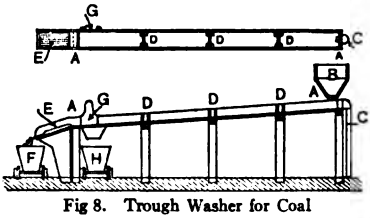

Fig 8. Trough Washer for Coal

washer is an improved trough made by attaching the dams to le chain. This forms a "flight ," which scrapes the refuse to d of trough, discharging into a car, while the coal is washed over the flights to a wer end (10).

kett washer (Fig 9) consists of a revolving barrel, set at an inclination size of coal washed, etc. A shallow worm, attached to inner surface of conveys the heavier "dirt" to upper end, where it is discharged, the coal being washed over the worm and delivered at lower end onto a drainer (15). This washer has probably not been used in U S.

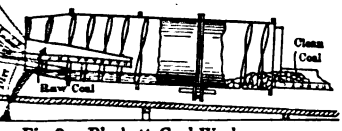

Fig 9. Blackett Coal Washer

Improved Scaife trough washer (9) (Scaife Foundry and Machine Co, Pittsburg, Pa) con- a trough and worm. The worm is a ribbon 0.5 in thick by 2 or 2.5 in onnected by spider arms to a shaft running the entire length, about 24 ft, ni-circular trough, which rocks like a cradle while worm revolves. Trough it pitch of about 1 in 12, depending on material. Coal and water enter at middle point of trough, the coal working over the rim of worm to end, while dirt is delivered by worm at upper end. Average speed, 40 rev m, and 40 double rockings of trough per min. Average capacity, 8 tons al per hr. Water, 3 000 gal per hr, which can be used again after settling. capacity, water, and pitch depend on material to be washed. Approx required, 24 ft long by 2.5 to 6 ft wide by 3 ft deep; approx weight, ; price, f o b Pittsburg, $500.

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6. Par

Stroh jig (Fig 10) was introduced 18; valve driving engine, which, as far as p ently of tank *abde*, to avoid leaks caused calked with oakum; posts 8 by 8 in, br

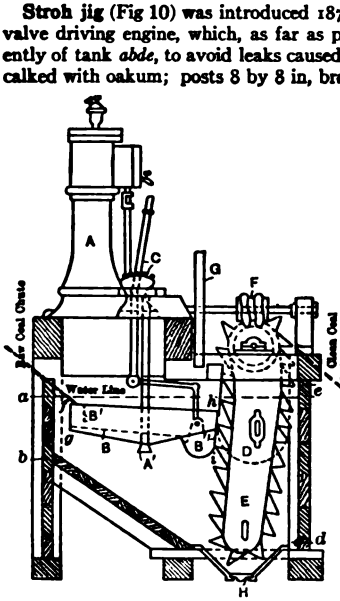

Fig 10. Stroh Coal Jig

use. The smaller the coal the higher the spee size of feed, percentage of refuse, and differen ranges from 5 to 12 tons raw coal per hr. T being added per 9-hr run, making total of 134 out the tank more than once a day. Space r Made by Mauch Chunk Iron Works; price,

Rotary jig is an improved pan jig b ation is similar to that of Stroh jig, exce has a rotary motion.

Raw coal fed at *A* is forced downward b over incline *C*. Refuse travels forward to f plates hinge at *D*, to open downward. This weighted at *D'* on lever *L*, and operated b Once fixed for the sp gr and size of feed, th When the weight of material is sufficient to o into tank below, where it joins the sludge w tom. The refuse is raised from tank by scra at *E* to belt conveyer *G*. The small diagra terial, as compared with other jigs. Norma character of feed. Capacity ranges betwee quired than for Stroh jig, conveyer *F* takin not be flushed more than once a week. Ta additional per day's run, amounts to 3 30 required, 14 by 5.5 by 8 ft; total weight, 8 5

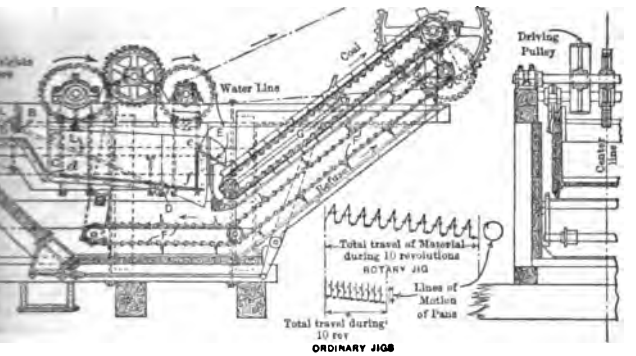

Fig 11. Mauch Chunk Rotary Coal Jig

b **Mauch** Chunk, Pa, $550 (Jan, 1914). Other makes of pan jigs are noted in
A successful FLOAT REGULATOR for the slate bed of coal jigs was brought out,
16, by Henry Reap, 318 S Main Ave, Scranton, Pa.

7. Plunger or Piston Jigs (See also Sec 28, Art 20)

ng these are the "New Century," and the machines of Cory & Co, Pitts-
nd American Concentrator Co, Springfield, O (Art 10).

ican jig is one of the newer
Fig 12). The V-shaped tank
el, unless wash water carries
h acid, when it is of wood.
gs are usually in pairs, each
2 plungers. Plungers *B*,
ther packing *C*, have alter-
strokes, keeping material on
f constantly agitated. Pure
es over lip *D*, refuse passing
automatic rotating valve *E*,
equired. Speed of *E* is varied
un of mechanism, according
centage of refuse. Tank is
through hutch gate *G*.
nal speed of plunger, 115 r p m.
ty of this No 600 jig, with 5 by

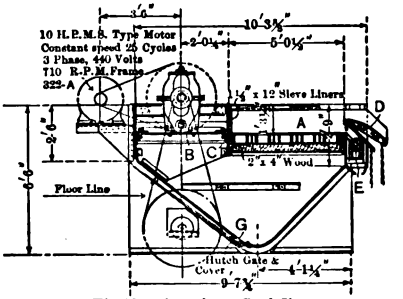

Fig 12. American Coal Jig

ves, 35 to 50 tons per hr. Water, about 15 000 gal per hr, which can be used
dly. Space required, about 10 by 12 ft; weight, 10 000 lb. Price, f o b Spring-
, $1 200 for a double jig, No 600 (Jan, 1914). Several other sizes are made.

8. Miscellaneous Coal-washing Machines

binson washer (11), an English design (R. Robinson, Bishop Auckland,
nd) is in high favor in Alabama and Tennessee. (Built in U S by Jeffrey
o, Columbus, O (Fig 13).

k is conical, with sliding gates *A* and *B* at bottom. Water is led by a pipe to ring
ve gate *A*), whence it passes through 0.5-in holes *J* into the tank. A vertical
e *D* carries stirring arms *EE* and *FF* and steel ring *G*, rotating about 18 r p m.
ntering through chute *H* is led down by *G*, and is agitated by the arms in the water
ntly flowing in through *J*. Pure coal discharges at *I*, refuse being drawn off
gate *A* into slate chamber, whence, by opening *B*, it goes to a car or bin.

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Capacity of No 2 machine shown, 40 to 55 t
2 500 to 3 000 gal per hr, which may be use

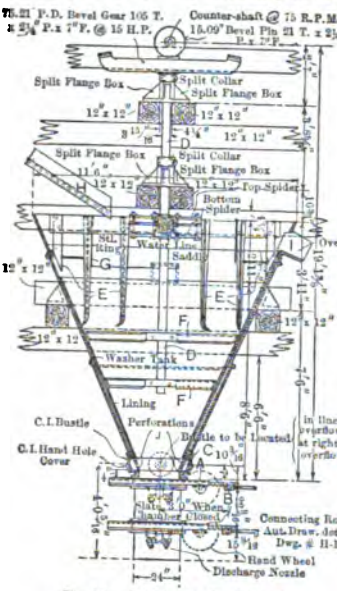

Fig 13. Robinson Coal Washer

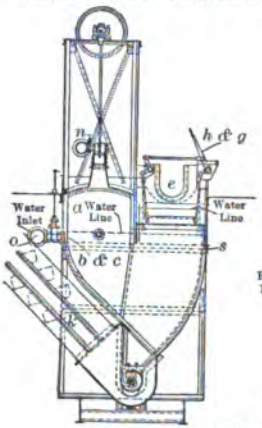

Fig 14. Baum C

Design of complete washing plants centage and nature of refuse, differences in

ion of suitable washer. A small machine (price $135) is made for this by the Pennsylvania Crusher Co, Philadelphia, and its use is also recommended for periodical analysis of washery refuse (*Coal Age*, July 20, 1912). For ons of complete washeries, see Bibliography, 8 to 18.

Washery Data (American Concentrator Co, Springfield, O)

ion	Rated hourly capacity tons†	Cost, washer only	Cost, per ton rated capacity	Construction	Size coal washed	No jigs	Style jigs	No Elmore dryers
Ala	250	$80 000	$320	Concrete and steel	4 in down	11	No 900 C	None
Pa	250	125 000	500	Steel	⅝ in	8	No 500, 2 compt	4
oal & o, Ala	100	67 700	677	Concrete and steel	"	2	No 500, 2 compt	2
rd o, Ala*	160	98 000	612	"	"	?	.....	.....
efield a*	160	70 000	440	"	?	?	.....	.....
l Coal e Co	70	14 000	200	Wood	3 in down	2	No 900 C	.....
ll Co, . Ill	125	40 000	320	Steel	2 in down	3	No 600	.....

uilt by American Concentrator Co. † For entire washery; for max and min pacities of individual jigs, see preceding paragraphs and next following table.

are used to dry the washed coal quickly. The Elmore centrifugal dryer (Amer noted above, has capacity of 60 to 70 tons per hr; price, f o b Springfield,

9. Dry Separators

eparators, without continuously moving parts, are common in anthra- preparation. At least four are on the market They have been in- to a very limited extent for bituminous

separator (Fig 15) is in use in Illinois, ield may widen (13). Having no moving o power is required. Sized feed enters d divides between chutes *B*, *C*, and *D*. friction, and centrifugal force combine ate refuse from the coal; the heavier avels on the narrow spirals near center ile the coal moves outward onto the wide On reaching the bottom, coal is delivered l refuse at *F*.

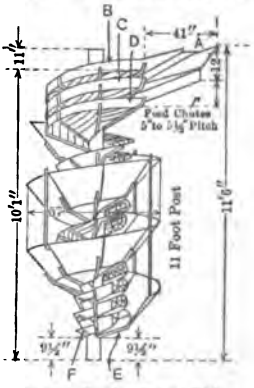

Fig 15. Pardee Spiral Picker

favorable conditions a further separation of al" from slate can be made. Since the operands on difference in friction of coal and refuse, eir sp gr to some extent, and as different coals hese respects, no useful comparison of results ade. The manufacturers make tests without

ty, 5 to 8 tons per hr; weight, 1 200 lb; space 8.5 by 11.5 ft. There are some slight changes in dimensions for different condi- lakers, Anthracite Separator Co; price, f o b Hazleton, Pa, $300 (Jan, 1914).

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10. Comparison of Coal Washer

Manufactured by	Name or type	Capacity	
		Min	Max
Pan Jigs			
Mauch Chunk Iron Works, Mauch Chunk, Pa	Stroh	5	12
Ditto	Rotary	10	17
Tamaqua Manufacturing Co, Tamaqua, Pa	Christ	...	30
Mount Carmel Iron Works, Mount Carmel, Pa	B & K	8	15
Plunger or Piston Jigs			
American Concentrator Co, Springfield, O	No 500	35	50
Ditto. "New Century Jigs"	No 600 No 900 C	35 40	50 60
Cory & Co, Pittsburg, Pa	Cory	40	60
Tub Washer			
Jeffrey Manufacturing Co, Columbus, O	Robinson	40	55
Ditto	"	55	Up
Trough Washer			
Scaife Foundry & Mchy Co, Pittsburgh, Pa	Scaife	...	8
Dry Separator			
Anthracite Separator Co, Hazleton, Pa	Spiral	5	8

Note. — Capacity and water vary with ma
over with all machines.

11. Coal

For coking, coal is usually crushed
crushes best, "run-of-mine" should be
there be too much impurity.

Pennsylvania hammer crusher (F
B are steel bars, ⅞ by 4.25 by 13.5 in,
rods are attached to 39-in diam disks, k
which makes 750 r p m. The botton
lowered by gear *F*, to adjust it close
quickly forced through screen, which us

Crusher size S-6 has 104 hammers; total
per hr; power, 125 h p; price f o b Philadelp
mers; weight, 37 500 lb; rated capacity, 40

e intermediate sizes, and larger sizes than S-15 could be built. Makers, Pennsyl-usher Co, Philadelphia, Pa.

ford coal cleaner (Fig 17) is often used in preparing coal for coking, here is appreciable difference in hardness of coal and impurities, also ce coal for the hammer crusher (Fig 16). It is a heavy cylindrical screen, with artly closed. mine coal en-fred end, and reen is about he weight be-force crushed ough the holes. der impurities crush nor pass screen, but y work to op-d, where they d up by wings o the spider, wn out at the

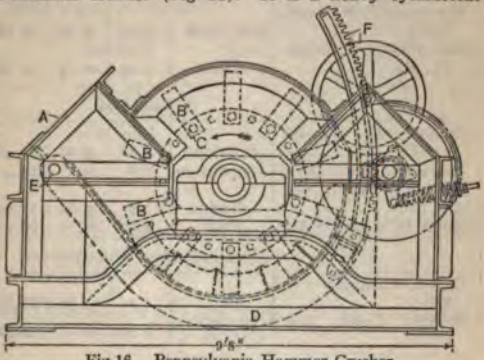

Fig 16. Pennsylvania Hammer Crusher

om 7 ft diam by 8 ft long to 12 by 17 ft; rated capacity, 35 to 360 tons per hr; ight, 17 000 to 67 000 lb; horse power, 20 to 45. A medium size, 9 by 11 ft,

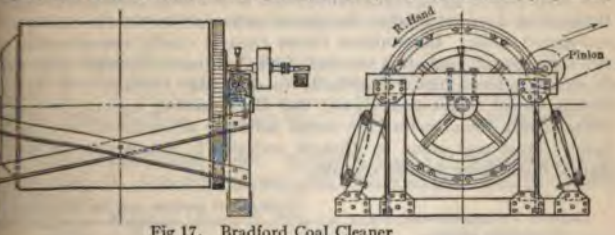

Fig 17. Bradford Coal Cleaner

apacity of 80 to 150 tons per hr; weight, 26 000 lb; price, f o b Philadelphia, or, 1914). Makers, Pennsylvania Crusher Co, Philadelphia, Pa.

12. Coal Analyses and Commercial Sizes

es. The use of coal for any particular purpose is controlled more by f supply than by quality. But there are certain generally recognized ; for example, New River, West Va, blacksmithing coal is trans-ross other coal-producing territory, to the far northwest of the U S. ses on following page, which may be called standards, are from U S ords. For standard method of coal analysis, see Sec 30, Art 15.

ercial sizes. There are no standard sizes for bituminous coal. oals for the same market and use have a general uniformity of size, ne used in one district may have a different signification in another. accurate tabulation of commercial sizes is practicable; and the fol-atements may not wholly accord with the views of other engineers. COKING is commonly crushed to ¾ or 0.25-in mesh. In some s in Connellsville field, LUMP COAL is taken off over 1.25-in bar screens,

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Typical Coal Analyses (U S

Location	Seam	Mois
Steam and Black		
Stonewall (New River), W Va	Pire Creek	3.0
Ehrenfield, Cambria Co, Pa	Miller	3.4
Coking		
Connellsville, Fayette Co, Pa	Pittsburg	2.4
Glamorgan, Wise Co, Va	Glamorgan	2.6
Ansted, Fayette Co, W Va	Eagle	1.9
Mulga, Jefferson Co, Ala	Pratt	2.0
Domestic Coal		
Gatliff, Campbell Co, Tenn	Regal	4.2
Straight Creek, Bell Co, Ky	Straight Creek	2.9

The domestic coals quoted are called standa
the analysis.

NUT COAL over 0.75-in, and COKING COAL
If the mine does not produce enough of la
are spaced wider. LOCOMOTIVE STEAM C
size called steam coal often consists of all
in shaker screens (equal practically to
POWER-HOUSE STEAM COAL is nut and s
shaker-screen round holes (2-in round ho
bars). Most coal going to Eastern seal
MESTIC COAL sizing is more complex.

Eastern Tennessee and Kentucky sizing	Equi
	Shaker (ro
Block	Over 3 to 4 in
Lump	Over 2.5 in
Round	Over 2 or 2.5 to 3 or 4 in
Nut and slack	Through 1.5,

In eastern Tennessee and Kentucky, which
block are often mixed, giving 40 to 50% L an
here corresponds nearly to egg in New River a

In Alabama, a product called "fancy lump
8-in holes); nut, through 5 and over 3.5-in ba
through 2-in.

In Illinois much egg, nut, pea and slack are
are usually as follows:

Lump, over bar screens.....
Egg, shaker and revolving screens, over
Nut, shaker and revolving screens, over
Pea, shaker and revolving screens, over
Slack, shaker and revolving screens, thr

Colorado and Oregon bar screens are largely used, making: lump, over 1.25 to space; nut, over 0.75 to 1-in; slack, through 1-in.
ch coal shipped on the Great Lakes is lump over 5-in and egg over 2-in bar screens, ro sizes frequently being mixed.

rcentages of sizes of bituminous coal shipped. No general statistics vailable. Some states report as follows:

	Alabama		Illinois			Indiana		Ohio
	1912	1913	1911	1912	1913	1912	1913	1912
np	12.8	10.6	39.1	37.6	33.7	....	....	69.4
	....	....	7.4	8.6	11.0	36.2	36.6	....
t	5.0	3.3	4.8	5.5	5.1	....	....	11.7
	....	....	20.4	19.4	22.2	19.9	19.3	....
ck	24.9	21 4	2.3	5.4	3.8	....	....	18.9
al screened	42.7	35.3	74.0	76.5	75.8	56.4	55.9	100.0
n-of-mine	57.3	64.7	26.0	23.5	24.2	43.9	44.1	....
al	100	100	100	100	100	100	100	100

st Virginia, Virginia, Kentucky, and Tennessee would doubtless show marked in- es in screened coal shipped, but tonnages by sizes are not reported.

13. Partial List of Machinery Manufacturers

ine car dumps. Jeffrey Mfg Co, Columbus, O. Link-Belt Engineering Co, Nice- Philadelphia, Pa. Phillips Mine and Mill Supply Co, Pittsburg, Pa.

ine scales. Fairbanks-Morse Co, Chicago, Ill. Standard Scale and Supply Co, burg, Pa.

avity and shaker screens. Bartlett & Snow Co, Cleveland, O. Link-Belt Engi- ng Co, Nicetown, Philadelphia, Pa. Jeffrey Mfg Co, Columbus, O. Phillips Mine Mill Supply Co, Pittsburg, Pa. Roberts & Shaeffer Co, Chicago, Ill. Savage, J. & Co, Knoxville, Tenn. Webster Mfg Co, Tiffen, O. Wilmot Engineering Co, eton, Pa.

ading booms and picking tables. Bartlett & Snow Co, Cleveland, O. Jeffrey Co, Columbus, O. Link-Belt Engineering Co, Nicetown, Philadelphia, Pa. Webster Co, Tiffen, O.

r retarders. Fairmont Mining Machine Co, Fairmont, W. Va.

el tipples. Fairbanks-Morse Co, Chicago, Ill. Link-Belt Engineering Co, Nice- , Philadelphia, Pa. Roberts & Shaeffer Co, Chicago, Ill. Wellman-Seaver-Morgan Cleveland, O.

rforated plates. Card Iron Works, Denver, Colo. Chicago Perforating Co, ago, Ill. Hendrick Mfg Co, Carbondale, Pa.

oncrete tipples. Associated Engineering Co, Somerset, Ky.

al washing machinery, pan jigs. Mauch Chunk Iron Works Co, Mauch nk Pa. Tamaqua Mfg Co, Tamaqua, Pa.

al washing machinery, plunger or piston jigs. American Concentrator Co, ngfield, O. Cory & Co, Pittsburg, Pa. Montgomery Washer Co, Birmingham, Ala. rts & Shaeffer Co, Chicago, Ill.

al washing machinery, other than jigs. Robinson washer, Jeffrey Mfg Co, mbus, O. Scaife trough washer, Scaife F'dry & Mch Co, Pittsburg, Pa.

yers for washed coal. American Concentrator Co, Springfield, O. Erskin Ramsay, ingham, Ala.

ty separators. Anthracite Separator Co, Hazleton, Pa.

l crushers for coking coal. Jeffrey Mfg Co, Columbus, O. Link-Belt Engine- Nicetown, Philadelphia, Pa. Pennsylvania Crusher Co, Nicetown, Philadelphi tts & Shaeffer Co, Chicago, Ill. Williams Patent Crusher and Pulv Co, Chi

14. Freight

Freight rates are quoted by districts, and ca
R R maps in connection with following tabulati

Tidewater Shipments, Carloads (

Via, to	Fr		
	Clear-field	Snowshoe	
Pennsylvania R R, to :			
Greenwich piers (Phila, Pa)	$1.35	$1.35	$
Allowances (Note 1)....	0.10	0.10	
Canton piers (Baltimore, Md)	1.35	1.35	
Allowances (Note 2)....	0.17	0.17	
South Amboy, N J.....	1.55	1.55	
Harsimus Cove, N J.....	1.60	1.60	
Allowances (Note 3)....	0.05	0.05	
(All above f o b vessels)			
	New Ri		
Chesapeake & Ohio Ry, to :			
Newport News, Va, piers ..	$1.40	{ Du tri	
	Pocahontas, T Clinch Val		
Norfolk & Western Ry, to:			
Lambert Point, Va, piers...	$1.40		
	For trans-shi		
	For deli		
	$1.50		

Note 1. Allowance for trans-shipment be
Note 2. Allowance for trans-shipment be
Note 3. Allowance for handling on New

	Pittsburg	C
Baltimore & Ohio R R, to :		
Baltimore, outside capes....	$1.60	
inside capes.....	1.43	
track.....	1.85	
Philadelphia, outside capes.	1.60	
inside capes ..	1.50	
track.....	1.85	
St. George, N Y,.....	1.80	
tidewater track	2.10	

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Lake Shipments, Carloads (40 000 lb) 2 000 lb per Ton

Via, to	Rate	From points on (approximate)
...sylvania R R, to:.....	$2.12	East Broad Top R R points
	1.62	Huntington & Broad Top Mtn R R
	1.62	Tyrone Div
	1.37	Part of Bellwood Div
	1.27	Parts of Bellwood and Cresson Divs
...eland, O.....	1.22	" " " " " "
...or, O.....	1.12	Kittanning Point, west to and including Johnstown and part Cresson Div
	0.93	Part Pittsburg Div (New Florence, etc)
...docks, Pa.....	0.90	Part Conemaugh Div (Indiana Br, etc)
...ote. These rates ap-	0.88	Southwest Br, Pittsburg Div
...only for trans-ship-	0.83	Part Conemaugh Div, Indiana Br, etc
...nt by vessel on Great	0.78	Apollo, Avonmore, Bradenville Brs, etc
...kes)		Parts Pittsb and Conem'gh Divs, near Pittsb
...peake & Ohio Ry*, to	$1.17	New River district
...do, O, for transship-	1.02	Kanawha, Kentucky and Big Sandy districts, f o b vessel
...nt on Great Lakes		
...lk & Western Ry*, to	$1.12	Pocahontas, Tug River & Clinch Valley dist
...usky docks, for trans-	0.97	Thacker dist, f o b dock ; 5¢ per ton for loading on vessel
...pment on Great Lakes		

	Ohio	Pittsburg	W Virginia	Cumb, Piedm't, Myersdale, Gauley
...O R R*, to Lake Ports:				
...ort.....Track	$1.00	$1.00	$1.15	$1.35
Lake cargo	.....	0.78	.....	1.12
...land.....Track	0.90	1.00	1.15	1.35
Lake cargo	.....	0.78	0.90	1.12
...n.....Track	0.90	1.00	1.15	1.35
Lake cargo	0.80	0.78	0.90	1.12
...usky.....Track	1.00	1.25	1.25	1.45
Lake cargo	0.80	0.78	0.90	1.12

...—Ohio rates are f o b vessels. For other districts, loading charge at Sandusky ...ain is 5¢ per ton. At Fairport and Cleveland docks, not owned by B & O R R, a ...s made for loading.)

Line Shipments, Carloads (40 000 lb) 2 000 lb per Ton

Via, to	From districts or rate groups			
	Ohio	Pittsburg	W Virginia	Cumb, Piedmont, Gauley, Myersdale
...more & Ohio R R*, to:				
...mbus, O.....	$0.65	$1.00	$0.90	$1.15
...go, Ill.....	1.65	1.90	1.90	2.05
...it, Mich.....	1.15	1.40	1.40	1.60
...napolis, Ind.....	1.30	1.55	1.55	1.65 Mysdl $1.70
...uis, Mo.....	2.10	2.35	2.35	2.50
...o, O.....	1.00	1.25	1.25	1.45

	L & E Ry	Cumberland valley dist	
	Hazzard, Ky	Pineville, Ky	Appalachia, Va
...ville & Nashville R R*, to:			
...ville, Ky.....	$0.95	$0.95	$1.05
...is, Mo.....	2.20	2.20	2.35
...go, Ill.....	1.90	1.90	2.05
...land, O.....	1.25	1.25	1.40
...ta, Ga.....	.....	1.45	.....
...ville, Tenn.....	.....	0.70	.....

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COKE

15. Nature and

Definition. Coke is the residue after of bituminous coal, out of contact with a (the "fixed carbon" of the coal), with ash gas. Color, silvery gray (from long-contin quick burning, at low heat); texture, vesicu 1912 (21, 22) 43 983 600 short tons, from 65 in 1913, 46 299 530 tons, from 69 239 190 t

Methods of coking. 1. Mounds or hea 2. Bee-hive and horizontal ovens. 3. Re

16. Bee-hive

Probably used in Europe since 1750. 2-oven plant was built in Fayette Co, P

Fig 18. Standard Bee-hive Oven Domes

bee-hive ovens in U S are now practically ent sizes still exist in many plants.

Standard oven for hand-drawing Pittsburg). Fig 18 shows design of dome

Fig 20. Section B

elev, 12.5-ft oven, solid lines showing doo machine drawing (24). Fig 20 is section 12.5-ft oven; door, 32 in wide (for hand

imate for one bee-hive oven (hand drawing), prices Jan, 1914, Connellsville, Compiled from figures from Harbison-Walker Refractories Co, Carnegie Steel Co, timate in *Coal Age*, Nov 22, 1913.)

5	cu yd excav'n (avg)	@	$ .50	$ 62.50
6	liners	@	19.00M	26.90
0	lime bond crowns	@	19.00M	66.50
0	bottom tile, 12 by 12 by 3 in	@	69.00M	10.35
1	set of arch and jambs	@	8.50 set	8.50
0	red brick for bottom of oven	@	8.00M	6.16
0	red brick for door	@	8.00M	5.28
1	trunnell head, 12-in opening	@	2.00	2.00
1	cast-iron door frame	@	5.00	5.00
0	ft 70-lb cross rail for larry track, 700 lb	@	28.50 ton	9.97
9	ft 70-lb rails for larry track, 655 lb	@	28.50 ton	9.33
9	ft 25-lb rails for power 242 lb	@	27.50 ton	3.33
0	lb metal ties, larry track	@	.02½	5.00
0	lb cast-iron 4-in bell pipe	@	22.00 ton	8.80
1	saddle for 4-in cast pipe, 1 in open	@	1.50	1.50
4	cu yd mortar wall	@	3.75	127.50
2	cu yd brickwork, front of oven	@	5.00	7.50
	labor building oven	@	30.00	30.00
3	tons fire clay	@	3.00	4.50
0	bbl lime	@	.75	37.50
7	tons sand	@	.90	6.30
1	coke-oven valve	@	1.00	1.00
1	pc 1-in gas pipe, 18 in long	@	.15	.15
	Total cost oven only			$445.57
	Loading tracks per oven as follows:			
7	R R ties, 8 by 8 in by 8 ft	@	$ .50	$ 3.50
9	ft 70-lb rail loading track	@	28.50 ton	9.33
3	R R spikes	@	.02½	.70
0	cu yd dry wall for yard	@	3.00	60.00
	Total for loading tracks			73.53
	Grand total per oven			$519.10

uded in this estimate is freight, as follows:

Brick, about	50 000 lb	@	$.75 ton	$18.75
Rails	2 252 lb	@	.10 cwt	2.25
Ties and cast pipes	1 000 lb	@	.10 cwt	1.00
Fire clay	1.5 ton	@	.75 ton	1.12
Wood ties	1 000 lb	@	.10 cwt	1.00
Total freight				24.12
Net cost of oven				$484.98

rting oven. A wood fire is gradually increased in intensity, adding lump during 2 to 4 days. Sudden heat may crack brickwork. Finally, a small prepared coal is dumped through trunnell head by larry (Fig 26), and front s bricked up, leaving air holes for draft. Product of this charge, called k jack," is discarded or used for starting other ovens. Usually, by fourth h day, a full charge for 48-hr burning may be made, and good coke results.

rge. There is little accurate knowledge of weight of oven charges. ls of coal produce tons of coke, and few superintendents know what a l of coal weighs. Tests in Connellsville field (14) show that average e for 48-hr coke is 5 tons. In S W Virginia, average charge for 48-hr ng seems to be 5.2 to 5.5 tons, for 12.5-ft ovens. General practice today rush coking coal to go through 0.25-in mesh.

eration. After heating oven, about ⅗ of height of door is closed with 6-in brick, and 5 tons of coal charged. To level the charge, a cross bar

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is set in notches in iron door frame, on whi
This is a 3 by 16-in iron bar, welded at rig
pipe, with a loop handle. Charge shoul
bricked up to about 1.5 in from top, and b
tight. The small opening is regulated by

In 48 hr (or nearer an avg of 44 hr) after
tight and trunnell head is closed wholly, or ir
day begins about 3 AM, opens the door, and i
circle at end, and perforated with small hol
hose from water-supply pipe, and the charge
From 1 000 to 1 200 gal is often used, but is

Hand drawing. The coke-puller uses
slicer) and a "beaver" (same as leveler, d
piece is about 5 by 20 in, and the handle
After drawing the coke, the puller loads it
it to R R car; or he throws it back on the
other men, who also use a fork, to sift out
14 tines, 18 in long, 1.25-in spaces.

Coking practice. Ovens are pulled 6
ovens are pulled each day, so that each is

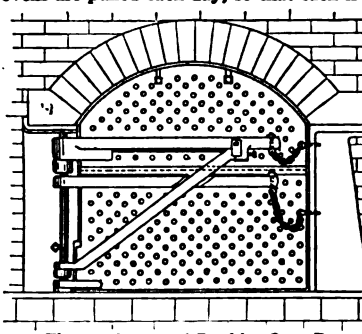

Fig 21. Improved Bee-hive Oven Door

sections are closed at once, thus conserving hea
applicable to other ovens, as well as bee-hives.

Tests on hand-drawn bee-hive oven

Tests in Connellsville

Time in oven, hr	Coal charged, lb	Ash made, lb	Fine coke, lb	Market coke, lb	Tota coke lb
44	10 400	84	209	6 922	7 13
70	12 400	103	264	8 060	8 32
70	11 430	94	282	7 640	7 92
44	9 420	75	249	6 260	6 50
Avg..	43 650	356	1 004	28 882	29 88

1.481 ton coal mak

Analyses of coal and coke in above tests

	Moist at 212° F, %	Volatile, %	Fixed carbon, %	Ash, %	Sulphur, %
Coal.....	1.25	31.27	59.79	7.16	0.53
Coke.....	0.63	1.37	85.99	11.12	0.89

Tests on Allegheny Mountain coal (8 days)

Coal charged, lb	Ash made, lb	Fine coke, lb	Market coke, lb	Total coke, lb	Per cent yield				Per cent lost
					Ash	Fine coke	Market coke	Total coke	
10 625	7 260	125	320	7 385	3.01	1.17	68.33	69.50	27.49
10 700	7 180	115	380	7 295	3.55	1.07	67.10	68.17	28.28
12 375	8 525	95	330	8 620	2.67	0.76	68.88	69.60	27.79
12 900	8 680	85	400	8 765	3.10	0.66	67.28	67.90	28.96
46 600	31 645	420	1 430	32 065	3.07	0.91	67.91	68.80	28.11

1.383 ton coal makes 1 ton coke

alyses of coal and coke in second test are not stated, but analyses at same plant tzin, Pa, "Lemon" or "E" seam) are:

	Moist at 212° F, %	Volatile matter, %	Fixed carbon, %	Ash, %	Sulphur, %	Phosphorus, %
l.....	1.20	23.68	68.77	5.73	0.62	0.0017
e.....	.....	1.20	89.25	9.55	1.46	0.0016

17. Machine Drawing (Bee-hive Ovens)

e hard physical labor of pulling coke has stimulated invention of drawing inery. Several successful machines are on the market (25); judging the sales, one of the best is the Covington. For using it, two changes are in the ovens: 1, the doors are wider (42 to 48 in); 2, no wharf (front urm) is necessary, though it is not objectionable.

22 shows design of a block of 50 ovens, at Glamorgan, Va, for machine ing with Covington apparatus. Cost complete, except for machine track some small details, $540 per oven (1910). Fifty ovens built previously $476 each. Higher cost of later block was due to soft ground, requiring er foundations (J. S. Cheyney).

vington coke-drawing machine (Fig 24) has two principal parts, *A* and *A* is a truck, with 4 wheels keyed on axles driven through bevel gears by a p electric motor. *B* is a separate truck, attached to *A* by a draw bar, carrying a horizontal and a boom conveyer, driven by a 15-h p motor. ruck *A* is a steel ram bar *a*, with a rack rail, mounted on a carriage on *A* 23).

e ram is swung in a plane parallel to oven bottom by a 2.5-h p motor. The ram has wings, which close up on entering oven and open on withdrawal by resistance e coke. In operation the duck-bill ram head is forced in under the coke, breaking , and is then pulled back by the 20-h p motor. The opened wings draw out the

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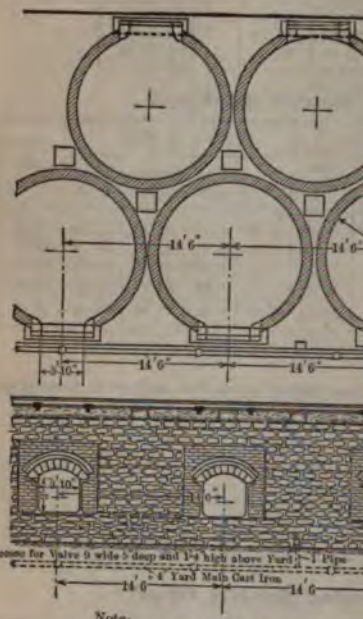

Note-
Oven Front to be Lime Mortar Ma

Fig 22. Machine-drawn Ov

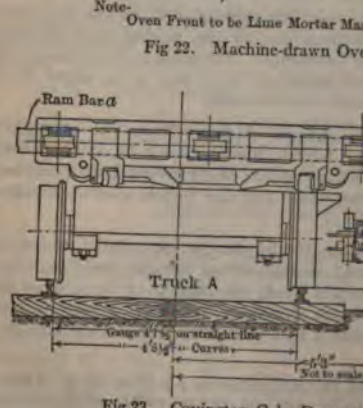

Fig 23. Covington Coke Drawer;

coke, which drops into horizontal apron conveyer *c*, composed of overlapping By this, the coke is delivered to conveyer *d*, on track *B*. *d* is of slats, spaced for ing the coke before it reaches the car. Conveyer *c* reaches across face of 2 ovens, o take the last of the charge when cleaned out by hand. The operator sits on left

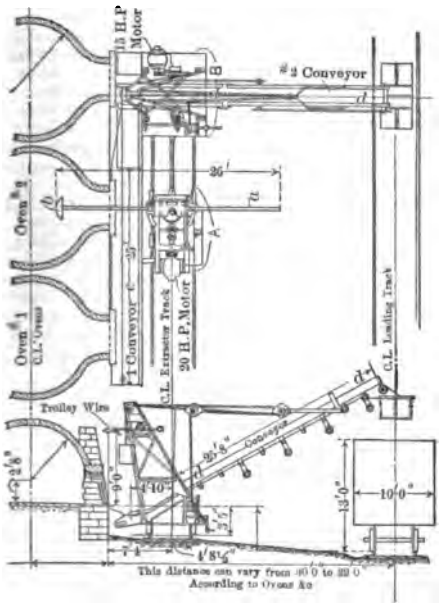

Fig 24. Covington Coke Drawer; General Views

bar, to watch the work in the oven, with controllers at hand. The machine runs odard track (4 ft 8.5 in); total weight, 60 000 to 70 000 lb; avg capacity, 50 ovens -hr day. Power is estimated at 5¢ per oven. Builders, Covington Machine Co, gton, Va; price, f o b works, $5 500 (Feb, 1914).

18. Hand versus Machine Drawing

vantages of machine drawing: 1, lower cost (see below); 2, fewer men, smaller investment in houses; 3, ovens drawn quickly, if enough cars, voiding burning up coke; 4, ovens open for drawing a shorter time, hence ving heat; 5, a charge as deep as can be burned off can be drawn.

advantages of machine drawing: 1, extra cost of oven maintenance , 1.9¢; machine, 7¢ per ton coke); 2, greater breakage of coke, but does not reduce price per ton shipped; 3, tendency of operators to hot coke, thus requiring outside watering; 4, cannot load readily into cars.

st of hand and machine drawing, for 50 ovens, assuming 4.2 tons coke ven; not including delays, which are generally much greater for hand than ine (part of data from catalogue of Covington Machine Works). De- ation on ovens not included.

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yses of above coals (Bureau of Mines, *Bull* 22, pp 226, 239).

	Moist	Volatile	Fixed carbon	Ash	Sulphur
Sewell seam	3.34	21.25	73.18	2.23	0.56
nes, Fire Cr					
.....	2.29	22.65	69.18	5.88	0.73

re low in volatiles (see analyses); (*b*) Weights of coal and coke osts (from payrolls) are on same basis.

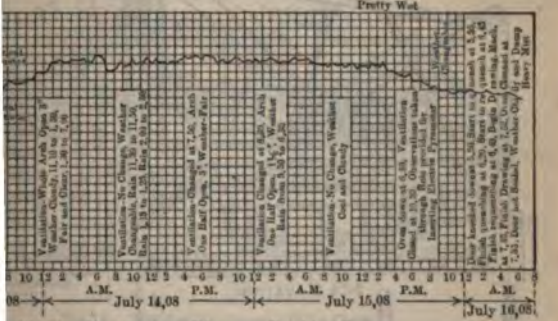

Fig 25. Temperature Diagram, Bee-hive Oven

iagram, machine-drawn charge of 150 bushels, is shown in Fig 25; ored, rather large cell structure, much smoky coke (25).

19. Larries and Levelers

ecial cars for hauling coal from bins to ovens. Fig 26 shows larry. Bottom slopes from all sides, to make shoveling un- e with funnel *A* is placed on both sides for block ovens (rows of ck). For charg- ed opposite cen- hute *A* dropped by turning hand are often hauled ower larry may aers without mo- ct ovens are gen- by traveling crane. larry shown, 7 tons; lb; motor, 36 hp; ips Mine & Mill tsburg; price, f o b, 4).

Fig 26. Electric-driven Larry

r (Fig 27) is a device of recent design (Covington Machine Co, for leveling bee-hive oven charges (25). It runs on larry track, trolley.

truck is a horizontally-rotating frame, carrying leveling shaft A. *e motor is geared to one* axle of truck; a similar motor operates

Hand drawing	
1 foreman	
1 watchman	
1 charger, 4¢ per oven	
1 dauber	
2 levelers, 12.5¢ per oven	
15 pullers, $1 per oven	
1 driver, cart, mule	
7 loaders, 35¢ per oven	
Repairs, 7.98¢ per oven 4.2 tons	
Deprec'n, crusher, bi trestle and larry, li of 20 yr, 2¢ per oven	
29 men ..... Total cost	
Cost operation per oven	
Cost per ton, at net yield 4.2 tons per oven	

Difference in favor of machin
Cost statements of Stone Gap
Yield per oven, 48-hr coke,
Yield per oven, 72-hr coke,
Yard cost, hand-drawn coke
Yield per oven, hand work,
At a Connellsville plant, machi
In cold weather larger charges are 500 to 1 000 lb greater t

Cost of hand and machi
River, West Va, seams. Tl

	Brooklyn	
	Sept	O
Charging labor	0.017	0.
Drawing labor	0.184	0.
Other labor	0.133	0.
Supplies	0.049	0.
Repairs	....	0.
Level'g & daubing	0.032	0.
Foreman	0.033	0.
Watering	....	..
Total costs	0.448	0.4
Coal charged, tons	2 605	3 06
Coke yield, tons	1 800	2 01
Coke yield, per cent	0.691	0.65

Average 1.495 tons coal to 1 ton

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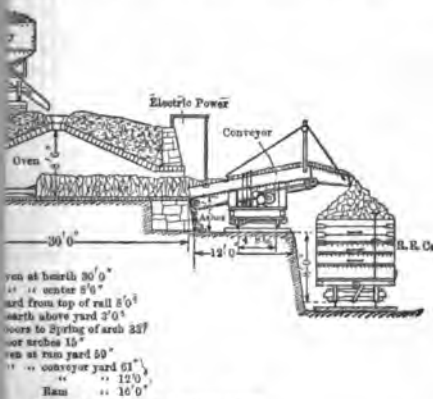
rated bee-hive. Yield from Connellsville coal, 75%. (M. J. Mitchell), not including depreciation, taxes

Mitchell oven. For description of an experimental plant (26). Small, vertical, tubular boilers were placed on



Mitchell Horizontal Coke Oven

s to be moved aside when charging. "Each unit (oven) 25 b h p." Coking time on 48-hr coke was reduced by and ash reduced. It should be stated that most of these from bee-hive oven waste-heat installations.

21. By-product Ovens

place of the wasteful bee-hive, coke manufacture becomes possibility of saving the by-products was known as early ial results came with the invention of Siemens' regener- with different width ovens for different coals, about 1883. literature of this subject is given in Bibliog (27 to 38).

of these ovens. The heat and gases from a coal fire in *A* where the illuminating gas is generated.

ovens, of a number of different designs, have all re- length *E* varies from 30 to 36 ft; width *C* is practically of charge *D*, from 4 ft 1 in to 9 ft. Fig 30 shows largest ese ovens are 4, 5, or 6-high, according to number of ole flue *F*. The oven shown is 6-high. Coal is charged off at *V*. Fuel gas for heating the oven enters through ntal flues in side walls, then through *F*, to regenerators f gases is heated, and finally to chimney *G*. Gases are regenerator to other about every half hour. Peep holes, of flues from end to end (31).

flue type (Fig 31). Its operation is similar to that of ence of design being that the flues are vertical (33, 34). product ovens, the two shown being the most prominent orm application of heat, and proper proportioning of ves, are essential in the design of all.

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Recovery and preparation of by-produ
are often installed at mines; but in U S to

Fig 29. Gas Re

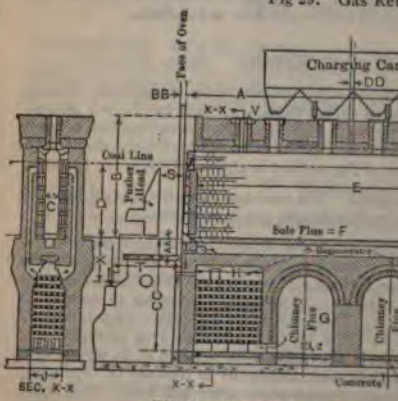

Fig 30. Semet-Solvay Regener

manufacturing centers. A brief description
30, when illuminating gas is wanted, outlet
oven. The first illuminating gas comes from o
and fuel gas is taken from other opening; the

duplicated. The chief products are ammoniacal liquor, tar and gas, g many derivatives.

ases are drawn through a liquor "SEAL," which gives a first washing. MAINS y" or "hydraulic." In the dry main, the seal is placed at point of collection for lant, and the troughs are kept free from carbonaceous matter by washing with iacal liquor or tar. In hydraulic mains, there is a seal at each oven or group of CONDENSERS are air or water-cooled, one or both. For air cooling there are a ric vertical cylinders, with 1-ft annular space through which gases pass and are by a current of cold air in inner cylinder. Water-cooled condensers often resemble al tubular boiler, with bottom and top chambers. Water is pumped up through es, gases passing down between them. Condensers remove a large part of the

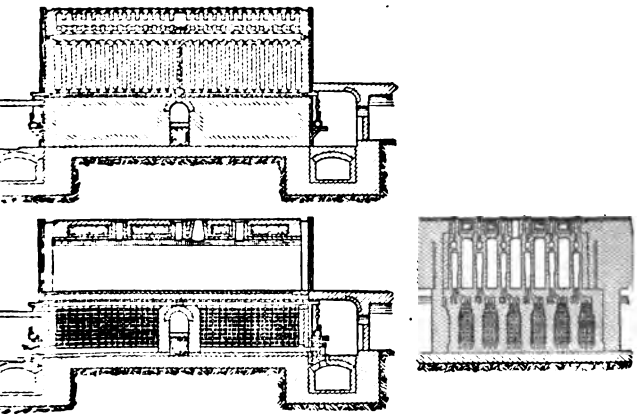

Fig 31. Kopper's By-product Coke Oven

ammonia, which are separated in settling tanks by difference in sp gr. WASHERS EXTRACTORS are commonly of "impact" type, consisting of a series of baffle set in a passageway. Half the plates are perforated, half plain. In passing the holes the tar adheres to next solid plate, and is drawn off through water settling tanks. Gases may subsequently be used for heating the ovens. SCRUB- ve traces of ammonia from gas for illuminating. A common type resembles ne fan (Sec 14) without the chimney, and kept half full of water. The vanes es, which keep the sides dripping with water to absorb the ammonia. Ex- may be placed before or after the scrubbers, to keep a steady pressure on the m ovens to receivers.

the above process, the gaseous by-products are separated into 3 chief prod- ee diagram below), besides coke dust and "breeze" (a size that will a 1 25-in mesh fork). All by-product systems are complicated and require ew wledge for efficient operation. The recovery of by-products may y varied. A system planned for one object may preclude the securing er: thus, if benzol be recovered, the illuminating gas produced is in- Above outline and parts of following diagram are compiled from e numbers 14, 15, 19, 21, 22 and 35.)

12. in U S, 14 767 543 short tons coal to by-product ovens produced 54 tons coke, or 25.72% of total coke production; in 1913, 17 095 369 al produced 12 714 700 tons coke. Yields: 1911, 75.1%; 1912, 75.26%; 74.4% of coal fed to ovens (21, 22).

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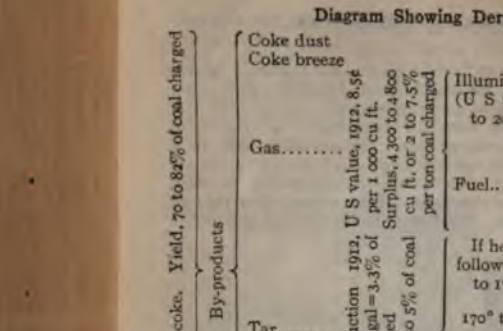
Diagram Showing Derivati


Product, metallurgical coke. Yield, 70 to 82% of coal charged
By-products
Coke dust
Coke breeze
Gas
Tar
Ammonia liquor or gas water
U S value, 1912, 8.5¢ per 1 000 cu ft. Surplus, 4 300 to 4 800 cu ft., or 2 to 7.5% per ton coal charged
U S production 1912, 94 306 583 gal = 3.3% of coal charged Gen avg, 2 to 5% of coal charged
App'x sp gr. 1.05. 3 to 5.5% of coal charged
Illuminati
(U S prod
to 20% o
Fuel...
If heated
following t
to 170°
170° to 23
230° to 27
270° to 36
Concentrat
Sulphate of
phuric ac
(U S produ

Uses of By-pro

Coke dust	Used in steel-mill soak Up to ¼-in size may b
Coke breeze	Domestic purposes
Gas	For illuminating and o For operation of gas en
Tar	Road making. Heavy ings. Waterproof p Saturation of felt for
Benzol	Motor fuel. To enric manufacture aniline lar substances. In m perfumery. Cleanin
Naphtha	
Naphthalene	
Carbolic acid	Disinfectant
Creosote	Medicinal Preservative for wood
Heavy oils	For lubrication
Anthracene	
Pitch	Paving. Water proofin
Concentrated ammonia	Medicinal. Cleaning. soda-ash. Manufactu
Ammonium sulphate	Converted to ammoniu manufacture, soap, al with acid phosphate,

In Great Britain considerable success ha
gas direct, in operating gas engines for driv
ton coal is about 10 000 cu ft, or 70 000 cu f
of this may be used for fuel or power. One pl
gas, of 500 B t u per cu ft, after extraction of

e tests are fully described in Bibliog (20, 39, 41, 42, 43).

ic coke. Many attempts have been made to lower sulphur content in coke during nufacture. Experiments have been conducted along the line of adding limestone charge; but a basic coke has not yet been successfully produced (44).

22. Comparison of Ovens (United States)

is of production, 100 000 tons per yr. Assumed coal analysis: Volatile, 33%; Fixed carb, 61%; Ash, 6%; Sulph, 0.95%

	Bee-hive	Mitchell horizontal	By-product
able days work per year	250	250	300
ber ovens required for output	200	150	65
of each oven	$500	$625	$2 000
of complete plant, including machinery, ises where necessary, etc	$160 000	$150 000	$325 000
per oven, complete plant	$800	$625	$3 000 to $5 000
Production and Yield			
coal charged per year	160 771	142 955	132 966
coke per year	100 000	100 000	100 000
uct per oven, tons per day (24 hr)	2.15	2.75	6.3
per ton charged	62.2%	70.0%	75.26%
Cost and Value			
at ovens, value per ton	90¢	90¢	$2.22 (incl frt)
value coal per year	$144 694	$128 659	$295 125
operating cost per ton coke, incl deprec	47¢	25¢	70¢
cost 100 000 tons coke	$191 694	$153 695	$365 185
est'd cost per ton coke at ovens	$1.92	$1.54	$3.65
ht per ton coke to Lake points	$1.95	$1.95	.....
By-products from 100 000 Tons Coke			
oke breeze (value at ovens, $1 per ton)			3 324 ton
urplus gas 40% (value at ovens, 8.5¢ per 1 000 cu ft)			531 864 000 cu ft
ar (value at ovens, 2.8¢ per gal)			850 982 gal
mmonia liq (value concent'd, 9¢; raw, 3¢ gal)			2 393 388 lb
st'd cost recovering by-products per ton coke			10.2¢
Total Value of By-products			
oke breeze (25% of coal charged)			$3 324
urplus gas (4 000 cu ft per ton coal charged)			45 208
ar (6.4 gal per ton coal charged)			23 827
mmonium sulphate (18 lb per ton coal chg'd, @ 0.0377¢ lb)			90 231
Total value of by-products			$162 590
receipts per ton (U S Geol Surv, 1912)	$3.37	$3.37	$3.83

al rate; no through rate is quoted; most bee-hive oven coke is marketed else-han the Lakes.

23. Freight Rates, etc.

made at the mines is distributed over a wide territory to manufacturing centers. eight rates are here given.

Coke Freight Rates, 2 000 lb per Ton (unless otherwise noted)

	From Connellsville	From Latrobe	From Gallitzin
lvania R R: reenwich piers, Phila, Pa, and ton piers, Baltimore, Md	$2.15	$1.95	$1.75

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Coke Freight Rates, 2 000 lb per Ton

Norfolk & Western Ry:	
To Norfolk, Va	
Lambert Point, Va	
	Fr R
Chesapeake & Ohio Ry:	
To Newport News, foreign shipments	
Toledo, O, local rate	
No through rates are quoted to Lakes.	
Louisville & Nashville R R:	
To Louisville	
St Louis	
Chicago	
Cleveland	
Atlanta	
Knoxville	
All above rates per ton of 2 000 lb	

Partial List of Material and

Brick. Harbison-Walker Refractories Co, Highland Park, Ky

Coke-drawing machines for bee-hive ov Va. Wellman-Seaver-Morgan Co, Cleveland,

By-product oven operators. Semet-Solv Gas Co, New York City. H. Koppers Co, Ch

Larries. Phillips Mine and Mill Suppl Columbus, O. Link-Belt Engineering Co, N Car Wheel Co, Barnsville, O.

Acknowledgment by author for data, dra of this section:

American Concentrator Co (Guy H. Elm rator Co (F. H. Blatch), Hazleton, Pa; Conne Cheyney, J. S., Glamorgan, Va; Covington and Covington, Va; Fairbanks-Morse Co (Ro Refractories Co (G. G. Coolidge), Pittsburg, P Chunk Iron Works (A. E. James), Mauch Ch Co, Uniontown, Pa; Nichol, Thos, McKell Phillips Mine and Mill Supply Co, Pittsburg, sylvania Crusher Co (G. W. Bolton), Philadel velt), Syracuse, N Y; United Coke and Gas C L. Ware), Jellico, Tenn; and others.

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SECTI

MATHEMATICS A

BY

C. H. BU

ASSOCIATE PROFESSOR OF MECH

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ALGEBRA

1. Elementary Operations

ıbols of operation are the same in algebra as in arithmetic. The sym- addition is often omitted in arithmetic. Thus, $3\frac{1}{4}$ means $3+\frac{1}{4}$. If a l of operation is omitted in algebra, it is that of multiplication. Thus,

ans $2\times\frac{x}{y}$. $4>x^2$ means that 4 is greater than x^2. $3<ab$ means that s than $a\times b$.

ıbols of aggregation are the bar, | ; vinculum, ——— ; parenthesis, ( ); t, [ ]; and brace, { }. Thus, each of the expressions $\begin{matrix}a\\+c\end{matrix}\Big|$, $\overline{a+c}$,), $[a+c]$, $\{a+c\}$, means that $a+c$ is to be treated as a single number. xpression within a parenthesis is preceded by a plus sign, the parenthesis e removed. If an expression within a parenthesis is preceded by a minus be parenthesis may be removed if the sign of every term within it be d. Thus, $2+(a-b)=2+a-b$; and $2-(a-b)=2-a+b$. If heses occur within parentheses, these may be removed, in succession, by ing first the innermost parenthesis; next, the innermost of all that re- nd so on:

$$\big[6+c-\{d+3\}\big]=a-\big[6+c-d-3\big]=a-\big[3+c-d\big]=a-3-c+d$$

lition. To add like terms, perform the indicated arithmetical operations numerical coefficients; this gives the coeff of the answer. Thus,

$$-7a+6a-2a+a=-2a$$

d terms which are not all like, combine the like terms and write down the , each preceded by its proper sign. Thus,

$$-2ax+4z+5ax-3z-c=3ax+z-c$$

omials are added by adding their respective terms.

s, to add $m^5-3m^4n-6m^3n^2$, $+m^3n^2+m^2n^3-5m^4n$ and $-n^5+2m^5+7m^4n$,

$$\begin{array}{r}
+\ m^5-3m^4n-6m^3n^2\\
\quad -5m^4n+\ m^3n^2+m^2n^3\\
\hline
+2m^5+7m^4n\qquad\qquad -n^5\\
\hline
+3m^5-\ m^4n-5m^3n^2+m^2n^3-n^5
\end{array}$$

btraction. One polynomial may be subtracted from another by changing gn of each term of the subtrahend and adding.

s, $(a^3x^2+2a^2x^3-4ax^4)-(a^5+4a^3x^2-3a^2x^3-4ax^4)=$
$a^3x^2+2a^2x^3-4ax^4-a^5-4a^3x^2+3a^2x^3+4ax^4=-a^5-3a^3x^2+5a^2x^3$

ltiplication. To find the product of monomials, annex the literal factors product of the numerical factors. Thus, $4a\times 2b\times 3z=24abz$. Like produce plus; unlike signs produce minus.

s: $4a\times 3c=+12ac$; $4a\times(-3c)=-12ac$; $-4a\times(-3c)=+12ac$; $a^2\times a^3$ $=aa\times aaa=aaaaa=a^5$; $4a^2c\cdot ac^3=4aac\cdot accc=4aaa\cdot cccc=4a^3c^4$; $6ab^2y^3$ $\times(-5a^2y)=-60a^3b^2y^4$.

multiply two polynomials, multiply each term of one factor by each term other factor and add the partial products.

s, to multiply $(-a^3+2a^2b-b^3)$ by $(+4a^2+8ab)$, the operation may be arranged rried out thus:

$$\begin{array}{r}
-a^3+2a^2b-b^3\\
4a^2+8ab\\
\hline
-4a^5+8a^4b-4a^3b^3\\
\quad -8a^4b\qquad\qquad +16a^3b^2-8ab^4\\
\hline
-4a^5\qquad -4a^4b^3+16a^3b^2-8ab^4
\end{array}$$

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$$\begin{array}{rr} 4\,b^2 & +9 \\ \hline 16\,b^4 & -36\,b^2 \\ & +36\,b^2-81 \\ \hline 16\,b^4 & -81 \end{array}$$

Division is the operation by which, if given, the other factor is determined. The given factor the DIVISOR; and the required is positive if dividend and divisor have and divisor have unlike signs.

$$m^3p^2x^4 + mp^2x^2 = \frac{m^3p^2x^4}{mp^2x^2} = m^2x^2; \qquad -5x$$

To divide a polynomial by a polynomial according to the ascending or descending keep this order throughout the operation.

Dividend $\Big|\,\dfrac{\text{Divisor}}{\text{Quotient}}$. Divide the first term of the divisor and write the result as tiply all terms of the divisor by the first product from the dividend. If there be a dend and proceed as before.

Thus: $(22\,a^4b^2 + 15\,b^4 + 3\,a^4 - 10\,a^3b - 22$

$$\begin{array}{r} 3\,a^4 - 10\,a^3b + 22\,a^2b^2 - 22\,ab^3 \\ \hline 3\,a^4 - 6\,a^3b + 9\,a^2b^2 \\ \hline -4\,a^3b + 13\,a^2b^2 - 22\,ab^3 \\ -4\,a^3b + 8\,a^2b^2 - 12\,ab^3 \\ \hline 5\,a^2b^2 - 10\,ab^3 \\ 5\,a^2b^2 - 10\,ab^3 \end{array}$$

The quotient is $(3\,a^2 - 4\,ab + 5\,b^2)$.

b^4. This type can sometimes be reduced to type 5 by adding multiple of a^2b^2. Thus:

$$x^2 + 4y^4 = (9x^4 + 12x^2y^2 + 4y^4) - 9x^2y^2$$
$$= (3x^2 + 2y^2)^2 - 9x^2y^2$$
$$= (3x^2 + 2y^2 + 3xy)(3x^2 + 2y^2 - 3xy)$$

$(a+b)(a^{n-1} - a^{n-2}b + a^{n-3}b^2 \ldots + b^{n-1})$
$(a-b)(a^{n-1} + a^{n-2}b + a^{n-3}b^2 \ldots + b^{n-1})$
} if n is odd.

$^n - b^n$) becomes type 5. In all other cases, if n is a multiple the special types,

$$a^2 + b^2 = (a+b)(a^2 - ab + b^2)$$
$$a^3 - b^3 = (a-b)(a^2 + ab + b^2)$$

$2ab + 2ac + 2bc = (a+b+c)^2$

2. Powers, Roots and Radicals

tween exponential and radical notation is expressed by the $\sqrt[b]{x})^a = x^{\frac{a}{b}}$, in which a and b are any numbers and b is not zero. al exponent, the numerator indicates the power to which the ised and the denominator gives the index of the root which is The fundamental laws of exponents are as follows:

a^n
$= a^{2n}$
$= -a^{2n+1}$
$^{n+n}$
$= a^{m-n}$

6. $a^m b^m = (ab)^m$
7. $a^m \div b^m = (a \div b)^m$
8. $1 \div a^m = (1 \div a)^m = a^{-m}$
9. $a^m = (1 \div a)^{-m} = 1 \div a^{-m}$
10. $(a^m)^n = a^{mn} = (a^n)^m$
11. $a^0 = 1$; $0^2 = 0$; 0^0 = indeterminate

ositive or negative; $\sqrt[2n+1]{+a}$ is positive; $\sqrt[2n+1]{-a}$ is negative.

$\sqrt[2n]{-a}$ is impossible or imaginary and the simplest one is $\sqrt{-1}$, ented by the symbol i. A complex quantity is partly real and y:

$i = \sqrt{-1}$; $i^2 = -1$; $i^3 = -i$; $i^4 = +1$; $\frac{1}{i} = -i$;
$= i^m$; $i^{4n} = +1$; $i^{4n+1} = +i$; $i^{4n+2} = -1$; $i^{4n+3} = -i$.

$\sqrt[n]{a}\sqrt[n]{b}$
$= \sqrt[n]{a} \div \sqrt[n]{b}$

16. $\sqrt[m]{\sqrt[n]{a}} = \sqrt[mn]{a} = a^{\frac{1}{mn}} = \sqrt[n]{\sqrt[m]{a}}$
17. $\sqrt{a} + \sqrt{b} = \sqrt{a+b+2\sqrt{ab}}$

$\frac{1}{\sqrt{a}} = \frac{1}{a^{\frac{1}{m}}} = a^{-\frac{1}{m}}$

g approximations are often convenient and are sufficiently accu- are small compared with 1, and if b is small compared with a:

$1 + 2x$
$= 1 + \frac{x}{2}$
$-x$
$+x$
$+y) = 1 + x + y$

23. $(1+x)(1-y) = 1 + x - y$
24. $(1+x)^2(1-2y)^2 = 1 + 2x - 4y$
25. $\sqrt{ab} = (a+b) \div 2$
26. $\sqrt{a^2 \pm b} = a \pm \frac{b}{2a}$
27. $\sqrt{a^3 \pm b} = a \pm \frac{b}{3a^2}$

$= 0.960a + 0.398b$. This is within 4% of the true value if $a > b$. A ion is $\sqrt{a^2 + b^2} = 0.9938a + 0.0703b + 0.3567\frac{b^2}{a}$.

$+ c^2} = 0.939a + 0.389b + 0.297c$. This is within 6% of the true value or instance, for the numbers 43, 42 and 41, the error $< 5.2\%$.

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8. Binomi

Binomial theorem is used to expa
follows:

$$(a \pm b)^n = a^n \pm na^{(n-1)}b + \frac{n(n-1)}{1 \cdot 2} a^{(}$$

The series is finite if n is a positive
tional or negative; and it is convergent
is called factorial r and may be writte

called binomial coeffs, and the rth coef

The binomial $(a \pm b)^n = a^n (1 \pm x)^n$, in
It is often more convenient to expa
four following special forms can be use
negative or fractional value of n, if x l

$$(1 \pm x)^n = 1 \pm nx + \frac{n(n-1)}{1 \cdot 2} x^2 \pm \frac{n(n-1}{1 \cdot}$$

$$\frac{1}{1 \pm x} = (1 \pm x)^{-1} = 1 \mp x + x^2 \mp x^3 + x^4$$

$$\sqrt{1 \pm x} = (1 \pm x)^{\frac{1}{2}} = 1 \pm \frac{1}{2}x - \frac{1}{2 \cdot 4} x^2 \pm$$

$$\frac{1}{\sqrt{1 \pm x}} = (1 \pm x)^{-\frac{1}{2}} = 1 \mp \frac{1}{2}x + \frac{1 \cdot 3}{2 \cdot 4} x^2$$

Proportion $a : b :: c : d$ may be writ

Hence, $ad = bc$, $\frac{b}{a} = \frac{d}{c}$, $\frac{a}{c} = \frac{b}{d}$, $\frac{a+b}{b}$

4.

Series is a succession of numbers w
Any one of the numbers is called a TE
continued indefinitely is called an INF
end with some particular term is called

Arithmetical series or progressi
between any two adjacent terms is eq
two adjacent terms. The common d
formula for the nth term is, $l_n = a$

$S_n = \frac{n}{2}(a + l_n)$, in which a is the first t

the above equations, any one of the ter
three are given.

Geometrical series or progressi
obtained by multiplying the precedin
is the first term, r the constant multi
GEOMETRICAL MEAN between two numb
them and makes, with them, a geom
numbers, then $G = \sqrt{ab}$ is their geome
means between the numbers, the cons

$r^{(m+1)} = b + a$. The sum of n terms is

Special Series

1. $1+2+3+4+\cdots+(n-1)+n=n(n+1)\div 2$
2. $p+(p+1)+(p+2)+\cdots+(q-1)+q=(q+p)(q-p+1)\div 2$
3. $2+4+6+8+\cdots+(2n-2)+2n=n(n+1)$
4. $1+3+5+7+\cdots+(2n-3)+(2n-1)=n^2$
5. $1^2+2^2+3^2+4^2+\cdots+(n-1)^2+n^2=n(n+1)(2n+1)\div(1\cdot 2\cdot 3)$
6. $1^3+2^3+3^3+4^3+\cdots+(n-1)^3+n^3=[n(n+1)\div 2]^2$
7. $1^4+2^4+3^4+\cdots+(n-1)^4+n^4=n(n+1)(2n+1)(3n^2+3n-1)\div 30$

$$\left.\begin{aligned}\frac{1+2+3+4+5\cdots+n}{n^2}&=\frac{1}{2}\\ \frac{1+2^2+3^2+4^2+\cdots+n^2}{n^3}&=\frac{1}{3}\\ \frac{1+2^3+3^3+4^3+\cdots+n^3}{n^4}&=\frac{1}{4}\end{aligned}\right\}\text{ as } n \text{ approaches } \infty$$

$$e=\lim_{n=\infty}\left(1+\frac{1}{n}\right)^n=1+\frac{1}{1!}+\frac{1}{2!}+\frac{1}{3!}+\frac{1}{4!}+\cdots=2.71828$$

which e is the base of the Naperian system of logarithms.

$$a^x=1+\frac{x}{1}\log_e a+\frac{x^2}{2!}(\log_e a)^2+\frac{x^3}{3!}(\log_e a)^3+\cdots$$

$$e^x=1+\frac{x}{1}+\frac{x^2}{2!}+\frac{x^3}{3!}+\frac{x^4}{4!}+\cdots$$

$$\left.\begin{aligned}\log_e(1\pm x)&=\pm x-\frac{x^2}{2}\pm\frac{x^3}{3}+\frac{x^4}{4}\pm\frac{x^5}{5}\cdots\\ \log_e\left(\frac{1+x}{1-x}\right)&=2\left(x+\frac{x^3}{3}+\frac{x^5}{5}+\frac{x^7}{7}+\cdots\right)\end{aligned}\right\}\text{ if } -1<x<+1$$

$$\log_e\left(\frac{x+1}{x-1}\right)=2\left(\frac{1}{x}+\frac{1}{3x^3}+\frac{1}{5x^5}+\frac{1}{7x^7}+\cdots\right)\text{ if } -1>x, \text{ or } x>+1$$

$$\left.\begin{aligned}\sin x&=x-\frac{x^3}{3!}+\frac{x^5}{5!}-\frac{x^7}{7!}+\cdots\\ \cos x&=1-\frac{x^2}{2!}+\frac{x^4}{4!}-\frac{x^6}{6!}+\frac{x^8}{8!}\cdots\\ \tan x&=x+\frac{x^3}{3}+\frac{2x^5}{3\cdot 5}+\frac{17x^7}{3^2\cdot 5\cdot 7}+\frac{62x^9}{3^2\cdot 5\cdot 7\cdot 9}+\cdots\end{aligned}\right\}\text{ The angle } x \text{ is in RADIANS}$$

5. Logarithms

Logarithm is the power to which a given fixed number must be raised to uce another number. In $y=b^x$, b is the base, x the logarithm or power, y the number which is produced. For common, or Briggs, logarithms the is 10. The only other system in common use is the Naperian, natural yperbolic system, for which the base is 2.71828. The abbreviation *log* is the logarithm of a number to the base 10. The notation Nap log 6, or 6, means the logarithm of 6 to the base e; and log 6 means the logarithm of the base 10. Thus, log 100 = 2, because $100=10^2$; log 10 = 1, because $10=10^1$; log 1 = 0, because $1=10^0$; log 0.1 = −1, because $0.1=10^{-1}$. Any ber > 1 has a positive logarithm and any number < 1 has a negative ithm. Numbers which are not whole powers of 10 must result from ng 10 to some fractional or irrational power. The decimal part of such ers, or logarithms, may be obtained from a table of logarithms. Since 10 ger than e (2.71828), it takes a higher power of e to produce a given number it does of 10. This means that the logarithm of a number, to base e, is r than to base 10. Relation between the two systems of logarithms is:

$$\log_e n=\log_{10} n+\log_{10} e$$
$$\log_e n=\log_{10} n+0.4343=\log_{10} n\times 2.303$$

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6. Choice an

Permutations or choice. If one thing another thing can be done in b different in $a \times b$ different ways. If one thing ca ways, a third in c ways and a fourth in d these things will be $a \times b \times c \times d$. The n elements or things, taken all at a time, is

$$n\,(n-1)\,(n-2)\,(n-3)$$

Probability or chance. If an event ways, each way being equally probable, t happen in a ways is $a \div (a+b)$; the chanc sum of the chances is 1. Odds in favor happening to chance of failure. Odds in

7. Quadratic and C

Linear equations, or those of the first

of the unknown, and have but one root. T

Quadratic equations, or those of the power of the unknown and have two roots. $ax^2 + bx + c = 0$, the roots are $x = \left(-b \pm \sqrt{}\right.$ the roots are real; if $(b^2 - 4\,ac) < 0$, the a perfect square, the roots are rational; i but opposite in sign. Following is an Eu the roots of a quadratic. The equation sh

$$x^2 - 2\,px + q = 0, \text{ or } x$$

Case 1. p and q are both positive factors. On intersecting axes, ox and oy, not necessari unity and $OE = q$, both upwards, and $OF = 2\,p$

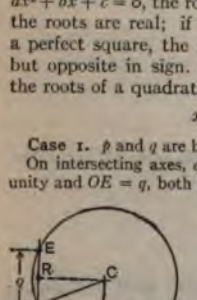

Fig 1

Case 4. p positive and q negative. Lay off $OE = q$ downwards, and proceed as before. On

Case 5. THE CIRCLE DOES NOT CUT OX, there part of the imaginary root can be found by drawin If T is the point of tangency, KT will be the re

Cubic equation, or one of the third deg unknown, may contain the second, first, an which may be imaginary. TRIGONOMETRIC equation is in the form $y^3 + by^2 + cy^2 + d =$ a cubic in which the second power does no

$x - (b + 3)$, expanding and collecting terms; it will then be in the form $px + q = 0$. Three cases must be distinguished:

$x^3 + px + q = 0$
$x^3 - px + q = 0$, and $(p^3 + 27) < (q^2 + 4)$
$x^3 - px + q = 0$, and $(p^3 + 27) > (q^2 + 4)$

} In these, p is necessarily positive and q is positive or negative.

rmulas for Case 1: $\tan A = \frac{2p}{3q}\sqrt{\frac{p}{3}}$, $\tan\frac{1}{2}B = \sqrt[3]{\tan\frac{1}{2}A}$;

roots are, $x_1 = -2\sqrt{\frac{p}{3}}\cot B$, $x_2 = \sqrt{\frac{p}{3}}\cot B + \operatorname{cosec} B\sqrt{p}\sqrt{-1}$, and

$\sqrt{\frac{p}{3}}\cot B - \operatorname{cosec} B\sqrt{p}\sqrt{-1}$

rmulas for Case 2: $\sin A = \frac{2p}{3q}\sqrt{\frac{p}{3}}$, $\tan\frac{1}{2}B = \sqrt[3]{\tan\frac{1}{2}A}$;

roots are, $x_1 = -2\sqrt{\frac{p}{3}}\operatorname{cosec} B$, $x_2 = \sqrt{\frac{p}{3}}\operatorname{cosec} B + \cot B\sqrt{p}\sqrt{-1}$, and

$\sqrt{\frac{p}{3}}\operatorname{cosec} B - \cot B\sqrt{p}\sqrt{-1}$

rmula for Case 3: $\sin A = \frac{3q}{2p}\sqrt{\frac{3}{p}}$;

oots are, $x_1 = 2\sqrt{\frac{p}{3}}\sin\frac{A}{3}$, $x_2 = 2\sqrt{\frac{p}{3}}\sin\left(60° - \frac{A}{3}\right)$

$x_3 = -2\sqrt{\frac{p}{3}}\sin\left(60° + \frac{A}{3}\right)$

8. Interest

mple interest. Let principal $= P$, yearly rate $= r\%$, amount of \$1 for ar $= R$, number of years $= n$ and amount of P for n years $= A$; then $1 + r$, the interest on P for n years $= Prn$, and $A = P(1 + nr)$. P is some- called the present worth of A. If d = any number of days, simple interest hat time is given by following formulas: At 4%, interest $= Pd \div 9\,000$; $Pd \div 7\,200$; 6%, $Pd \div 6\,000$; 8%, $Pd \div 4\,500$.

6% is a very common rate, it is worth recalling that interest for 1 month $\times 6) \div (12 \times 100) = P \div 200$; divide this by 30 to get the interest for 1 and multiply the last result by d to get the interest for d days. Proportional parts of these results may be taken for other percentages.

mple. The interest on \$5 284 for 30 days at 6% $= 5\,284 \div 200 =$ \$26.42; interest for 1 day $= 26.42 \div 30 =$ \$0.8807; interest for 12 days $= 12 \times 0.8807 =$ \$10.568. e 6% formula given above the interest for 12 days would be $(\$5\,284 \times 12) \div 6\,000 =$ 68. At 4%, the interest for the 12 days would be ⅔ \$10.57 = \$7.04. At 7%, the at would be \$10.57 + ⅙ 10.57 = \$12.33.

mpound interest. If, at the end of a period, interest is added to the ipal, then interest charged on the new principal, the second period's in- added to form a new principal and so on, the interest is said to be com- ded. With notation as before, $A = PR^n$, or $P = A \div R^n$.

mple. In 12 years at 6%, \$400 amounts to $A = 400\,(1.06)^{12} =$ \$804.88.

ent worth of \$5 000 due in 15 yr at 5% comp'd interest is $P = \frac{5\,000}{(1.05)^{15}} =$ \$2 405.09.

nterest is to be added to principal g times each year, $A = P\left(1 + \frac{r}{g}\right)^{gn}$

Example. The amount of $400 in 12 y
quarterly will be $A = 400\left(1 + \frac{0.06}{4}\right)^{4\times 12} =$
yearly, and instead of $688.00 by simple inte

Sinking fund is created to pay a de
penditure that must be made, at some
each time to be put at compound interes

$$A = S(R^n - 1) \div (R - 1) = S(R^n -$$

Amortization. $d\%$ = annual dividen
interest, and to furnish annual instalme
in a sinking fund at $s\%$ compound inte
end of n years:

$$d = [s + r(S^n - 1)] \div (S^n - 1), \quad \text{and}$$

in which S is the amount of $1 for one

Buying on instalments. Interest is
and amount of instalment deducted.

Q = amount of debt, P = amount of inst
$r\%$ = rate of interest, R = amount of $1 fo
of debt after payment of n instalments;
cancel debt, make $A_n = 0$; then

$$P = QR^n(R - 1) \div (R^n - 1), \quad \text{and} \quad n$$

GEOMETRY AND

9. Const

Lines. To divide a given line into tw
Let AB be the given line. With any
describe 2 arcs with A and B as centers.
section of the arcs, is a perpendicular bi
To divide a given line into any numb
Let AB be the given line and let the
line AC at any convenient angle with A.

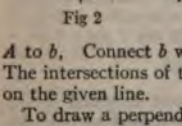

Fig 2 Fig 3

A to b, Connect b with B, and draw p
The intersections of these parallels with
on the given line.

To draw a perpendicular to a given li

Case 1. Point C is near the middle of t
scribe arcs of equal radius intersecting AB at
radius greater than Ca, describe arcs intersect

Case 2. Point C is near the extremity of
center, and radius OC, describe an arc inter
the arc at D. CD is the required perpendicu

To draw a perpendicular to a given li
the line.

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1. Point C is opposite, or nearly opposite, the middle of the line AB (Fig 6). as center, describe an arc intersecting AB at a and b. With a and b as centers, e radius as before, describe arcs intersecting at D. CD is the required perpen-

2. Point C is opposite, or nearly opposite, the extremity of line AB (Fig 7). C, draw any line intersecting AB at a. Divide line Ca into 2 equal parts, ab (method given above). With b as center, and radius bc, describe an arc inter- B at D. CD is required perpendicular.

es. To bisect a given angle.

1. Vertex B is accessible (Fig 8). Let ABC be the given angle. With B as nd a large radius, describe an arc intersecting AB and BC at a and c respectively.

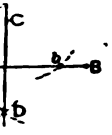

Fig 6

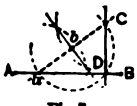

Fig 7

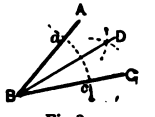

Fig 8

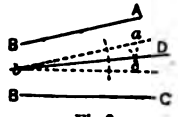

Fig 9

nd c as centers, describe arcs of equal radius intersecting at D. DB is the re- isector.

2. Vertex B is inaccessible (Fig 9). Let the given angle be the inclination lines AB and BC. Draw lines ab and bc parallel to the given lines, and at equal from them, intersecting at b. Let Db bisect angle abc (method given above). required bisector.

Fig 10

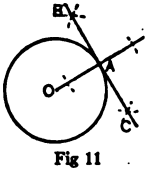

Fig 11

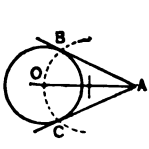

Fig 12

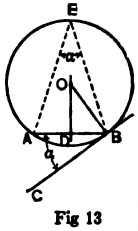

Fig 13

es. To draw a circle through 3 given points not in same straight line Let A, B, and C be the given points. Bisect the imaginary chords AB Draw bisecting lines and produce them to intersect at O. Point O of required circle.

aw a tangent to a circle from any given point.

1. Point A is on the circumference of circle O (Fig 11). Draw radius OA. A, perpendicular to OA, draw BAC, the required tangent.

2. Point A is not on circumference of circle O (Fig 12). Two tangents may Join O and A. With OA as diam, describe a circumference intersecting the le at B and C. BA and CA are the required tangents.

nstruct, upon a given chord, a segment of a circle in which a given angle inscribed (Fig 13). Let AB be the given line, and α the given angle. ct angle ABC equal to angle α. Bisect line AB by the perpendicular Draw a perpendicular to BC from point B. With O, the point of inter- of the perpendiculars, as center, and OB as radius, describe a circum- AEB is the required segment, and E may be located anywhere on AEB.

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To construct points on a circular arc
having given the chord and the rise (
and CD the given rise. Draw BD. D
$KE = KC$. Draw DE. Choose any di
AG, produced, meets BF at H, a point
on arc $ADHB$, use other values of DF

Parabola is a curve for which the
called DIRECTRIX, is equal to the focal
(Fig 15).

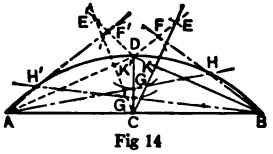

Fig 14

To inscribe a parabola within a rect
given rectangle. Divide CC' and BC,
ber of equal parts. Number the divis
way. Number divisions on $C'C$ and A'
divisions on BC and AB, draw paralle
CC' and AA', draw lines converging t
corresponding lines, such as 1 with 1
required parabola. This construction
parabola within a parallelogram. In th
the given parallelogram.

To draw a tangent to a parabola from

Case 1. Point P is on the curve (Fig
With N as center, and radius = 2 NP, desc

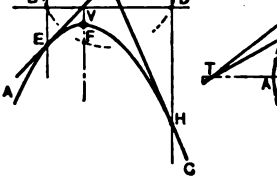

Fig 17 Fig

With P as center, and PF as radius, draw
Through B and D draw lines parallel to a
PE and PH are the required tangents.

Ellipse is a curve for which the sum
stant, that is, $FQ + QF' = AB$ = constan
To inscribe an ellipse within a rectang
length and width, respectively, of the gi

id OR as radii, describe circles. From O draw any radial line intersecting rcles at M and N. Through M draw a line parallel to OR, and through ine parallel to OB. These lines intersect at H, a point on the ellipse. it the construction to obtain other points. AB and RS are major and axes, respectively.

locate the FOCI of an ellipse, having given the axes (Fig 18). With R as , and radius = AO, describe arcs intersecting AB at F and F', the required

draw an ellipse, having given an axis and the foci (Fig 18). A cord or l, of length equal to the major axis, should be pinned or fixed, at its ends, foci F and F'. With a pencil inside the loop, keeping the cord taut so guide the pencil point, trace the outline of the ellipse (Q represents the point and length FQF' the cord).

draw a tangent to an ellipse from any given point.

e 1. The point P is on the curve (Fig 18). With O as center, and OB as radius, e a circle. Through P draw a line parallel to OR, intersecting the circle at K. th K draw a tangent to the circle, intersecting the major axis at T. PT is the d tangent.

e 2. Point P is the curve (Fig With P as center, dius PF', describe . With F as cen- d radius AB, de- an arc intersect- e first arc at M . Draw FM and intersecting the t E and G. PE are the required ts.

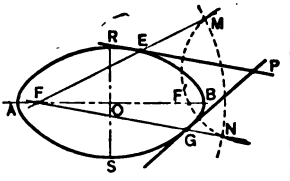

Fig 19

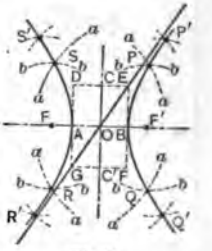

Fig 20

perbola is a for which the difference of the focal radii to any point is a constant; that 20), $FQ - F'Q = FQ' - F'Q' = AB$ = constant. ¶

onstruct an hyperbola by points, having given the foci and the constant difference focal radii. Let F and F' (Fig 20) be the foci, and transverse axis AOB the difference of the focal radii. $AF = F'B$. $AO = OB$. A and B are points on the required curve. With centers F and F', and any radius greater than FB or $F'A$, describe arcs a–a. With same centers, and radius equal to the difference between the first radius and transverse axis AOB, describe arcs b–b, intersecting arcs a–a at P, Q, R, and S, points on the required curve. Repeat the construction for additional points.

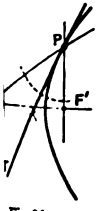

Fig 21

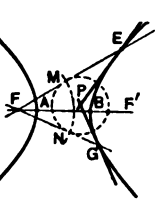

Fig 22

Make $BC = BC' = OF = OF'$, and construct the rectangle $DCEBFC'GAD$. The diagonals DF and EG, produced, are called TOTES. The hyperbola is tangent to its asymptotes at infinity. If the axes CO and given, the foci can be determined by making $OF = OF' = BC$.

raw a tangent to an hyperbola from any point.

e 1. Point P is on the curve (Fig 21). Draw lines connecting P with the foci. the angle $F'PF$. The bisecting line TP is the required tangent.

e 2. Point P is on the convex side of the hyperbola (Fig 22). With P as center dius PF', describe an arc. With F as center, and radius AB, describe an arc rting the first arc at M and N. Produce lines FM and FN to intersect the cu nd G. PE and PG are the required tangents.

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Reference units for angles. The
PERIGON, and equals 360 degrees of
One minute equals 60 seconds. In ci
One radian is the measure of an angle s
arc equal in length to the radius.

360 degrees of angle = 2π radians of
1 radian = $180 \div \pi$ degrees = 57.29578
Length of arc = $r\theta$; where r = radius
radians.

10. Lines

$A =$

Fig 23. Rectangle

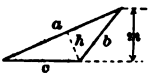


Fig 24. Triangle

Peri

If $s = \frac{(a+b+}{2}$

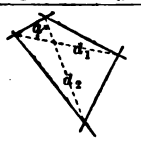


Fig 25. Quadrilateral

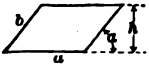


Fig 26. Parallelogram

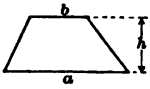


Fig 27. Trapezoid

Perimet

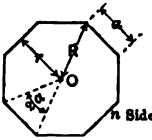


Fig 28. Regular Polygon

A_{OAE}

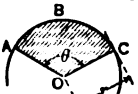


Fig 29. Circular Sector

Lines and Areas (*Continued*)

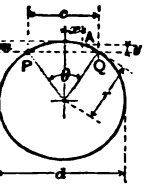


Fig 30. Circle

Circumference $= 2\pi r = \pi d$

Length of arc $PAQ = \theta$ (radians) $\times r$

$= \theta$ (degrees) $\times \frac{\pi}{180} \times r$

$= \theta$ (degrees) $\times 0.017453 \times r$

Relation of radius, chord, and rise:

$$\text{Radius, } r = \frac{m^2 + \frac{1}{4}c^2}{2m} = \frac{\frac{1}{2}c}{\sin \frac{1}{2}\theta}$$

$$\text{Chord, } c = 2\sqrt{2mr - m^2} = 2r \sin \tfrac{1}{2}\theta$$

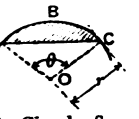
Rise or mid-ordinate,

$$m = r \mp \sqrt{r^2 - \frac{c^2}{4}} \quad \begin{bmatrix} \text{use } - \text{ if arc} \leqq 180° \\ \text{use } + \text{ if arc} \geqq 180° \end{bmatrix}$$

$$m = \tfrac{1}{2}c \times \tan \tfrac{1}{4}\theta = 2r \sin^2 \tfrac{1}{4}\theta = r + y - \sqrt{r^2 - x^2}$$

Side ordinate, $y = m - r + \sqrt{r^2 - x^2}$

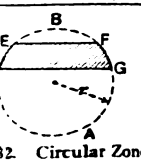
$A = \pi \times r^2$



1. Circular Segment

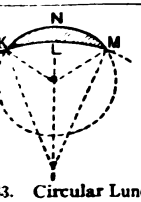
A_{ABCA} = Area of sector A_{OABCO} minus area of triangle A_{OAC}

$$= \frac{r^2}{2}\left(\frac{\pi\theta^{(\text{degrees})}}{180} - \sin\theta\right)$$



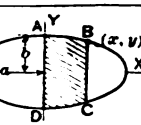
32. Circular Zone

A_{DEFGD} = Area of circle minus areas of segments $DAGD$ and $EBFE$



3. Circular Lune

A_{KLMNK} = Area of segment A_{KNMK} minus area of segment A_{KLMK}



ig 34. Ellipse

Circumference $= \pi(a + b) \times k$

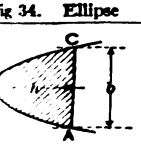
where $k = \left(1 + \frac{c^2}{4} + \frac{c^4}{64} + \frac{c^6}{256} + \cdots\right)$

and $c = (a - b) \div (a + b)$

Circumference $= \pi\sqrt{2(a^2 + b^2)}$ [approx]

$A = \pi \times a \times b$

$A_{ABCD} = (x \times y) + ab \sin^{-1}(x \div a)$



x 35. Parabola

If l = length of curve ABC and $n = h \div b$

$$l = b\left\{\tfrac{1}{2}\sqrt{1 + 16n^2} + \frac{1}{8n}\,\text{nap log} \times \left[\tfrac{1}{4}n + \sqrt{1 + 16n^2}\right]\right\}$$

$$l = b\left(1 + \frac{8n^2}{3} - \frac{32n^4}{5} + \cdots\right) \text{ [approx]}$$

$A_{ABC} = \frac{2}{3} \times b \times h$

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Lines and Area

Fig 36. Irregular Figures

Div
strips
an od
numb
the re
tween
by h_0.

$(n +$

The t

Trapezoidal Rule	$A =$
Simpson's Rule	$A =$
Durand's Rule	$A =$
Weddle's Rule	$A =$

If

11. Surfaces a

S = lateral or convex surface; T = total
A_1 = area of a right section as indicated in sk

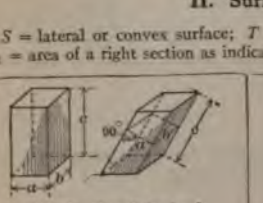


Fig 37. Parallelopiped

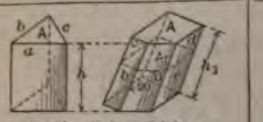
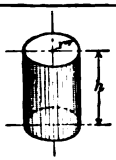


Fig 38. Prism, Right or Oblique, Regular or Irregular

Fig 39. Cylinder (Any Cross-section)

Surfaces and Volumes (*Continued*)



g 40. **Right Circular Cylinder**

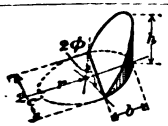
$$S = 2\pi rh$$
$$T = 2\pi r(r + h)$$
$$V = \pi r^2 h$$

Fig 41. **Frustum of Right Circular Cylinder**

$$S = \pi r(h_1 + h_2)$$
$$T = \pi r\left[h_1 + h_2 + r + \sqrt{r^2 + \left(\frac{h_1 - h_2}{2}\right)^2}\right]$$
$$V = \frac{\pi r^2}{2}(h_1 + h_2)$$

g 42. **Ungula or Wedge of Right Circular Cylinder [Semicircular Base]**

$$S = 2rh$$
$$V = \tfrac{2}{3}(r^2 h)$$



g 43. **Ungula or Wedge of Right Circular Cylinder se greater or less than semi-circle]**

$$S = \frac{2rh}{b}\left[c + (b - r) \times \frac{\phi^\circ \times \pi}{180^\circ}\right]$$
$$V = \frac{h}{3b}\left[c\,(3r^2 - c^2) + 3r^2(b - r)\,\frac{\phi^\circ \times \pi}{180^\circ}\right]$$
$$= \frac{hr^2}{b}\left[\sin\phi - \frac{\sin^3\phi}{3} - \frac{\phi^\circ \times \pi}{180^\circ}\cos\phi\right]$$

g 44. **Pyramid or Cone, ght or Oblique, Regular or Irregular**

$$V = \frac{Ah}{3}$$

45. **Right Regular Pyramid**

$$S = \text{perimeter of base } A \times \frac{k}{2}$$
$$V = \frac{Ah}{3}$$

46. **Right Circular Cone**

$$S = \pi r\sqrt{r^2 + h^2} = \pi rk$$
$$T = \pi r(r + k)$$
$$V = \frac{\pi r^2 h}{3}$$

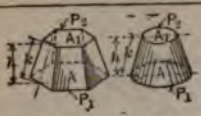


Fig 47. Frustum of Pyramid or Cone, Right, Regular, Parallel Ends

Fig 48. Wedge (Parallelogram Back)

Fig 49. Sphere

Fig 50. Spherical Sector

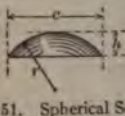


Fig 51. Spherical Segment

Fig 52. Spherical Zone

Fig 53. Circular Ring (Circular Cross-section)

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Surfaces and Volumes (*Continued*)

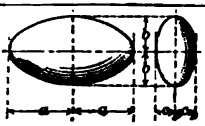


Fig 54. Ellipsoid

$V = \frac{4}{3} \times \pi \times a \times b \times c$

Prolate spheroid (revolution about major axis)

$V = \frac{4}{3}(\pi a b^2)$

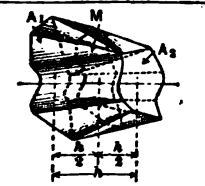
Oblate spheroid (revolution about minor axis)

$V = \frac{4}{3}(\pi b a^2)$

Fig 55. Paraboloid

$$S = \frac{2}{3} \times \pi \times \frac{b}{h^2}\left[\left(\frac{b^2}{16} + h^2\right)^{3/2} - \left(\frac{b}{4}\right)^3\right]$$

$$V = \frac{\pi}{8} \times b^2 \times h$$



56. Prismatoid [Irregular with bases in parallel planes]

If the planes connecting the bases are curved or warped they may be replaced by flat surfaces, best suited to the contour, formed by straight lines from one of the two parallel bases to the other. These planes may be parallelograms, triangles, or trapezoids.

Let A_1 = the area of one of the bases.

A_2 = the area of the other base.

M = the area of the cross-section parallel to, and midway between, the two bases.

h = the altitude of the prismatoid. (The perpendicular distance between the two parallel bases).

V = the volume of the solid.

smoidal Formula..... $V = \frac{1}{6} h [A_1 + 4M + A_2]$

[The area of the cross-section M should be computed from the dimensions of the polygon formed by the intersection, with the solid, of a plane parallel to the bases through the middle point of the altitude. In only exceptional cases is this cross-section the average of the two areas. However, the perimeter of this mid-polygon is half the sum of the basal perimeters]

Example. Find the volume of the irregular solid shown in Fig 57. *AEFGD* is the lower base; *HLMOP* is the upper base.

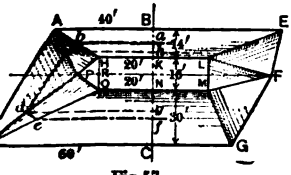


Fig 57

To simplify the problem, let the perpendicular plane *BKNC* divide the solid into two parts; find the volumes of the parts separately and sum them for total volume.

For the left portion of the solid: *ABCD* is the trapezoidal lower base, *HKNOP*, the pentagonal upper base. The given data is as follows:

Altitude, or perpendicular distance between bases = 50 ft

$AB = 40$ ft $\quad BK = 14$ ft $\quad KN = 16$ ft $\quad BC = 60$ ft

$HK = ON = 20$ ft $\quad DC = 60$ ft $\quad NC = 30$ ft $\quad HR = RO = PR = 8$ ft

imensions necessary to calculate area of cross-section M are:

40 ft, $dg = 44$ ft, $ch = 34$ ft, $ba = 30$ ft, $fg = 4$ ft, $gh = 30$ ft, $ha = 4$ ft

heck: Perimeter of lower base = 223.24 ft; perimeter of upper base = 78.62 ft; meter of mid-polygon = 150.93 ft;

$$\text{half sum of basal perimeters} = \frac{223.24 + 78.62}{2} = 150.93 \text{ ft}$$

$ABCD = A_1 = 3\,000$ sq ft; area $HKNOP = A_2 = 384$ sq ft; area $afedcb = M$ 466 sq ft; hence $V_{left} = \frac{50}{6}[3\,000 + 4(1\,466) + 384] = \frac{50}{6}[9\,248] = 77\,067$ cu ft.

he volume to the right of plane *BKNC* may be found in a similar manner.

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PLANE TRIG

12. Line Value

Let the radius of a circle (Fig 58) called a UNIT CIRCLE, and let A = angle

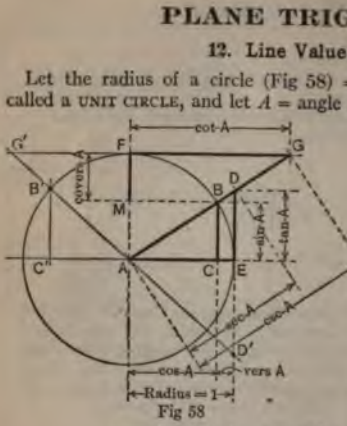

Fig 58

versed sine of A = $vers\ A$ =
coversed sine of A = $covers\ A$ =

If the radius AB of the circle (Fig 58 the center A from the initial horizontal p by the arc described by point B, and ANGLES are generated by counterclockw by clockwise turning. For any magnitud

$sin\ A$ = vertical projection of moving radiu
$cos\ A$ = horizontal projection of moving ra
$tan\ A$ = distance from E along a tangent to
$cot\ A$ = distance from F along a tangent to
For angle EAG': $sin\ A = B'C'$; $cos\ A =$

Algebraic signs of functions. Assu first quadrant to be positive, then:

(*a*) sines and tangents extending from the DOWNWARD, negative.
(*b*) cosines and cotangents extending from are positive; TOWARD THE LEFT, negative.

13. Functions of Righ

In any right-angled triangle ABC (Fig $AC = b$, and $BC = a$.

1. $sin\ A = \frac{a}{c} = cos\ B$
2. $cos\ A = \frac{b}{c} = sin\ B$
3. $tan\ A = \frac{a}{b} = cot\ B$

$\text{vers } A = \frac{c-b}{c} = \text{covers } B$

$\text{covers } A = \frac{c-a}{c} = \text{vers } B$

$C = 90° = A + B$

$a = c \sin A = b \tan A$

$a = c \cos B = b \cot B$

$a = \sqrt{c^2 - b^2} = \sqrt{(c+b)(c-b)}$

13. $b = c \cos A = a \cot A$
14. $b = c \sin B = a \tan B$
15. $b = \sqrt{c^2 - a^2} = \sqrt{(c+a)(c-a)}$
16. $c = \frac{a}{\sin A} = \frac{b}{\cos A}$
17. $c = \frac{a}{\cos B} = \frac{b}{\sin B}$
18. $c = \sqrt{a^2 + b^2}$
19. $\text{Area} = \frac{ab}{2} = \frac{a}{2}\sqrt{c^2 - a^2} = \frac{a^2}{2}\cot A = \frac{b^2}{2}\tan A = \frac{c^2}{2}\sin A \cos A$

nctions of complementary angles. Each function of an acute angle ual to the co-named function of the complementary angle, that is, $\sin A$ $s\ (90° - A)$; $\tan A = \cot (90° - A)$; $\sec A = \csc (90° - A)$; $\cos A = \sin$ $- A)$; $\cot A = \tan (90° - A)$; $\csc A = \sec (90° - A)$.

14. Formulas

$\sin A = \frac{1}{\csc A} = \pm\sqrt{1 - \cos^2 A} = \tan A \cos A$

$\sin A = 2 \sin \tfrac{1}{2} A \cos \tfrac{1}{2} A = \text{vers } A \cot \tfrac{1}{2} A$

$\sin A = \pm\sqrt{\tfrac{1}{2}(1 - \cos 2A)} = \pm\sqrt{\tfrac{1}{2}\text{ vers } 2A}$

$\cos A = \frac{1}{\sec A} = \pm\sqrt{1 - \sin^2 A} = \cot A \sin A$

$\cos A = \cos^2 \tfrac{1}{2} A - \sin^2 \tfrac{1}{2} A = 2\cos^2 \tfrac{1}{2} A - 1 = 1 - 2\sin^2 \tfrac{1}{2} A$

$\cos A = \pm\sqrt{\tfrac{1}{2} + \tfrac{1}{2}\cos 2A} = 1 - \text{vers } A$

$\tan A = \frac{1}{\cot A} = \frac{\sin A}{\cos A} = \pm\sqrt{\sec^2 A - 1}$

$\tan A = \pm\sqrt{\frac{1}{\cos^2 A} - 1} = \frac{\pm\sqrt{1 - \cos^2 A}}{\cos A} = \frac{2\tan \tfrac{1}{2} A}{1 - \tan^2 \tfrac{1}{2} A}$

$\tan A = \frac{1 - \cos 2A}{\sin 2A} = \frac{\sin 2A}{1 + \cos 2A} = \pm\sqrt{\frac{1 - \cos 2A}{1 + \cos 2A}} = \frac{\text{vers } 2A}{\sin 2A}$

$\cot A = \frac{1}{\tan A} = \frac{\cos A}{\sin A} = \pm\sqrt{\csc^2 A - 1}$

$\text{vers } A = 1 - \cos A = \sin A \tan \tfrac{1}{2} A = 2\sin^2 \tfrac{1}{2} A$

$\text{covers } A = 1 - \sin A$

$\sin \tfrac{1}{2} A = \pm\sqrt{\frac{1 - \cos A}{2}} = \pm\sqrt{\frac{\text{vers } A}{2}}$

$\cos \tfrac{1}{2} A = \pm\sqrt{\frac{1 + \cos A}{2}}$

$\tan \tfrac{1}{2} A = \pm\sqrt{\frac{1 - \cos A}{1 + \cos A}} = \frac{1 - \cos A}{\sin A} = \frac{\sin A}{1 + \cos A} = \frac{\tan A}{1 + \sec A} = \frac{\text{vers } A}{\sin A}$

$\cot \tfrac{1}{2} A = \pm\sqrt{\frac{1 + \cos A}{1 - \cos A}} = \frac{\sin A}{\text{vers } A}$

$\text{vers } \tfrac{1}{2} A = \frac{\tfrac{1}{2}\text{ vers } A}{1 + \sqrt{1 - \tfrac{1}{2}\text{ vers } A}} = \frac{1 - \cos A}{2 + \sqrt{2(1 + \cos A)}}$

$\sin 2A = 2 \sin A \cos A$

$\cos 2A = 2\cos^2 A - 1 = \cos^2 A - \sin^2 A = 1 - 2\sin^2 A$

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39. $\tan 2A = \frac{2 \tan A}{1 - \tan^2 A}$

40. $\cot 2A = \frac{\cot^2 A - 1}{2 \cot A}$

41. $\text{vers } 2A = 2 \sin^2 A = 2 \sin A \cos A \tan A$

42. $\sin (A \pm B) = \sin A \cdot \cos B \pm \sin B \cdot \cos$

43. $\cos (A \pm B) = \cos A \cdot \cos B \mp \sin A \cdot \sin$

44. $\tan (A \pm B) = \frac{\tan A \pm \tan B}{1 \mp \tan A \cdot \tan B}$

45. $\cot (A \pm B) = \frac{\cot A \cdot \cot B \mp 1}{\cot B \pm \cot A}$

46. $\sin A + \sin B = 2 \sin \frac{1}{2}(A + B) \cos \frac{1}{2}(A$

47. $\sin A - \sin B = 2 \cos \frac{1}{2}(A + B) \sin \frac{1}{2}(A$

48. $\cos A + \cos B = 2 \cos \frac{1}{2}(A + B) \cos \frac{1}{2}(A$

49. $\cos A - \cos B = -2 \sin \frac{1}{2}(A + B) \sin \frac{1}{2}$

50. $\tan A + \tan B = \frac{\sin (A + B)}{\cos A \cdot \cos B}$

51. $\tan A - \tan B = \frac{\sin (A - B)}{\cos A \cdot \cos B}$

52. $\sin^2 A - \sin^2 B = \cos^2 B - \cos^2 A = \sin (A$

53. $\cos^2 A - \sin^2 B = \cos^2 B - \sin^2 A = \cos (A$

15. Oblique T

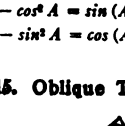

Fig 60

Given Data	To Find	
Two Angles and a Side A, B, a	b, c, C	$b = a \cdot \frac{\sin B}{\sin A}$ $C = 180° - ($
	Area	$K = \frac{a^2}{2} \cdot \frac{\sin B}{\sin}$
Two Sides and an Angle Opposite One of Them a, b, A	B, C, c	If $a < b$ and A No solution. If $a \leq b$, or if a If $a < b$, but $>$ two solutions $\sin B = (b + a$ B may have t $C_1 = 180° - ($ $c_1 = a \cdot \frac{\sin C_1}{\sin A}$
	Area	$K = \frac{1}{2} ab \cdot \sin$

Oblique Triangles (*Continued*)

		$\frac{1}{2}(A+B) = 90° - \frac{1}{2}C$
		$\tan \frac{1}{2}(A-B) = \frac{a-b}{a+b} \cdot \tan \frac{1}{2}(A+B)$
		$A = \frac{1}{2}(A+B) + \frac{1}{2}(A-B)$
o Sides and he Included ngle a, b, C	A, B, c	$B = \frac{1}{2}(A+B) - \frac{1}{2}(A-B)$
		$c = a \cdot \frac{\sin C}{\sin A} = b \cdot \frac{\sin C}{\sin B}$
		$c = (a+b)\frac{\cos \frac{1}{2}(A+B)}{\cos \frac{1}{2}(A-B)} = (a-b)\frac{\sin \frac{1}{2}(A+B)}{\sin \frac{1}{2}(A-B)}$
	Area	$K = \frac{1}{2}ab \cdot \sin C$
		Let $S = \frac{1}{2}(a+b+c)$ = the semi-perimeter
		$\sin \frac{1}{2}A = \sqrt{\frac{(s-b)(s-c)}{bc}}$
		$\cos \frac{1}{2}A = \sqrt{\frac{s(s-a)}{bc}}$
ee Sides , b, c	A, B, C	$\tan \frac{1}{2}A = \sqrt{\frac{(s-b)(s-c)}{s(s-a)}}$
		$\sin A = \frac{2}{bc}\sqrt{s(s-a)(s-b)(s-c)}$
		$\tan \frac{1}{2}B = \sqrt{\frac{(s-a)(s-c)}{s(s-b)}}$
		$C = 180° - (A+B)$
	Area	$K = \sqrt{s(s-a)(s-b)(s-c)}$

ANALYTICAL GEOMETRY

16. Straight Line

ctangular and polar coordinates (Fig 61). Rectangular coordinates and polar coordinates (ρ, θ) of any point P are related thus: $x = \rho \cos \theta$, $\sin \theta$; $x^2 + y^2 = \rho^2$; $\tan \theta = y \div x$.

e. — The x's and y's in following formulas are rectangular coordinates.

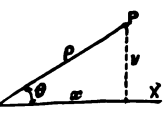

Fig 61

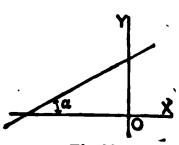

Fig 62

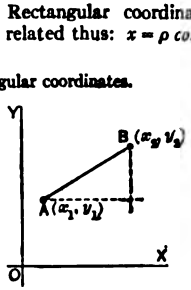

Fig 63

pe. The slope of a line is the tangent of the angle α from the x-axis to **ne.** It is denoted by m, $m = \tan \alpha$. If two lines are parallel, their slopes, m', are equal. If the lines are perpendicular to each other, $m = -1 \div m'$.

ope of the line through A (x_1, y_1) and B (x_2, y_2), Fig 3, is

$$m = (y_2 - y_1) \div (x_2 - x_1)$$

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Distance between two points, A (

$$d = \sqrt{(x_1 - x_2)}$$

Line divided in given ratio. The divides the line from A (x_1, y_1) to B $(x_2,$

$$x = (x_1 + rx_2) \div (1 + r),$$

Equation of straight line. Every as $Ax + By + C = 0$, represents a straigh

Line with slope m and y-intercept b, $y = m$
Line with slope m and through point $(x', y'$
Line through two points (x', y') and $(x'', y'$
Line with x-intercept a and y-intercept b, (

Angle θ from line with slope m to line

$$\tan\theta = (m' - m)$$

Distance from point (x', y') **to line**

$$d = (Ax' + By' + C$$

Area of triangle with vertices (x_1, y_1)

$$A = [x_1 (y_2 - y_3) + x_2 (y_2 -$$

17. Ci

Equation. The equation of the circle is $x^2 + y^2 = r^2$. If the center is (a, b) t Every equation of the form

$$x^2 + y^2 + 2Ax +$$

represents a circle. The center is $(-A,$

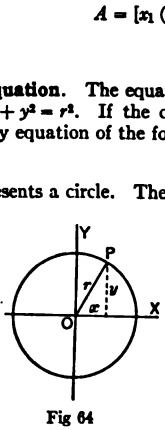
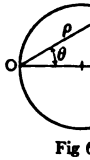


Fig 64



Fig

Tangents. Equation of tangent to th on the curve is $x'x + y'y = r^2$. Equations

$$y = mx \pm r\sqrt$$

Length of tangent from any point (x', y')

$x^2 + y^2 + 2Ax + 2By + C = 0$, is $l =$

Circle through three points. To fi points, as $(3, 1)$, $(-1, 2)$, $(5, 4)$, substitute equation (C) above, and find A, B, C fro

Polar equation. Origin on circumfere line, equation of circle is $\rho = a\cos\theta$. Dia (Fig 66) is $\rho = a\sin\theta$.

18. Ellipse

ation of ellipse with center at origin and foci F' and F on the x-axis $\overline{F'F}$), is

$$(x^2 + a^2) + (y^2 + b^2) = 1 \qquad (E)$$

a is the major semiaxis and b the minor semiaxis.

tance $OF = \sqrt{a^2 - b^2}$; distance $BF = OA =$ al width $QR = 2b^2 + a$.

centricity e of ellipse is $e = \sqrt{a^2 - b^2} + a =$ OA, and e is always less than 1. When e is o, the ellipse is nearly a circle, and the re near the center. When e is nearly 1, is long and narrow, and foci are near ends major axis. If two ellipses have the same ricity, the ratio of semiaxes, $a : b$, is the for both, and conversely. The equation

$$(x^2 + a^2) + (y^2 + b^2) = k$$

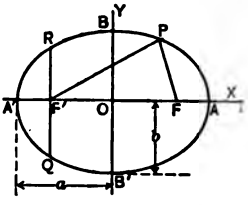


Fig 67

a and b are fixed, represents, whenever any positive value is assigned to ellipse; and all these ellipses have the same e.

ocal ellipses. If two ellipses $(x^2 + a^2) + (y^2 + b^2) = 1$ and $(x^2 + a'^2) + (y' + b^2)$ e the same foci, then $a^2 - b^2 = a'^2 - b'^2$. In other words, $a^2 - a'^2 = b^2 - b'^2$.

l radii to the point $P(x', y')$ on the ellipse are $F'P = a + ex'$ and $FP = a - ex'$. akes $F'P + FP = 2a = A'A$.

gents to the ellipse (E) at (x', y') on the curve $(x'x + a^2) + (y'y + b^2) = 1$; with $y = mx \pm \sqrt{a^2m^2 + b^2}$.

neters (Fig 68). Equation of diameter AB, which bisects all chords having as CD, EF, in ellipse (E) is $y = (-b^2 + a^2m)\,x$. If the slopes m and m' of two ers of (E) are such that $mm' = -b^2 + a^2$, each diam bisects all chords parallel other. They are called conjugate diameters; for example, AB and JK.

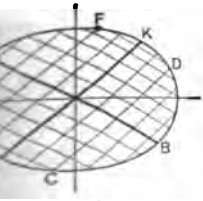


Fig 68

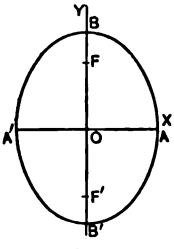


Fig 69

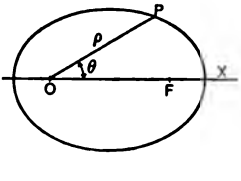


Fig 70

on y-axis (Fig 69). If the foci of an ellipse are on the y-axis, its equation is (E) . but b is the major semiaxis, a the minor, so that $OF = \sqrt{b^2 - a^2}$, $AF = b$, and $\sqrt{b^2 - a^2} + b$.

equation (Fig 70). Initial line along major axis of ellipse, origin at left-hand $\rho = b^2 + a(1 - e\cos\theta)$. With origin at right-hand focus, $\rho = b^2 + a(1 + e\cos\theta)$. **igin at center,** $\rho^2 = b^2 + (1 - e^2\cos^2\theta)$, where e is the eccentricity.

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19. Hy

Equation. Equation of hyperbola origin at center (Fig 71):

$$(x^2 \div a^2) -$$

where a is the principal semiaxis, b = √ pal a width

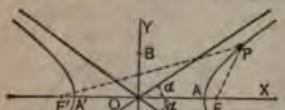

Fig 71

As hype 2α is $\alpha = b$ the p P to to a^2(

Eccentricity of hyperbola is great where α is half the angle between the a angle 2α between the asymptotes is a and when e is between $\sqrt{2}$ and 1, 2α i

If two hyperbolas, such as (H), have the s totes, and conversely. They also have the s

The equation $(x^2 \div a^2) - (y^2 \div b^2) = k$, wh values, represents various hyperbolas, all of semiaxes are $a\sqrt{k}$ and $b\sqrt{k}$.

Focal radii to any point $P(x', y')$ on hy $F'P = ae + x'$ and $FP = ae - x'$ (e = eccen

Tangents to hyperbola (H) at (x', y') on slope m, $y = mx \pm \sqrt{a^2m^2 - b^2}$.

Diameter (Fig 72). Equation of diam that have slope m is $y = (b^2 \div a^2m)\,x$. If t are such that $mm' = b^2 \div a^2$, each bisects parallel to the other. They are called diameters.

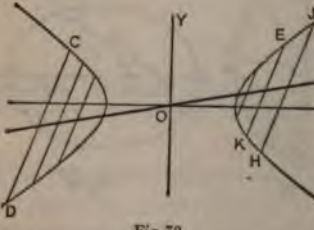

Fig 72

Foci on y-axis (Fig 73). If the foci of th at center, the equation of the hyperbola is

$$(y^2 \div b^2) - (x^2$$

where b is principal semiaxis, and a is conjuga

eccentricity $e = \sqrt{a^2 \div b^2} \div b$; asym

jugate hyperbolas, that is, the hyperbolas (H) and (H'), have the same asymp- Fig 74). The diam which bisects a set of parallel chords in one bisects the same parallel chords in the other. The two segments intercepted on any straight line o conjugate hyperbolas are equal. Eccentricities e and e' of conjugate hyperbolas ated thus: $(1 + e^2) + (1 + e'^2) = 1$.

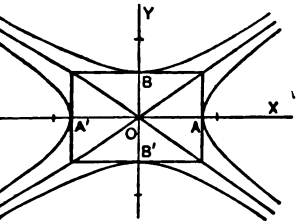

Fig 74

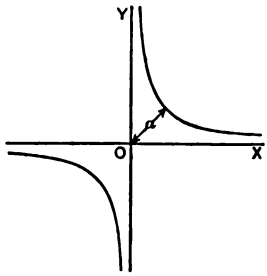

Fig 75

ctangular or equilateral hyperbola is one having its axes equal, so that $a = b$. quation (H) is then $x^2 - y^2 = a^2$. (H'')
rmptotes of (H'') are perpendicular to each other.

ymptotes as axes of coordinates (Fig 75). In this case, the equation of the gular hyperbola is $xy = a^2 \div 2$. If the hyperbola is not rectangular (Fig 76), the otes are not perpendicular. But if they are taken as axes and the coordinates x and y are measured parallel to the asymptotes, the equation of the hyperbola is

$$4xy = a^2 + b^2$$

Polar equation same as for ellipse, with $-b^2$ in place of b^2.

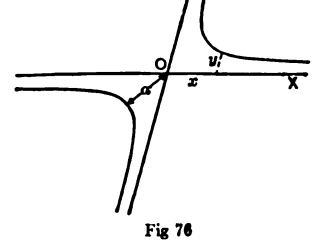

Fig 76

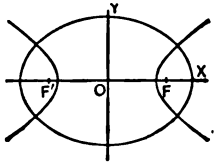

Fig 77

nfocal hyperbolas. If two hyperbolas, $(x^2 \div a^2) - (y^2 + b^2) = 1$ and $(x^2 + a'^2)$ $+ b'^2) = 1$, have the same foci, then $a^2 + b^2 = a'^2 + b'^2$.

nfocal ellipse and hyperbola. If an ellipse and a hyper- have the same foci, they intersect at right angles (Fig 77).

20. Parabola

quation. With focus on x-axis and origin at vertex, tion of parabola is $y^2 = 4px$, where $p = OF$ = distance vertex to focus (Fig 78). If p is positive, parabola to the right; if negative, to the left.

ocal width, $QR = 4p$.

ocal radius to any point $P(x', y')$ on parabola is $FP =$ x'.

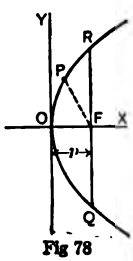

Fig 78

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Tangent to parabola $y^2 = 4px$ at $(x', y'$ with slope m, $y =$

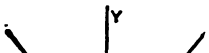


Fig 79

Diameter. E of parabola $y^2 =$ Every diam of a

Focus on y-a having focus on $4py$, where $p =$ of positive end of y-axis (usually taken u of negative end of y-axis if p is negative.

Polar equation of parabola, origin bein

21. Curves in

An equation of second degree in x and y

$$ax^2 + 2hxy + by^2 + 2$$

represents a conic section; an ellipse, if parabola, if $h^2 = ab$; a circle, if $a = b$ a $abc - af^2 - bg^2 - ch^2 + 2fgh = 0$, the curve lines, or a point.

Inclination of axis. The angle θ of i conic (G) to the x-axis is given by $\tan 2\theta$

Center (x', y') of ellipse or hyperbol equations: $ax' + hy' + g = 0$, $hx' + by' + f$

Intersections of two curves are found by so

Intercepts. To find x-intercepts of a curv x; to find y-intercepts, set $x = 0$ in the equati

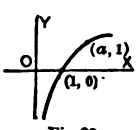


Fig 80

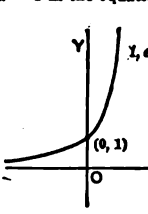


Fig 81

Special Curves. Logarithmic and exponen

$y = \log_a x$ (Fig 80) $y = e^x$ (Fig 81)

Change of axes changes the equation of a origin is moved to the point (h, k), axes rema $x + h$ for x, and $y + k$ for y. If axes are rotat substitute $x\cos\theta - y\sin\theta$ for x, and $x\sin\theta +$

22. Solid G

Distance from $P_1(x_1, y_1, z_1)$ to $P_2(x_2,$

$$d = \sqrt{(x_2 - x_1)^2 + (y$$

, $P_1Q = x_2 - x_1$, $QR = y_2 - y_1$, $RP_2 = z_2 - z_1$, these segments being parallel
ie axes (Fig 83).

irection cosines of P_1P_2 are

$$= \frac{x_2 - x_1}{d}, \quad \cos\beta = \frac{y_2 - y_1}{d}, \quad \cos\gamma = \frac{z_2 - z_1}{d},$$

e d is the distance from P_1 to P_2, given
e. The sum of the squares of direction
es is always 1, $\cos^2\alpha + \cos^2\beta + \cos^2\gamma = 1$.
e direction cosines of a line are propor-
l to a, b, c, they are $\cos\alpha = a \div k$, $\cos\beta =$
, $\cos\gamma = c \div k$, where $k = \sqrt{a^2 + b^2 + c^2}$.

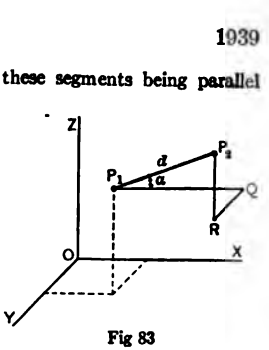


Fig 83

ıne. Every equation of first degree in x, y, and z, as

$$Ax + By + Cz = D \qquad (P)$$

ents a plane. Its intercepts on the x-, y- and z-axes are $D \div A$, $D \div B$,
. If the plane goes through the origin, $D = 0$. The direction cosines of
perpendicular to the plane are $\cos\alpha = A \div q$, $\cos\beta = B \div q$, $\cos\gamma = C \div q$,
$q = \sqrt{A^2 + B^2 + C^2}$.

stance from point (x', y', z') **to plane** (P) is

$$r = (Ax' + By' + Cz' - D) \div q$$

nce from origin to plane is $p = D \div q$.

gle θ **between two planes,** $Ax + By + Cz = D$ and $A'x + B'y + C'z = D'$,
n by $\cos\theta = (AA' + BB' + CC') \div \sqrt{A^2 + B^2 + C^2} \cdot \sqrt{A'^2 + B'^2 + C'^2}$,

$$\cos\theta = \cos\alpha\cos\alpha' + \cos\beta\cos\beta' + \cos\gamma\cos\gamma'$$

$\cos\alpha$, $\cos\beta$, $\cos\gamma$ are direction cosines of a perpendicular to one plane,
s α', $\cos\beta'$, $\cos\gamma'$ are direction cosines of a perpendicular to the other.
E BETWEEN TWO LINES, the direction cosines of which are known, is given
: second of the foregoing formulas.

pendicular lines or planes. Two lines are perpendicular if

$$\cos\alpha\cos\alpha' + \cos\beta\cos\beta' + \cos\gamma\cos\gamma' = 0$$

same condition holds for planes. More simply, the two planes

$$Ax + By + Cz = D \quad \text{and} \quad A'x + B'y + C'z = D'$$

rpendicular if $AA' + BB' + CC' = 0$.

e of intersection of two planes, $Ax + By + Cz = D$ and $A'x + B'y + C'z$
has the direction cosines: $\cos\alpha = a \div s$, $\cos\beta = b \div s$, $\cos\gamma = c \div s$, where
$= BC' - B'C$, $b = AC' - A'C$, $c = AB' - A'B$, and $s = \sqrt{a^2 + b^2 + c^2}$

ıations of straight line through two points (x_1, y_1, z_1) and (x_2, y_2, z_2):

$$(x - x_1) \div (x_2 - x_1) = (y - y_1) \div (y_2 - y_1) = (z - z_1) \div (z_2 - z_1)$$

ions of line through (x_1, y_1, z_1), with direction cosines $\cos\alpha$, $\cos\beta$, $\cos\gamma$:

$$(x - x_1) \div \cos\alpha = (y - y_1) \div \cos\beta = (z - z_1) \div \cos\gamma$$

ne through three points, P (x_1, y_1, z_1), Q (x_2, y_2, z_2), R (x_3, y_3, z_3). The
on of the plane may be taken as $ax + by + cz = 1$, by dividing (P) above by
ess $D = 0$. Substitute in this equation, for x, y, and z, the coordinates of
n those of Q and finaliy those of R. This gives three equations, from

which the unknowns a, b, and c can be found. This process fails if the pl through P, Q, R goes through the origin, in which case, the equation is

$$Ax + By + Cz = 0$$

and may be divided by A, becoming $x + by + cz = 0$.

Sphere. Equation of sphere, with center (a, b, c) and radius r, is

$$(x-a)^2 + (y-b)^2 + (z-c)^2 = r^2$$

Cylinder. Equation of circular cylinder, with axis on z-axis and radi is $x^2 + y^2 = r^2$. Every equation containing only two of the variables x, represents in solid geometry a cylinder, the base of which is the curve d mined in one of the coordinate planes by the given equation.

Surface of revolution. If any curve $y = f(x)$ rotates about the x- the equation of the surface generated is $y^2 + z^2 = [f(x)]^2$. Thus, if the el $(x^2 \div a^2) + (y^2 \div b^2) = 1$ (that is, $y = (b \div a)\sqrt{a^2 - x^2}$) rotates about the x- the equation of the resulting ellipsoid of revolution is $y^2 + z^2 = (b^2 \div a^2)(a^2 -$ or $(x^2 + a^2) + (y^2 \div b^2) + (z^2 \div c^2) = 1$.

Intercepts. The x-intercept of any surface is found by setting $y = 0$ $z = 0$ in the equation of the surface; similarly for y and z intercepts.

Traces. The trace of any surface on the xy-plane is the curve found putting $z = 0$ in the equation of the surface; similarly for the xz-trace and trace. Thus, the xy-trace of the plane $2x - y + 3z = 11$ is the line $2x - y =$ The yz-trace is $-y + 3z = 11$.

Cross-section of surface. The equation of the cross-section of any face the equation of which is known, taken parallel to the xy-plane at the tance k from the xy-plane, is found by putting $z = k$ in the equation of surface. For cross-section parallel to yz, put $x = k$. For cross-sec parall xz, put $y = k$. Thus, cross-sec of sphere $x^2 + y^2 + z^2 = 36$, parallel to xy a tance $z = 2$, is $x^2 + y^2 + 4 = 36$, or $x^2 + y^2 = 32$. It is a circle with radius $\sqrt{}$

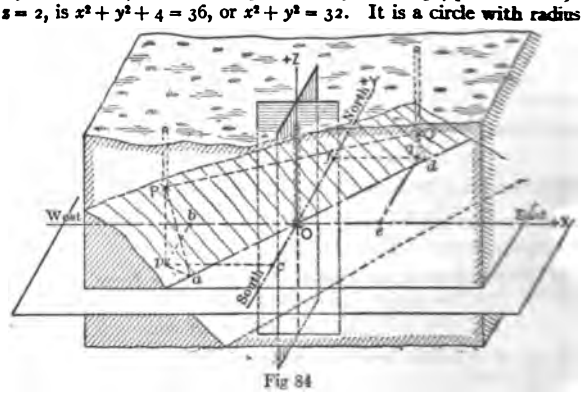


Fig 84

Dip and strike of stratum. (See also Sec 10.) Suppose three points of a str (Fig 84), which is approx a true plane, are determined as follows:

Point	Latitude	Departure	Elev	x	y	z
O.....	−312.5	305.4	517.4	0	0	0
P.....	−624.2	124.6	759.7	−311.7	−180.8	24: 1
Q.....	− 48.0	558.7	621.4	264.5	−353.3	104

e a horiz plane through the lowest point O as xy-plane, and the vertical through O is (elev). Then the coordinates of O, P, and Q are as above. Let the equation of atum plane be $x + by + cz = 0$. (See Equation of Plane Through Three Points) Substituting the coordinates of P, and then of Q, for x, y and z,

$$-311.7 - 180.8\,b + 242.3\,c = 0, \text{ and } 264.5 + 253.3\,b + 104\,c = 0;$$

$;6\,b + c = +1.286$, and $2.435\,b + c = -2.543$. Hence $b = -2.267$, $c = 2.977$, e equation of the stratum plane is

$$x - 2.267\,y + 2.977\,z = 0 \qquad (S)$$

on of xy or horiz plane is $z = 0$, or

$$0x + 0y + z = 0 \qquad (H)$$

is angle between (H) and (S) (see Angle between Two Planes),

$$\cos\theta = \frac{1 \times 0 - 2.267 \times 0 + 2.977 \times 1}{\sqrt{1^2 + (-2.267)^2 + 2.977^2} \times \sqrt{0^2 + 0^2 + 1^2}} = 0.769$$

$\theta = 39° 40'$ approx = dip.

ind the strike, find the intersection of stratum plane (S) with horiz plane (H). s, put $z = 0$ in (S). This gives the horiz line

$$x - 2.267\,y = 0, \text{ or } y = 0.441\,x \qquad (L)$$

ope of this line, $\tan\alpha$, is the tangent of the complement of the strike angle. But = 0.441. Hence, $\alpha = 23° 50'$, $\beta = 90° - \alpha = 66° 10'$, and strike = N 66° 10′ E.

CALCULUS

23. Derivative

$f(x)$ be any function of x, and denote it by y; then $y = f(x)$. When x es by any amount, Δx, then y changes by some amount, Δy. The de-

ve of the function with respect to x, denoted by $\frac{dy}{dx}$ or $f'(x)$, is

$$\lim_{\Delta x \doteq 0} \frac{\Delta y}{\Delta x} = \lim_{\Delta x \doteq 0} \frac{f(x+\Delta x) - f(x)}{\Delta x} = \frac{dy}{dx} = f'(x)$$

a table of derivatives see Art 25. (Symbol $\doteq$ means "approaching.")

pe. Value of the derivative for a particular value of x, $x = x_1$, is the of the curve $y = f(x)$ at the point (x_1, y_1).

gent and normal. Equation of the tangent to any curve $y = f(x)$ at oint (x_1, y_1) is $y - y_1 = f'(x_1)(x - x_1)$, where $f'(x_1)$ denotes the value of when $x = x_1$.

ation of normal at (x_1, y_1) is $y - y_1 = (-1 + f'(x_1))(x - x_1)$.

xima and minima. To find maximum or minimum of a given function set $f'(x) = 0$ and find its roots, say $x = x_1$, $x = x_2$, . . . Then $f(x)$ has a

maximum for $x = x_1$ if $f''(x_1)$ is negative,
minimum for $x = x_1$ if $f''(x_1)$ is positive,

$f''(x_1)$ means the result of substituting x_1 for x in $f''(x)$, the second tive of $f(x)$ with respect to x.

owever, $f'(x_1) = 0$, compute $f'''(x_1)$, $f''''(x_1)$ and further derivatives, until one n, say the kth, is found which is not zero when $x = x_1$. Then $f(x)$ has a maximum $= x_1$ if k is even and $f^k(x_1)$ is negative, a minimum if k is even and $f^k(x_1)$ is posi- ut a point of inflexion on a horizontal tangent if k is odd.

at of inflexion. To find points of inflexion of the curve $y = f(x)$, find ots of $f''(x) = 0$. Any such root x_1 gives a point of inflexion, unless $) = 0$ also. For this exceptional case, see preceding note.

Curvature. Curvature k of curve $y = f(x)$, at any point (x, y), is

$$k = f''(x) \div \sqrt{(1 + [f'(x)]^2)^3}$$

Radius R of curvature is the reciprocal of curvature, $R = 1 \div k$.
Center (a, b) of curvature,

$$a = x - f'(x)\,(1 + [f'(x)]^2) \div f''(x)$$
$$b = y + (1 + [f'(x)]^2) \div f''(x)$$

Angle from radius vector to tangent, in polar coordinates. If the tion of the curve is $\rho = f(\theta)$, the angle ϕ from radius vector to tangent is by $\tan \phi = \rho\,(d\theta \div d\ )$.

Taylor's theorem.

$f(x) = f(a) + f'(a)(x-a) + [f''(a)\,(x-a)^2 \div 2!] + [f'''(a)(x-a)^3 \div 3!] +$ ·

where $3!$ denotes $3 \cdot 2 \cdot 1$. If $a = 0$, the theorem is

$$f(x) = f(0) + f''(0)\,x + [f''(0)\,x^2 \div 2!] + [f'''(0) \div 3!] + \cdots$$

Usually this series is more and more nearly equal to $f(x)$ when more and terms are taken, but there are many exceptions.

Derivative of arc. If s denotes length of arc of a curve $y = f(x)$, s va as the point (x, y) moves along the curve, then $\dfrac{ds}{dx} = \sqrt{1 + \left(\dfrac{dy}{dx}\right)^2}$

Function of a function. If y is a function of u and u is a function $\dfrac{dy}{dx}$ may be found by $\dfrac{dy}{dx} = \dfrac{dy}{du} \times \dfrac{du}{dx}$.

24. Integration

Integral. The integral of $f(x)$, denoted by $\int f(x)\,dx$, means the fun the derivative of which is $f(x)$. For example, $\int 3x^2\,dx = x^3 + C$, C being constant.

For a table of integrals of a few functions, see Art 26.

Definite integral. The symbol $\int_a^b f(x)\,dx$ denotes the result of fin $\int f(x)\,dx$, substituting in the resulting function b for x, then a for x and tracting, thus: $\int_a^b f(x)\,dx = \phi(b) - \phi(a)$, where $\phi(x) = \int f(x)\,dx$

The definite integral gives the value of the limit of the sum of term structed as follows: Divide the interval from a to b into n equal parts denote the abscissas x of the points of division by $x_0, x_1, x_2, \ldots x_{n-1}, x_n$ be and x_n being b. Let each part of the interval thus divided be Δx. Then

$$\int_a^b f(x)\,dx = \lim_{n \doteq \infty} [f(x_1)\,\Delta x + f(x_2)\,\Delta x + \cdots + f(x_n)\,\Delta x]$$

Area. In rectangular coordinates, the area under the curve $y = f$ cluded between ordinates, at $x = a$ and $x = b$ is $A = \int_a^b f(x)\,dx$. If the c crosses the x-axis between a and b, compute area to crossing, and then beyo In polar coordinates, area between the curve $\rho = f(\theta)$ and two radii at θ

$= \theta_2$ is given by $A = \frac{1}{2}\int_{\theta_1}^{\theta_2} \rho^2\, d\theta$. For double integration the element of
A is $dA = dy\,dx$, or $dA = \rho\, d\rho\, d\theta$.

gth of arc of curve $y = f(x)$ from $P_1(x_1, y_1)$ to $P_2(x_2, y_2)$:

$$s = \int_{x_1}^{x_2} \sqrt{1 + m^2}\, dx, \qquad \text{where } m = f'(x)$$

$$s = \int_{\theta_1}^{\theta_2} \sqrt{\rho^2 + k^2}\, d\theta, \qquad \text{where } k = \frac{d\rho}{d\theta}$$

ume and surface of revolution. If the curve $y = f(x)$ revolves about the volume generated by area under the curve and between ordinates x_1 and $x = x_2$, is $V = \pi \int_{x_1}^{x_2} y^2\, dx$, and surface generated by arc of curve is

$$S = 2\pi \int_{x_1}^{x_2} x\sqrt{1 + m^2}\, dx, \qquad \text{where } m = f'(x)$$

ter pressure. Pressure on vertical area bounded by the curve $y = f(x)$ wo horizontal lines at $y = y_1$ and $y = y_2$ is $P = k \int_{y_1}^{y_2} hw\, dy$, where k is pth below water surface of a narrow horiz strip at distance y from origin, e width of this strip, and ight per unit volume. h nust be expressed in terms fore proceeding with the tion.

nent of volume, multi-tegration. Element of under surface the equa- which is $z = f(x, y)$:

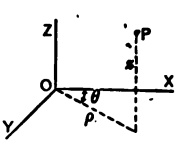


Fig 85. Cylindrical Coordinates

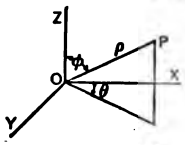


Fig 86. Spherical Coordinates

$dV = z\,dx\,dy$, for double integration
$dV = dz\,dx\,dy$, for triple integration
} rectangular coordinates.
$dV = \rho\, d\rho\, d\theta\, dz$, cylindrical coordinates,
$dV = \rho^2 \sin\phi\, d\phi\, d\theta\, d\rho$, spherical coordinates.

25. Table of Derivatives

is table u and v denote functions of x; a, c, and n are arbitrary constants.

$c = 0$

$(uv) = u\frac{dv}{dx} + v\frac{du}{dx}$

$\left(\frac{u}{v}\right) = \left(v\frac{du}{dx} - u\frac{dv}{dx}\right) \div v^2$

$u^n = nu^{n-1}\frac{du}{dx}$

$e^u = e^u\frac{du}{dx}$ (where $e = 2.718+$)

$a^u = a^u \log_e a \cdot \frac{du}{dx}$

$u^v = vu^{v-1} + u^v \log_e u\frac{dv}{dx}$

$\frac{d}{dx}\log_e u = \frac{1}{u}\frac{du}{dx}$

$\frac{d}{dx}\log_{10} u = \frac{\log_{10} e}{u} \cdot \frac{du}{dx}$

$\frac{d}{dx}\sin u = \cos u\frac{du}{dx}$ (u in radians)

$\frac{d}{dx}\cos u = -\sin u\frac{du}{dx}$

$\frac{d}{dx}\tan u = \sec^2 u\frac{du}{dx}$

$\frac{d}{dx}\cot u = -\csc^2 u\frac{du}{dx}$

$\frac{d}{dx}\sec u = \sec u \tan u\frac{du}{dx}$

$$\frac{d}{dx}\csc u = -\csc u \cot u \frac{du}{dx}$$

$$\frac{d}{dx}\sin^{-1} u = \frac{1}{\sqrt{1-u^2}}\frac{du}{dx}$$

$$\frac{d}{dx}\cos^{-1} u = -\frac{1}{\sqrt{1-u^2}}\frac{du}{dx}$$

$$\frac{d}{dx}\tan^{-1} u = \frac{1}{1+u^2}\frac{du}{dx}$$

$$\frac{d}{dx}\cot^{-1} u = -\frac{1}{1+u^2}\frac{du}{dx}$$

$$\frac{d}{dx}\sec^{-1} u = \frac{1}{u\sqrt{u^2-1}}\frac{du}{dx}$$

$$\frac{d}{dx}\csc^{-1} u = -\frac{1}{u\sqrt{u^2-1}}\frac{du}{dx}$$

26. Table of Integrals

The base e is assumed for logarithms unless otherwise noted. A constant is understood to be added to each result.

$$\int dx = x + C$$

$$\int x^n dx = \frac{x^{n+1}}{n+1}, \quad \text{except when } n = -1$$

$$\int x^{-1} dx = \int \frac{dx}{x} = \log x$$

$$\int \frac{dx}{ax+b} = \frac{1}{a}\log(ax+b)$$

$$\int (ax+b)^n dx = \frac{(ax+b)^{n+1}}{a(n+1)}, \quad \text{except when } n = -1$$

$$\int \frac{dx}{ax^2+b} = \frac{1}{\sqrt{ab}}\tan^{-1}\sqrt{\frac{a}{b}}\,x, \quad \text{when } a > 0 \text{ and } b > 0$$

$$\int \frac{dx}{ax^2-b} = \frac{1}{2\sqrt{ab}}\log\frac{\sqrt{a}\,x-\sqrt{b}}{\sqrt{a}\,x+\sqrt{b}}, \quad \text{when } a > 0 \text{ and } b > 0$$

$$\int \sqrt{x^2+a^2}\,dx = \tfrac{1}{2}\left[x\sqrt{x^2+a^2} + a^2\log\left(x+\sqrt{x^2+a^2}\right)\right]$$

$$\int \sqrt{x^2-a^2}\,dx = \text{same as above, with } -a^2 \text{ for } a^2$$

$$\int \sqrt{a^2-x^2}\,dx = \tfrac{1}{2}\left(x\sqrt{a^2-x^2} + a^2\sin^{-1}\frac{x}{a}\right)$$

$$\int \frac{dx}{\sqrt{a^2-x^2}} = \sin^{-1}\frac{x}{a}$$

$$\int \log x\,dx = x(\log x - 1)$$

$$\int \log_{10} x\,dx = x\log_{10}\frac{x}{10}$$

$$\int e^{ax}dx = \frac{e^{ax}}{a}$$

$$\int a^x dx = \frac{a^x}{\log a}$$

$$\int xe^x dx = e^x(x-1)$$

$$\int \sin x\,dx = -\cos x$$

$$\int \cos x\,dx = \sin x$$

$$\int \sin^2 x\,dx = \frac{x}{2} - \frac{\sin 2x}{4}$$

$$\int \cos^2 x\,dx = \frac{x}{2} + \frac{\sin 2x}{4}$$

$$\int \tan x\,dx = -\log\cos x$$

$$\int \cot x\,dx = \log\sin x$$

$$\int \sec x\,dx = \log(\sec x + \tan x)$$

$$\int \sec^2 x\,dx = \tan x$$

$$\int \csc x\,dx = \log(\sec x - \tan x)$$

$$\int \csc^2 x\,dx = \cot x$$

$$\int x\sin x\,dx = \sin x - x\cos x$$

$$\int x^2\sin x\,dx = 2x\sin x + (2 - x^2)\cos x$$

$$\int x\cos x\,dx = \cos x + x\sin x$$

$$\int x^2\cos x\,dx = 2x\cos x + (x^2 - 2)\sin x$$

STATICS

27. Definitions

:hanics is that branch of science which treats of forces and motion. s deals with the action of forces on bodies at rest. DYNAMICS deals ie action of forces on bodies in motion.

ce is that which changes or tends to change the state of rest or motion ody. A force is completely specified by its magnitude, direction and f application. The word SENSE as applied to a force refers to one of the rections along the line of action of the force. The effect of any force l to a rigid body at rest is the same, no matter where in its own line of the force is applied. This is known as the principle of the TRANSMISSI-OF FORCE. A force may be represented graphi- magnitude and direction by a straight line parallel to its line of action, the length being ional to the magnitude of the force; its sense is ed by an arrowhead placed on the line. The engineers' UNIT OF FORCE is the pound, or the pull on a mass of 1 lb. A drawing which indi- he lines of action of the various forces acting on ine or structure is called a SPACE DIAGRAM; the which vectors are drawn to represent the magni- nd directions of the forces is a VECTOR DIAGRAM. A force is indicated on diagram by two lower-case letters placed on opposite sides of the line of of the force; the vector, representing its magnitude and direction, by ne capital letters placed at the ends. Thus, in Fig 87, AB represents gnitude and direction of the force W, and ab its action line. The vector ead as AB indicates a downward sense; read as BA, an upward sense. es are classified according to the arrangement of their action lines:

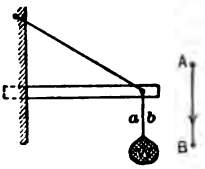


Fig 87

planar: *a*. concurrent; *b*. nonconcurrent; *c*. parallel
oncoplanar: *a*. concurrent; *b*. nonconcurrent; *c*. parallel

number of forces taken collectively is a system or set of forces. A is coplanar or noncoplanar according as the lines of action of the forces o not lie in the same plane, and is concurrent or nonconcurrent according do or do not intersect in a point. Two or more forces which are the ent of a single force are COMPONENTS of the force. The operation of g a system of forces by a simpler system is called COMPOSITION of forces. TION of forces is the operation of replacing a single force by a system es. RESULTANT of a force system is the simplest equivalent system. e cases the simplest equivalent is a single force; in other cases, it is a couple; in still others it is a pair of forces which are not parallel and do not intersect (noncoplanar).

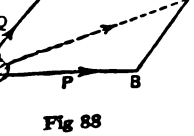


Fig 88

28. Resultants of Concurrent Forces

Resultant of two concurrent forces. PARALLELOGRAM LAW. If two concurrent forces P and Q (Fig 88), acting at the point O of a re represented in magnitude and direction by the adjacent sides OB and parallelogram $OACB$, their resultant is represented by diagonal OC through point of concurrency O. TRIANGLE LAW. If in the triangle

ABC (Fig 89) AB and BC represent two concurrent forces in magni direction and sense, then AC will represent their resultant in magni direction and sense; its action line will be ac, through the point currency, parallel to AC. The resultant may be found algebraically In Fig 90, let α be the angle between the action lines of forces P and Q,

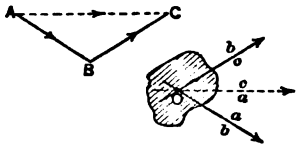


Fig 89

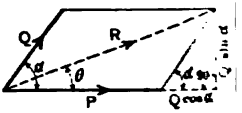


Fig 90

the angle between R and P. Then, $R^2 = P^2 + Q^2 + 2PQ \cos\alpha$, and $\tan\theta = \frac{Q \sin\alpha}{P + Q\cos\alpha}$. If $\alpha = 90°$, $R^2 = P^2 + Q^2$, and $\tan\theta = Q \div P$.

A force may be resolved into an infinite number of pairs of compone constructing different triangles as in Fig 91. The action lines of the c

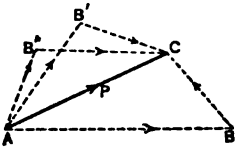


Fig 91

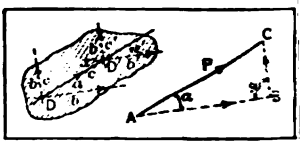
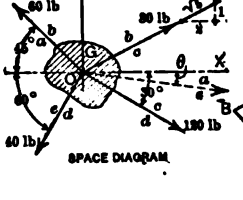


Fig 92

nents must be concurrent at a point on the action line of the force. A con problem is to resolve a force into rectangular components (often called re parts). In Fig 92, AB and BC are the rectangular components of the fo Expressed algebraically, $AB = P\cos\alpha$ and $BC = P\sin\alpha$. In general, th solved part of a force along any line equals the magnitude of the force time cosine of the angle between the lines of action of the force and its comp Action lines of the components are concurrent on the space diagram a point on ac, as at D or D'.



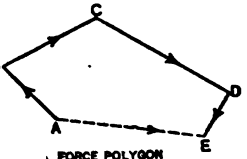


Fig 93

Resultant of coplanar concurrent forc found graphically as follows: In Fig 93, body G acted the 4 forces Construct a fo gon. Plat AB pa to ab, and scale represent 60 l. B plat BC par bc, and scale it resent 80 lb; manner plat DE, so that the arrows lead confluently from A to E. The resultant system is AE in magnitude and sense = 114 lb, and its action line is ac, an angle of $\theta = \tan^{-1} 0.146 = 8°\ 20'$ with the horizontal. ALGEBRAIC SO Choose rectangular axes OX and OY. Resolve each force into its x

ents, considering components acting upward or to the right as positive, ose acting downward or to the left as negative. Arrange the results in : form, placing the forces in the first column, the x components in the and the y components in the third. ΣF_x = algebraic sum of x com-s, and ΣF_y = algebraic sum of y components.

F, lb	F_x, lb	F_y, lb
= 60	$-60 \times 0.707 = -42.4$	$+60 \times 0.707 = +42.4$
= 80	$+80 \times 2/\sqrt{5} = +71.4$	$+80 \times 1/\sqrt{5} = +35.7$
= 120	$+120 \times 0.866 = +104$	$-120 \times 0.5 = -60$
= 40	$-40 \times 0.5 = -20$	$-40 \times 0.866 = -34.6$
	$\Sigma F_x = +113$	$\Sigma F_y = -16.5$

$R = \sqrt{\Sigma F_x^2 + \Sigma F_y^2} = \sqrt{13\,041} = 114$ lb. Sense is downward and to right (Fig 94). *Tan* $\theta = \dfrac{\Sigma F_y}{\Sigma F_x} =$ $\theta = 8° 20'$. If a concurrent system esultant, it is a single force.

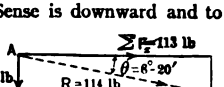


Fig 94

ultant of noncoplanar concurrent

Parallelopipedon law. Consider the 3 rectangular forces, P, Q, and 95). On these forces construct to scale a parallelopiped. Resultant of the system is represented in magnitude and direction by the diagonal; its value is $R = \sqrt{P^2 + Q^2 + S^2}$. Its direction cosines with respect to the axes are: $\cos \alpha = P \div R$, $\cos \beta = Q \div R$, and $\cos \gamma = S \div R$.

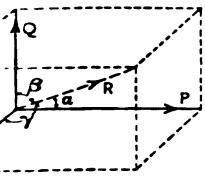


Fig 95

Resultant of any number of concurrent forces. Let the forces be specified with respect to 3 rectangular axes passing through the point of concurrency: (*a*) Resolve each force into components along the X, Y and Z-axes; (*b*) Find ebraic sums of the x, y and z components, and indicate them by ΣF_x, nd ΣF_y; (*c*) Find the resultant of these three partial resultants by the opipedon law; its value is $R = \sqrt{\Sigma F_x^2 + \Sigma F_y^2 + \Sigma F_z^2}$; its direction angles $\cos^{-1} \dfrac{\Sigma F_x}{R}$; $\beta = \cos^{-1} \dfrac{\Sigma F_y}{R}$; $\gamma = \cos^{-1} \dfrac{\Sigma F_z}{R}$

29. Moments and Couples

ent or torque of a force about a point is the product of the force mag- and the distance from the point to its action line. This perpendicular is called the arm of the force, and the point is the r center of moments. The product is the measure otational tendency of the force. The name of the moment is a combination of the names of force tance units: thus: foot-pound, inch-pound or inch- Moments tending to produce counterclockwise rota- a body will be considered positive, and clockwise e, unless contrary convention is mentioned. Thus, 96, the moment of force P about $O = +P \times OA$.

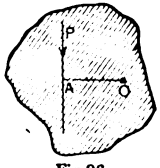


Fig 96

Moment of force about a line or axis. At any point on its action resolve the force into 2 rectangular components, one being parallel to the The product of the perpendicular component and the perpendicular dis from the axis to the force is the moment of given force about the axis. (Fig 97), $P \sin \alpha$ is the component parallel to the axis, and it has no tu effect. All the moment or turning is caused by the perpendicular component, and its value is $P \sin \alpha \times d$, being the perpendicular distance be the axis and the parallel plane ABC.

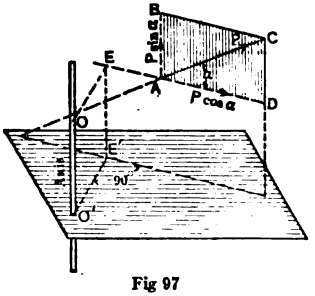


Fig 97

Principle of moments. Sum moments of any coplanar force sy about any point in their plane is eq the moment of their resultant about point. Thus (Fig 98), R is the res of P and Q, and $R \times r = P \times p - Q$ Sum of the moments of any force sy about an axis is equal to the mom their resultant about the axis. (Fig 99), R is the resultant of the 3 rectangular forces P, Q, and S. M of R about axis $X = S \times y - Q \times z$. P contributes nothing to the m sum, as it is parallel to the axis of moments. In such a case, counterclockwise moment is called positive and clockwise negative, the observer looking toward the origin O, from the positive ends of the axes. Thus, Q has positive moment about axis Z, but negative moment about axis X. Moment of a force passing through the origin of moments is zero. Moment of a force intersecting the axis of moments is zero.

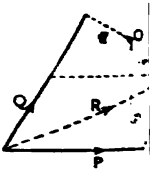


Fig 98

Couples. Two equal and parallel forces of opposite sense are called a couple. THE ARM OF THE COUPLE is the perpendicular tance between the lines of action of the forces. Moment of a couple is stant and independent of the origin of m it is equal to one of the forces times the the couple. Its sense is positive or ne according as rotational tendency is co clockwise or clockwise. Couples of equ ments, in the same or parallel planes, equivalent and may be replaced one by other. That is, a couple may be twist moved about in its plane, or transfer any parallel plane without altering the res motion of the body on which it acts.

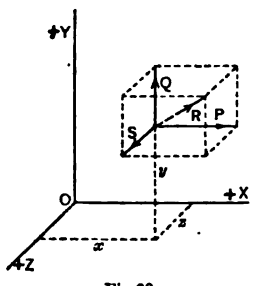


Fig 99

Resultant of any number of coplanar cou of couples in parallel planes is a couple. M ment and sense equal the algebraic sum of the ments of the component couples. A couple m represented by a vector. Length of the vector to scale represents the mag of the moment; it is drawn perpendicular to the plane of the couple from any and an arrow is placed on it to represent the way in which the couple would a right-hand screw to advance. Resultant of any number of couples (in oblique or allel planes) is a couple. The composition is effected thus: (*a*) Refer the couples to three rectangular coordinate axes. (*b*) Resolve each vector into components to the axes. (*c*) Take the sum of the components along the X, Y, and Z-axes

senting these sums by ΣC_x, ΣC_y, ΣC_z, and the resultant vector by C, its value is $\sqrt{\Sigma C_x^2 + \Sigma C_y^2 + \Sigma C_z^2}$. Its direction angles are

$$\phi_x = \cos^{-1}\frac{\Sigma C_x}{C};\ \phi_y = \cos^{-1}\frac{\Sigma C_y}{C};\ \phi_z = \cos^{-1}\frac{\Sigma C_z}{C}$$

orce may be resolved into a force acting at a chosen point and a couple. g 100, P_1 is the given force and O the n point. Through O apply a pair of forces, ite in sense, equal and parallel to P_1. As d P_3 balance, no change is produced in n of body due to the addition. P_1 and P_3 tute a couple of moment = $P \times a$, which is me as moment of P_1 about O; and P_2 is a just like P_1, but acting through the chosen O. Conversely, a force and a couple in a compound into a single force parallel to force, of same sense and magnitude, and ced that its moment about any point in action line of given force is equal ment of given couple.

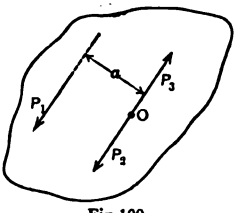


Fig 100

30. Resultants of Coplanar Nonconcurrent Forces

sultant of coplanar parallel forces may be (*a*) a single force or (*b*) a couple. **se** (*a*). Resultant is a single force. To find the resultant of the 4 par- orces shown in Fig 101:

phical solution. (*a*) Plat AB to represent magnitude and sense of ab; then BC resent magnitude and sense of bc; then CD and DE to represent the other two forces; (*b*) choose the pole O in any convenient position, and draw the rays AO, BO, CO, DO, and EO; (*c*) from any point on ab, draw strings parallel to AO and OB; from intersection of ob and bc draw oc parallel to OC until it intersects cd; from that point draw od to intersect de; then oe to intersect oa at K; (*d*) AE, from the beginning to the end of vector diagram, gives the magnitude and sense of the resultant = downward; its action line is ae, through K, parallel to AE, 5.3 ft to right of ab.

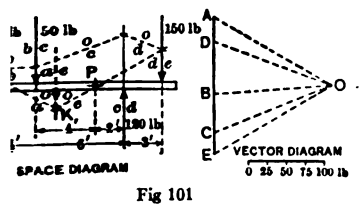


Fig 101

polygon formed on the space diagram is called a string or funicular polygon; its epresent the action lines of the components that replace the parallel forces. Thus, cting in the line bo, is replaced by BO = 140 lb, and OC = 160 lb acting in the lines l oc. AO and OE are components of the resultant AE. The object of the funic- olygon is to locate one point on the action line of the resultant.

ebraic solution. The magnitude and sense of the resultant are given by $R = \Sigma F$; ition is determined by use of the principle of moments, the moment of the result- ing equal to the algebraic sum of the moments of the forces. Hence arm = $\Sigma M \div R$. est to call upward forces positive, and distances to right of moment origin positive. lving example shown in Fig 101, tabulate the values as follows:

rce, lb	Arm from M	Moment about M	Arm from P	Moment about P
−100	0	0	−8	+800
− 50	+ 4	−200	−4	+200
+120	+10	+1 200	+2	+240
−150	+13	−1 950	+5	−750
= −180 lb	ΣM_M = −950 ft-lb		ΣM_P = +490 ft-lb	

$R = 180$ lb downward. Distance from $M = -950 \div -180 = +5.3$ ft. Dist from $P = +490 \div -180 = -2.7$ ft. Either of these distances locates the point Fig 101, and checks the graphical solution.

Case (b). $\Sigma F = 0$, and resultant is a couple.

Graphical solution. Consider the 4 parallel forces in Fig 102. The force po begins at A and ends at E. same point; hence the result not a single force (this is call closed force polygon). Const the funicular polygon as fo The first and last strings, ao oe, being parallel, do not inter The forces acting in those AO and OE, being equal opposite in sense, form a co Hence the resultant of the sy is a couple. The forces $AO = OE = 750$ lb The arm $MN = 3.67$ ft. The mom $750 \times 3.67 = 2\,752$ ft-lb. The sense, by inspection of the space diagram, is cl wise. The moment of the couple is the same, no matter what pole O is selected

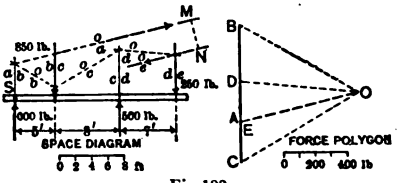


Fig 102

Algebraic solution of the same example (see table):

Forces, lb	Arm from S	Moment about S
+600	0	0
−850	+ 5	−4 250
+500	+13	+6 500
−250	+20	−5 000
$\Sigma F = 0$		$\Sigma M_s = -2\,750$ ft-l

Hence the resultant is a clockwise couple of moment equal to 2 750 ft-lb. Since couples of equal moment and sense in the same plane are equivalent, it is unnecessary to give more detail.

Special cases. Two parallel forces. Case (a). Forces alike in sense. Given the 2 forces P and Q, locate Fig 103. The resultant lies between these forces, and is equal in magnitude to P The action line of the re ant is found from the equa $a = (Q \times b) \div P$. Case Forces opposite in sense. the 2 forces P and Q as Fig 104. Resultant lies outside the forces P and Q, is adja to the larger force, and is in magnitude to $Q - P$. action line of the resultan found from the equation, $s = (Q \times b) \div P$. The sense of the resultant is the same that of larger force. In either case, the resultant divides the distance between forces into segments inversely proportional to the adjacent forces; or

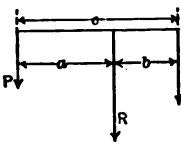


Fig 103

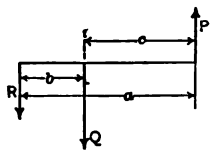


Fig 104

$$P \div b = Q \div a = R \div c.$$

Resultant of coplanar nonconcurrent forces. Case (a). Resultant equals a single force.

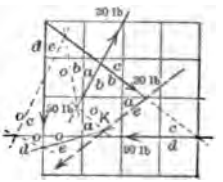


Fig 105

Graphical Solution. Construct a force polygon as if the forces were concurrent. The closing line gives the magnitude, sense, and angular direction of the resultant. The action line is found drawing a funicular polygon. Thus, to find the resultant of the 4 forces in Fig 10 Construct the force polygon $ABCDE$; (b) choose a convenient pole O, and draw rays; (c) start at any point on ab and draw the funicular polygon; (d) determine K

ection of the first and last strings. This is one point on the action line of the re-
nt. Resultant = AE = 67 lb, downward to the left, at angle $\theta = \tan^{-1}\frac{0.784}{1.0} =$
' to horizontal. Action line is ae, parallel to AE, through K.
ond graphical method. Use the triangle law to find the resultant of two forces.
he same law to find the resultant of this partial resultant and the third force.
nt until the final resultant is obtained.
ebraic solution. The resultant is found in
itude, sense, and angular direction as if the
were concurrent.

$$\sqrt{\overline{\Sigma F_x}^2 + \overline{\Sigma F_y}^2}, \quad \cos\theta = \frac{\Sigma F_x}{R}, \quad \sin\theta = \frac{\Sigma F_y}{R}$$

ich θ is the angle between R and the X-axis,
he principle of moments to locate the action
Choose a convenient origin of moments and
ute ΣM. The arm of the resultant = $\Sigma M \div R$,
perpendicular to the angular direction of R
above. R is so located with respect to the
that the sign of its moment is the same as that
f. As an example, find the resultant of the 3
in Fig 106.

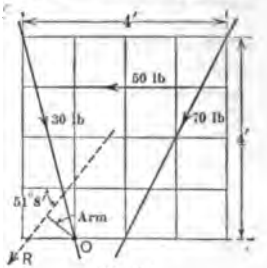


Fig 106

Force, lb	F_x, lb	F_y, lb	M_o, ft-lb
50	−50	0	+150
70	−31.3	−62.6	− 62.6
30	+ 7.3	−29.1	0
	$\Sigma F_x = -74$	$\Sigma F_y = -91.7$	$\Sigma M_o = 87.4$ ft-lb

$R = \sqrt{(74)^2 + (91.7)^2} = 118$ lb downward to the left.

ngle with X-axis = $\tan^{-1}\frac{91.7}{74} = 51° 8'$. Arm about $O = 87.4 \div 118 = 0.74$ ft.

s case, R is to the left of O because its moment must be = $\Sigma M_O = +87.4$ ft-lb
se (b). Resultant is a couple.

phical solution. Let forces N, P, Q, T, and S (Fig 107) represent a system of
the force polygon of which closes, and which reduces to the 2 components AO and
OF, acting in ao and of. Since these components are equal, opposite and parallel, the resultant is a couple. The arm of the couple is the perpendicular distance MN, between the first and last strings. The moment is the product of OA (according to force scale) and the arm MN (according to scale of space diagram). The sense of the couple is counterclockwise, as found by inspection of the space diagram.

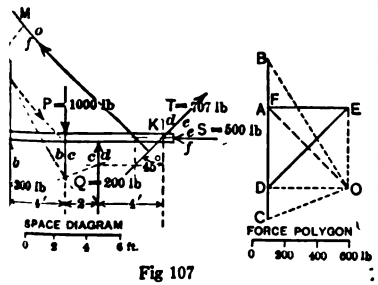


Fig 107

The moment = 710 × 3.1
= 2 200 ft-lb.

ebraic solution. Any couple the moment of which equals the algebraic sum of
oments of the given forces about any point may be regarded as the resultant. In
7, assume forces acting as indicated. The algebraic sum of the x and y compo-
is zero. Taking moments about point K,

$$\Sigma M_K = 1\,000 \times 6 - 300 \times 10 - 200 \times 4 = +2\,200 \text{ ft-lb}.$$

Hence, the resultant of the system is a couple, of moment equal to 2 200 ft-lb and counterclockwise sense.

31. Resultants of Noncoplanar Nonconcurrent Forces

Resultant of noncoplanar parallel forces. The possible resultants a single force or a couple.

Choose a set of 3 rectangular axes, with the Z-axis parallel to the forces, let the positions of the forces be specified by the x and y coordinates of action lines. As an example, consider the parallel forces shown in Fig 108 to be perpendicular to plane of the paper. Tabulate the computations as below:

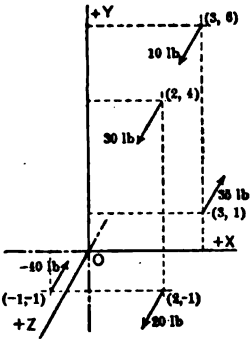


Fig 108

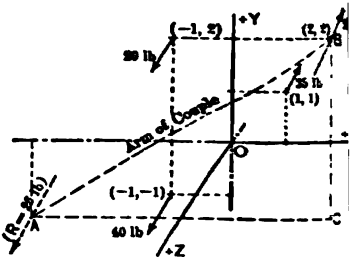


Fig 109

Force, lb	x, ft	y, ft	M_x or $F \times y$, ft-lb	M_y or $F \times x$,
+10	+3	+6	+ 60	+30
+30	+2	+4	+120	+60
+20	+2	−1	−20	+40
−40	−1	−1	+40	+40
−35	+3	+1	−35	−105
ΣF = −15 lb			ΣM_x = +165	ΣM_y = +65

$R = \Sigma F = -15$ lb, in the $-Z$ sense, or backward. $x_0 = \dfrac{\Sigma M_y}{\Sigma F} = \dfrac{+65}{-15} = -4.33$ ft. $y_0 = \dfrac{\Sigma M_x}{\Sigma F} = \dfrac{+165}{-15} = -11$ ft. Hence the coordinates of the action line of the result are x_0, y_0, or (−4.33 ft, −11 ft). Position of the resultant is not indicated in Fig In the above computation, the algebraic rules of signs for multiplication and division followed throughout, so that the values of x_0 and y_0 may be interpreted algebraically. This method agrees with the usual convention of signs for computing moments about the X-axis, but is the reverse of the usual convention for moments about the Y-axis.

If $\Sigma F = 0$, and if ΣM_x or ΣM_y does not equal zero, the resultant is a couple and not a force. The plane of the couple will be parallel to the Z-axis. It is necessary to determine the moment of the couple, its sense, and the inclination of its plane with the plane. Consider the 4 forces in Fig 109 as an example:

Force, lb	x, ft	y, ft	M_x, ft-lb	M_y, ft-lb
+20	−1	+2	+40	−20
+40	−1	−1	−40	−40
−35	+1	+1	−35	−35
−25	+2	+2	−50	−50
ΣF = 0			ΣM_x = −85	ΣM_y = −145

$\Sigma F = 0$, the resultant of all except one of the forces, with the omitted force, is e. Omitting the last force, $R = \Sigma F = +25$ lb. Its $x_0 = \frac{\Sigma M_y}{\Sigma F} = \frac{-95}{+25} = -3.8$ ft, $= \frac{\Sigma M_x}{\Sigma F} = \frac{-35}{+25} = -1.4$ ft. AB is the arm of the couple, and from the triangle s length is found to be 6.72 ft. The moment of the couple $= 25 \times 6.72 = 168$ BAC is angle its plane makes with horiz plane. Angle $BAC = \tan^{-1}\frac{3.4}{5.8} = 30° 21'$. the couple from plus end of Y-axis, its sense is observed to be counterclockwise.

ultant of noncoplanar nonconcurrent nonparallel forces is gener- single force acting at a chosen point and a couple not coplanar with the The magnitude, sense, and angular direction of the single force is the s if the forces were concurrent.

$$\sqrt{(\Sigma F_x)^2+(\Sigma F_y)^2+(\Sigma F_z)^2};\ \phi_x = \cos^{-1}\frac{\Sigma F_x}{R};\ \phi_y = \cos^{-1}\frac{\Sigma F_y}{R};\ \phi_z = \cos^{-1}\frac{\Sigma F_z}{R}$$

acts through the selected origin of reference. To determine the couple, te the sums of the moments of the forces about the coordinate axes of ce. These moment sums represent 3 couples, which are axial compo- f the resultant couple. Consider ΣM_x as a vector along the X-axis, ΣM_y ctor along Y, and ΣM_z along Z. The moment of the resultant couple $\sqrt{(\Sigma M_x)^2 + (\Sigma M_y)^2 + (\Sigma M_z)^2}$, and the direction angles of its vector are

$$\theta_x = \cos^{-1}\frac{\Sigma M_x}{C};\ \theta_y = \cos^{-1}\frac{\Sigma M_y}{C};\ \theta_z = \cos^{-1}\frac{\Sigma M_z}{C}$$

d C may be compounded into 2 forces which do not intersect. In a special a R might be zero or C might be zero. This means the resultant might uple only, or might be a single force.

32. Conditions of Equilibrium

dy is in equilibrium with respect to adjacent bodies if it remains at rest spect to them, or if it moves at constant velocity; that is, its state of does not change. A force system is in equilibrium if its resultant is Such a system causes no change in the state of motion of the body to t is applied. If a body is in equilibrium with respect to adjacent bodies, e external force system applied to it is in equilibrium. By this external stem is meant the pull of gravity and all the pulls or pushes exerted on ly by other bodies.

nding on the kind of force system involved, varying numbers of tests litions must be applied to prove that the system is in equilibrium, or resultant exists. In the usual problem, a body is known to be in equi- ; hence the system composed of all the external forces acting upon it e in equilibrium. In such case the tests are not needed to ascertain if ium exists; but they are used to set up relations involving unknown distances, or angles, and the unknown elements are then computed d they are not too numerous. Such computations are conducted either ically or graphically. (See following table.)

ial conditions of equilibrium. If 3 forces are in equilibrium they e coplanar, and must be concurrent or parallel; if concurrent, each proportional to the sine of the angle between the other two; if parallel, ce is proportional to the distance between the other two. If a force is in equilibrium, the resultant of any part must balance the resultant ther part.

Necessary and sufficient independent conditions of equilibrium for the vari force systems

	System	Algebraical		Graphical	
		No	Conditions	No	Conditions
Coplanar	Collinear	1	$\Sigma F = 0$	1	Force polygon
	Concurrent at point O	2	$\Sigma F_x = 0$, $\Sigma F_y = 0$, if the angle between x and y does not equal 180°; or, $\Sigma F_x = 0$, $\Sigma M_a = 0$, if x direction is not perpendicular to aO; or, $\Sigma M_a = 0$, $\Sigma M_b = 0$, if aOb is not a straight line	1	Force polygon c
	Parallel	2	$\Sigma F = 0$, $\Sigma M = 0$; or, $\Sigma M_a = 0$, $\Sigma M_b = 0$, if line ab is not parallel to forces	2	Force and fu lar polygons Last item first ar strings con
	Nonparallel nonconcurrent	3	$\Sigma F_x = 0$, $\Sigma F_y = 0$, $\Sigma M = 0$; or, $\Sigma F_x = 0$, $\Sigma M_a = 0$, $\Sigma M_b = 0$, if x is not perpendicular to ab; or, $\Sigma M_a = 0$, $\Sigma M_b = 0$, $\Sigma M_c = 0$, if abc is not a straight line		Force and fun polygons clos
Noncoplanar	Concurrent at point O	3	$\Sigma F_x = 0$, $\Sigma F_y = 0$, $\Sigma F_z = 0$; or, ΣF_s in every direction and ΣM about every axis = 0. Combinations of moment and resolution equations can be arranged, but are not common	2	Force polygon It is warped plan and must show Not com used
	Parallel	3	$\Sigma F_z = 0$, $\Sigma M_x = 0$, $\Sigma M_y = 0$, forces parallel to z-axis. Other combinations possible but not common		Not used
	Nonconcurrent nonparallel	6	$\Sigma F_x = 0$, $\Sigma F_y = 0$, $\Sigma F_z = 0$, $\Sigma M_x = 0$, $\Sigma M_y = 0$, $\Sigma M_z = 0$. ΣM about every axis = 0, and it is often convenient to employ more than three moment equations, instead of using so many resolution equations		The projecti system plane is librium. braical cal condi be used such proje tems

33. Equilibrium Problems

General method of solution. (*a*) Sketch or specify the free body in equilibrium, and indicate the complete external force system acting (*b*) Discuss the problem as to number and kinds of unknown quan see if a solution is possible. If the first system tried cannot be solved, the original body discussed (say some machine) into simpler parts, and

cuss them until one is found that permits of solution. (*c*) Apply the iate conditions of equilibrium and solve for the unknown forces, angles, nces. (*d*) Make use of the acquired information to solve other sys- there are any others.

olution equations are used, it is desirable each time to resolve perpen- to one of the unknown forces, so as to avoid the solution of simultaneous ns and to make the answers independent of each other. If moment are used, it is best to select them on the action lines of some of the un- forces. If moment axes are used they should be selected so as to inter- ne of the unknown forces.

rtain equilibrium problems often occur in practice, it is convenient to hem into typical problems, and to detail the method of solution of each.

cal problem I. A SYSTEM OF COPLANAR CONCURRENT FORCES is in ium and all but two are known; the action lines of these are known; l their magnitudes and senses. ALGEBRAIC SOLUTION. (*e*) Assume senses h unknown forces. (*c*) Write a pair of equilibrium equations involving nown forces, and solve for the two unknowns. A plus answer indicates se correctly assumed. A minus answer indicates incorrect assumption. CAL SOLUTION. (*f*) Letter the action lines of the wholly known forces, en the action lines of the two unknown forces. (*g*) Draw the force to the end of the last known vector, and close it by drawing parallels nknown forces through the ends of the first and last vectors, and mark rsection of these lines as the last vertex in the polygon. (*h*) The senses ad confluently from starting point back to same point, the last two vectors ting the two unknown forces in magnitude and sense.

ple. Two smooth cylinders rest upon a 30° plane and against a vertical wall as Fig 110. Determine all forces acting on each cylinder. (*a*) The forces involved lb, 200 lb, *P*, *Q*, *R*, and *S*, the last g normal to the surfaces of contact surfaces). (*b*) Consider the two as a single free body. The external tem is 100 lb, 200 lb, *P*, *R*, and *S* are internal). The system is non- nt, so does not come under typical I. Consider the large cylinder as a . The external force system is 100 and *S*. While this system is con- it can not be solved because there than two unknown quantities. Next the small cylinder as a free body. e system is 200 lb, *P* and *Q*, and pical problem I. ALGEBRAIC SOLU- hoose *X* and *Y* directions parallel

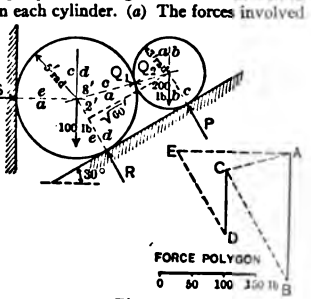


Fig 110

ndicular to the plane. By (*c*) and (*e*), $\Sigma F_x = 0 = Q_1 \frac{\sqrt{60}}{8} - 200 \sin 30°$. Hence

$= 103.3$ lb. $\Sigma F_y = 0 = P - 200 \cos 30° - 103.3 \times \frac{6}{8}$. Hence $P = 199$ lb.

ider the large cylinder as a free body. $Q_1 = Q_2 = 103.3$ lb. Use the same

directions. $\Sigma F_x = 0 = S \cos 30° - 103.3 \times \frac{\sqrt{60}}{8} - 100 \sin 30°$. Hence,

lb. $\Sigma F_y = 0 = R - 100 \cos 30° - 173.2 \sin 30° + 103.3 \times \frac{6}{8}$. Hence $R =$

GRAPHICAL SOLUTION. Discussions (*a*) and (*b*) are same as above. The free small cylinder. The force system is 200 lb, *ab*, *bc*, and *ca*, and the polygon le *ABC* (Fig 110). $BC = 199$ lb, $CA = 103.3$ lb. Next, consider the large cylin- body. The force system is *ac*, *cd*, *de* and *ea*. Plat the known forces *AC* and

CD. From *D* draw *DE* parallel to *de*, and from *A*, *AE* parallel to *ae*; these lines i sect at *E*. *DE* = 147.4 lb, and *EA* = 173.2 lb are the magnitudes and senses of the unknowns.

If the system is concurrent and all the forces are known except one, the two unk elements will be one angle and one magnitude and sense. Both may be determ writing the equilibrium equations, or by drawing the force polygon.

Typical problem II. A SYSTEM OF COPLANAR PARALLEL FORCES is in librium and all are known except two; the action lines of these two are k required their magnitudes and senses. ALGEBRAIC SOLUTION. (*e*) Assume s for both unknown forces. (*c*) Write a pair of equilibrium equations in the unknown forces and solve for the two unknowns. A plus answer in the sense correctly assumed. A minus answer indicates incorrect assu As a check on the results apply a third equilibrium equation. For pr of this type the algebraic solution is preferable to the graphical. GRAP SOLUTION. (*f*) Letter the action lines of the wholly known forces and the action lines of the two unknown forces. (*g*) On a line of indefinite parallel to the forces, often called the load line, form the force poly plotting vectors of the known forces continuously to end of last known The vector of the first unknown force extends from this point to so known point on the load line. The unknown point is to be found. struct a funicular polygon. Draw first string between first known for last-lettered unknown force. Continue funicular polygon to intersec last known string with first-lettered unknown force. Draw closing strin this point, to intersection of first string with last-lettered unknown force. a ray through pole parallel to closing string; its intersection with the k is the point sought. senses must read out from the starting po to the same point. T two vectors repres two unknown forces nitude and sense.

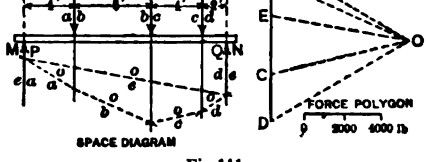


Fig 111

Example. A beam as shown in Fig 111 a ported at the points M Determine the reactions supports. (*b*) Consider th as a free body. The external force system consists of the forces 1 000 lb, 5 2 400 lb, *P*, and *Q*. This is a coplanar parallel system and is typical par ALGEBRAIC SOLUTION. (*c*) Assume senses for the reactions. By (*c*),

$$\Sigma M_M = 0 = -4 \times 1\,000 - 10 \times 5\,000 - 14 \times 2\,400 + 16\,Q = 0$$

Hence $Q = 5\,475$ lb; correct sense was assumed.

$$\Sigma M_N = 0 = 2 \times 2\,400 + 6 \times 5\,000 + 12 \times 1\,000 - 16\,P = 0$$

Hence $P = 2\,925$ lb; correct sense assumed. As a check, apply condition, $\Sigma F =$

$$\Sigma F = 0 = -5\,475 + 1\,000 + 5\,000 + 2\,400 - 2\,925 = 0$$

GRAPHICAL SOLUTION of the same problem involves the construction of a cl polygon and a closed funicular polygon. Draw vectors *AB*, *BC*, *CD* to re lb, 5 000 lb, 2 400 lb. Let *DE* represent *Q* and *EA* represent *P*. The problem point *E*. This is done with the aid of the funicular polygon. Draw the ra *OC*, and *OD*. Start on the action line of *P* and draw the string polygon. T line is *oe*. Draw the ray *OE* parallel to *oe*. The unknown reactions are *D*

(**Note.** — This principle can not be applied to beams having more than tw support. Such problems require special treatment.)

Typical problem III. A SYSTEM OF COPLANAR NONCONCURRENT PARALLEL FORCES is in equilibrium, and all are known except two; th

f one of these and a point in the action line of the other are known. De-
ne the magnitude and sense of the one, and the magnitude, sense, and
ir direction of the other. ALGEBRAIC SOLUTION. (*e*) Assume sense for
nknown force. (*c*) Write a moment equation, center of moments be-
given application point, to find first unknown force. (*e'*) Assume sense
ngular direction for second unknown force. (*c*) Write a pair of equilib-
equations involving the magnitude and angular direction of the second
wn force and solve for the two unknown quantities. A plus answer for
agnitude indicates the sense correctly assumed. A plus answer for the
ir direction indicates it correctly assumed. A minus answer for either of
antities indicates incorrect assumption for the one involved. Check re-
y means of an additional equilibrium equation. GRAPHICAL SOLUTION.
tter the action lines of the wholly known forces, then the action line of
st unknown, and finally the force passing through the given application
(*g*) Draw the force polygon for the known forces. Through the end of
t vector draw a line parallel to the action line of the first unknown force.
: a pole and draw the rays. (*i*) Construct a funicular polygon, passing
st string through the application point of the second unknown force,
aking the last, or closing, string, pass through the same point. Draw
through the pole, parallel to closing string; its intersection with the
drawn parallel to first unknown is last vertex in the force polygon.
this point, draw the closing line of the force polygon. (*h*) The senses
ead confluently from start-
at back to same point. The
o vectors represent the two
wn forces.

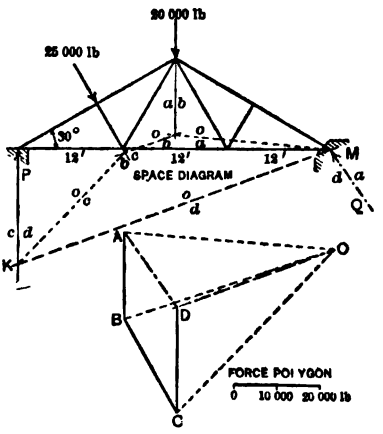


Fig 112

ple. A roof truss is loaded as
12. The left end of the truss
a smooth horiz support. The
d is secured to a wall by means
Determine the reactions. (*a*)
rnal forces acting on the truss
given loads, the left reaction
cal, on account of the smooth
), and the right reaction Q (in-
through point M). (*b*) The
n quantities are the reactions
). This is typical problem III.
AIC SOLUTION. (*e*) Assume P
By (*c*),

$= 0 = 20\,000 \times 18 + 25\,000 \times$
$30° - 36P$

$P = 24\,430$ lb; correct sense

ssume Q upward to the left at
vith horizontal. By (*c*),

$0 = 25\,000 \sin 30° - Q \cos\theta, \quad \Sigma F_y = 0 = 25\,000 \cos 30° + 20\,000 - 24\,430 - Q \sin\theta$

simultaneously, $Q = 21\,300$ lb, and $\theta = 54°$. Sense and direction were cor-
sumed, hence Q acts upward to the left at 54° to the horiz. GRAPHICAL SOLU-
ab and *bc* are the action lines of the given loads, *cd* of the reaction P and *da*
action Q. (*g*) Draw the vectors AB and BC, and a line through C, parallel to
ose a pole and draw the rays. (*i*) Construct the funicular polygon, drawing
gh M, and draw closing string *od* from K to M. Draw OD through O to inter-
at D. Draw DA. (*h*) Vectors CD and DA represent the two unknown forces,
,30 lb and $Q = 21\,300$ lb. The action line of Q is *da*, making angle with
54°.

Typical problem IV. A SYSTEM OF COPLANAR NONCONCURRENT N PARALLEL FORCES is in equilibrium and all except three are wholly known; action lines of these three are known; required their magnitudes and sen ALGEBRAIC SOLUTION. (*e*) Assume senses for the unknown forces. (*c*) determine first unknown force, write a moment equation, center of mom at intersection of action lines of other unknown forces. (*c*) Write a pai equilibrium equations involving the other two unknowns and solve for th Check results by means of an additional equilibrium equation. GRAPH SOLUTION. Assume any two of the unknown forces replaced by their resul at the point where their action lines intersect. This resultant, the third known force, and the known forces are in equilibrium and correspond to ty problem III. Proceed as specified in (*f*), (*g*), (*i*) and (*k*) for typical problem to determine the resul and the third unknown f (*j*) Resolve the resultan the selected two unkn forces into component allel to their action lines, determining their mu tudes and senses.

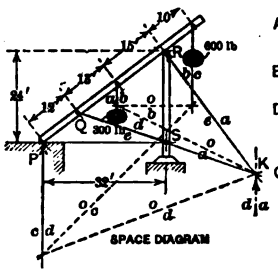


Fig 113

Example. The bar *PR* (Fig 113), resting against a wa floor at *P* and against a sm post at *R*, is held by an ins cord *QS* and carries two l Neglect the weight of the and determine the forces a on the bar at *P* and *R*, and the tension in the cord *QS*. (*a*) The external forces a on the bar are the given loads, the reaction of the support at *P* (vertical on accou smooth surface), the reaction of the post at *R* (normal to the bar on account of sm surface), and the pull of the cord at *Q* (direction is *QS*). (*b*) The unknown force the reactions *P* and *R*, and the tension *Q*. This is typical problem IV. ALGEBRAIC TION. (*e*) Assume *P* upward. By (*c*),

$$\Sigma M_K = 0 = 300 \times 35.6 + 600 \times 15.6 - P \times 55.6$$

Hence, $P = 360$ lb; correct sense assumed. Choose *X* and *Y* directions paralle perpendicular to the bar. (*e*) Assume *R* upward to the left on bar, *Q* downw right. By (*c*),

$$\Sigma F_x = 0 = 300 \times \tfrac{3}{5} + 600 \times \tfrac{3}{5} - 360 \times \tfrac{3}{5} - Q\tfrac{13.6}{23.5}$$

Hence $Q = 560$ lb; correct sense assumed.

$$\Sigma F_y = 0 = 300 \times \tfrac{4}{5} + 600 \times \tfrac{4}{5} + 560 \times \tfrac{19.2}{23.5} - 360 \times \tfrac{4}{5} - R$$

Hence $R = 890$ lb; correct sense assumed.

As a check, apply condition, $\Sigma M_Q = 0$.

$$\Sigma M_Q = 890 \times 28 - 360 \times \tfrac{4}{5} \times 12 - 300 \times \tfrac{4}{5} \times 13 - 600 \times \tfrac{4}{5} \times 38 = 24\,900 - 22\,$$

GRAPHICAL SOLUTION. Assume *Q* and *R* replaced by their resultant. Its act passes through point *K*. (*f*) *ab* and *bc* are the action lines of the given loads, reaction *P*, and *da* of the resultant of *Q* and *R*. (*g*), (*i*), and (*k*). Draw fu funicular polygons to determine the unknown vectors *CD* and *DA*. *CD* re reaction $P = 360$ lb. (*j*) Resolve *DA* into components *DE* and *EA*, parallel t lines *de* and *ea* of *Q* and *R*. (*k*) Vectors *DE* and *EA* represent the other tw forces, $Q = 560$ lb, and $R = 890$ lb.

General noncoplanar system. Example. The crane (Fig 114), is supported socket at the foot of the post at *D*, is kept from overturning by the back-stays *A*

and carries a load of 600 lb. , F, G, D, are in the vertical XY-.) Determine the axial compo- of the reaction on the post at D he tensions in the back-stays. (b) external forces acting on the post e load, the reaction at D, and the ns in the back-stays at A. The n of forces is noncoplanar, non- rrent, nonparallel. Moment equa- are the most convenient to apply is solution. By (c),

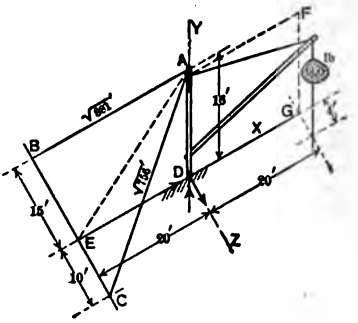


Fig 114

$$\Sigma M_y = 0; \quad D_y = \frac{600 \times 40}{20} = 1\,200 \text{ lb.}$$

$$\Sigma M_z = 0; \quad D_z = \frac{600 \times 20}{16} = 750 \text{ lb.}$$

$$\Sigma M_x = 0; \quad D_x = \frac{600 \times 4}{16} = 150 \text{ lb.}$$

$$\Sigma M_{x_C} = 0 = AB \times \frac{16}{\sqrt{881}} \times 25 - 1\,200 \times 10 + 600 \times 6, \qquad AB = 624 \text{ lb.}$$

$$\Sigma M_{x_B} = 0 = AC \times \frac{16}{\sqrt{756}} \times 25 + 600 \times 19 - 1\,200 \times 15; \qquad AC = 454 \text{ lb.}$$

senses of the components of the reaction at D are shown in Fig 114.

FRICTION

34. Definitions

rfectly smooth surface is one that offers no resistance to the sliding of y upon it. The force exerted by such a surface is normal to the surface tact. If the surface is rough, the resultant thrust of the surface is oblique, called the TOTAL REACTION. The tangential component of the total re- is called FRICTION and the normal component is called NORMAL REACTION. on is a passive resistance, and is developed as it is needed to prevent n. For friction to exist there must be a tendency for a body to slip over ace, due to the fact that the resultant of all the applied forces acting on dy (omitting total reaction of the surface) has a component parallel to rface. If the surface is smooth, such a component causes motion of the If the surface is rough, it exerts a frictional force upon the body which lizes the tendency to move. For any given conditions, there is a limit amount of friction, reached when slipping impends, called LIMITING ON. When the limiting friction is attained the angle between the total n and the normal to the surface is the ANGLE OF FRICTION. In case a no slipping, friction is called STATIC, and may have any magnitude from o the limiting friction, depending upon the tendency to move. If this cy to move exceeds limiting friction, the body slips and KINETIC FRICTION pposes motion; its value is less than limiting static friction.

fficient of friction is the ratio of the limiting friction to the correspond- rmal reaction. If the angle of friction be ϕ, the normal reaction N, the g friction F', and the coeff of friction f, then $f = \tan \phi = F' \div N$. If a e placed upon an inclined plane, the angle of inclination for which slip- f the body impends is the ANGLE OF REPOSE for the two rubbing surfaces qual to the angle of friction.

35. Friction Cone

Consider a body, resting on a rough plane, to be acted upon by several f
(Fig 115). Draw the normal OP to the surface at the point where the re
of all the applied forces, except the total reaction, cuts the surface; the
OQ, making the angle ϕ with the normal equal to the angle of friction.
cone generated by revolving OQ about the axis OP, is the CONE OF FRICT
The body will not slip if the resultant of all applied forces lies within this
it will slip if the resultant lies outside.

Assuming the data shown in Fig 115, a graphic solution is as follows: Draw a
polygon for the applied forces, $ABCD$. The resultant is AD, in magnitude and direc

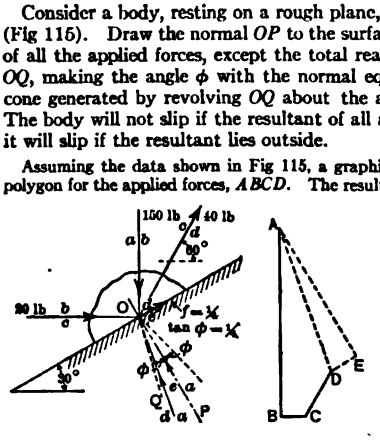


Fig 115

Its action line od falls inside the
of friction, hence slipping will not
Total reaction of the plane is DA,
site in sense to the resultant. Its
ponents, parallel and normal
surface, are DE and EA. Fro
the body is DE = 23 lb, upward
the surface, and the normal re
EA = 120 lb, upward normal
surface.

From the algebraic viewpoint,
normal reaction on the surface is

$$N = 150 \cos 30° + 20 \sin 30° - 40 \sin 30° = 120 \text{ lb}$$

Available or limiting friction = $\frac{1}{4} \times 120 = 30$ lb. Sum of the components parallel to the plane = $150 \sin 30° - 20 \cos 30° - 40 \cos 30° = 23$ lb, downward. If needed, the surface can supply 30 lb of friction; it actually does supply 23 lb acting upwards on the body to preserve equilibrium.

36. Friction Circle

The reaction of a bearing on a journal is usually assumed to pass through the center of the journal. If the bearing is worn (as shown much exaggerated in Fig 116, for the journal A), the action line of the reaction still passes through center A if the surfaces are smooth. Otherwise, when slipping impends, the action line makes the friction angle ϕ with the normal AD to the contact surfaces on the line of contact between the journal and the bearing. If motion impends for clockwise rotation of the journal, DB is the action line; if for counterclockwise rotation, it is EC. If r be the journal radius, the perpendicular to the action lines from the journal center is AB or $AC = r \sin \phi$. For a given journal and bearing, this value is constant and may be represented by the radius of a circle with center at A. This is the FRICTION CIRCLE, and the bearing reaction is always tangent to it if slipping impends. For practical conditions, angle ϕ is so small that $r \tan \phi = r \sin \phi$ (approx). Radius of the friction circle may be taken as $r \times f$, nearly.

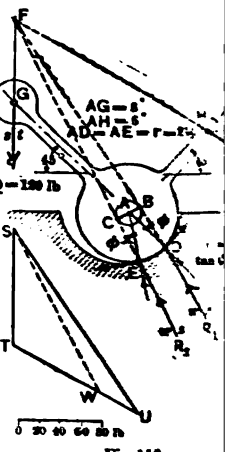


Fig 116

In a given problem, it is often impossible to observe D, the line of contact between

al and bearing, but the action line of the bearing reaction must be tangent e friction circle. Other known conditions may fix another point in this i line so that its position is readily found.

Fig 116, with data as given, find what force P is needed to overcome Q (motion im- ig for clockwise rotation). The journal is under the action of 3 forces, P, Q, and R, rrent at F. As R must be tangent to the friction circle, its action line is either FD . The correct position is determined by resolving R into tangential and normal nents, and making the choice so that friction opposes impending motion. The olygon for the 3 forces is STU; then $R_1 = US = 226$ lb, and $P = TU = 142$ lb. greatest force P which Q can overcome (motion impending for counterclockwise n), is found from the triangle STW, corresponding to R_2; then $R_2 = WS = 188$ lb, $= TW = 96$ lb.

37. Coefficients of Friction

fficients of static friction depend upon character of the surfaces, kind iterial and lubricant used. Recent experiments prove that the coeff of friction is not independent of the normal pressure on the contact surfaces, the time of contact between the surfaces.

Coefficients of Static Friction (Compiled from Rankine)

nasonry and brick-		Masonry on dry clay.....	0.51
k.....	0.6 to 0.7	" " moist clay....	0.33
ry and brickwork		Earth on earth.....	0.25 to 1.00
damp mortar.....	0.74	" " " dry sand,	
r on stone.... about	0.4	clay, and mixed earth...	0.38 to 0.75
n stone.....	0.3 to 0.7	Earth on earth, damp clay.	1.00
r on timber	0.2 to 0.5	" " " wet clay..	0.31
" metals.....	0.2 to 0.6	" " " shingle and	
on metals.....	0.15 to 0.25	gravel.....	0.81 to 1.11

fficients of kinetic friction depend upon the velocity of the body in n, and particularly upon the kind of lubricant. If a lubricant is used it greater influence upon the coeff of friction than has the kind of material bodies.

Coefficients of Kinetic Friction (Rough averages, compiled from Rankine)

on wood, dry.....	0.25 to 0.5.	Leather on metals, oily....	0.15
" " soaped....	0.20 to 0.04	Metals on metals, dry.....	0.15 to 0.20
on oak, dry.....	0.50 to 0.60	" " " wet.....	0.30
" " wet.....	0.24 to 0.26	Smooth surfaces, occasion-	
" " soaped....	0.20	ally greased.....	0.07 to 0.08
" elm, dry.....	0.20 to 0.25	Smooth surfaces, continu-	
on oak, dry.....	0.53	ously greased.....	0.05
" " wet.....	0.33	Smooth surfaces, best re-	
r on oak.....	0.27 to 0.38	sults.....	0.05 to 0.036
" metals, dry....	0.56	Bronze on lignum vitæ,	
" " wet....	0.36	constantly wet.....	0.05
" " greasy.	0.23		

38. Belt or Coil Friction

band is placed about a rough cylinder and a tensile force applied to each hese forces may be very unequal without causing slipping of the band the cylinder, and are unequal even if slipping occurs. The band may be rope, or brake band, while the cylinder may be a pulley or sheave, post, n, or similar device.

T_1 be the larger of the forces acting upon the band, T_2 the smaller one, correspond- the tensions in the tight and loose sides of a belt; and W the friction between band

and cylinder. Then, $W = T_1 - T_2$. Let the arc covered by the band subtend an α at the center of the cylinder, α radians = (α degrees $\times \pi$) ÷ 180; e (= 2.718) the base of the Naperian system of logarithms, and f the coeff of friction, care being to select a suitable value for either static or kinetic friction, depending on whether not slipping occurs. Then, $T_1 = T_2 e^{f\alpha} = T_2 10^{\frac{\pi}{180}(0.434)\, f\alpha^\circ}$. The first α is expressed in radians, the last α in degrees. For belting, the coeff of friction is often taken as

Values of $e^{f\alpha}$ (slipping impending)

Angle of contact in decimal parts of circum = $\alpha \div 2\pi$ radians	Values of f (coeff of friction)								
	0.10	0.15	0.20	0.25	0.30	0.35	0.40	0.45	
0.1	1.06	1.1	1.13	1.17	1.21	1.25	1.29	1.33	
0.2	1.13	1.21	1.29	1.37	1.46	1.55	1.65	1 76	
0.3	1.21	1.32	1.45	1.60	1.76	1.93	2 13	2.34	
0.4	1.29	1.46	1.65	1.87	2.12	2.41	2.73	3 10	
0.425	1.31	1.49	1.70	1.95	2.23	2.55	2.91	3 33	
0.45	1.33	1.53	1.76	2.03	2.34	2.69	3.10	3.57	
0.475	1.35	1.56	1.82	2.11	2.45	2.84	3.30	3.83	
0.5	1.37	1.60	1.87	2.19	2.57	3.00	3.51	4 11	
0.525	1.39	1.64	1.93	2.28	2.69	3.17	3.74	4 41	
0.55	1.41	1.68	2.00	2.37	2.82	3.35	3.98	4 74	
0.6	1.46	1.76	2.13	2.57	3.10	3.74	4.52	5 45	
0.7	1.52	1.93	2.41	3.00	3.74	4.66	5 81	7.21	
0.8	1.65	2.13	2.73	3.51	4.52	5.81	7.47	9.60	
0.9	1.76	2.34	3.10	4.11	5.45	7.24	9.60	12 74	
1.0	1.87	2.57	3.51	4.81	6.59	9.02	12.35	16 90	
1.5	2.57	4.11	6.59	10.55	16.90	27.08	43.38	69.49	
2.0	3.51	6.59	12.35	23.14	43.38	81.31	152.40	285 68	
2.5	4.81	10.55	23.14	50.75	111.32	244.15	535.49	1 174 5	
3.0	6.59	16.90	43.38	111.32	285.68	733.14	1 881.5	4 836 5	
3.5	9.02	27.08	81.31	244.15	733.14	2 199.90	6 610.7	19 851	
4.0	12.35	43.38	152.40	535.49	1 881.5	6 610.7	23 227	81 610	

CENTERS OF GRAVITY

39. Centroid and Center of Gravity

Centroid of a system of parallel forces with fixed application points is the point through which their resultant always passes, no matter how the forces may be turned (but still remaining parallel). The force of gravitation acting on the particles of a body constitutes a system of forces practically parallel, and the centroid of these forces is the CENTER OF GRAVITY of the body. Referring the application points of such a force system to a set of coordinates, the coordinates of the centroid, or center of gravity, are given by:

$$\bar{x} = (\Sigma F \cdot x) \div \Sigma F, \quad \bar{y} = (\Sigma F \cdot y) \div \Sigma F, \quad \bar{z} = (\Sigma F \cdot z) \div \Sigma F$$

In these equations F represents one force (or the weight of one particle), x the distance of its application point from the YZ-plane.

If the body is composed of parts of known weight, and the positions of the centers of gravity are known, the coordinates of the center of gravity of the whole body are:

$$\left.\begin{aligned} \bar{x} &= \frac{w_1\bar{x}_1 + w_2\bar{x}_2 + w_3\bar{x}_3 + \cdots}{w_1 + w_2 + w_3 + \cdots} \\ \bar{y} &= \frac{w_1\bar{y}_1 + w_2\bar{y}_2 + w_3\bar{y}_3 + \cdots}{w_1 + w_2 + w_3 + \cdots} \\ \bar{z} &= \frac{w_1\bar{z}_1 + w_2\bar{z}_2 + w_3\bar{z}_3 + \cdots}{w_1 + w_2 + w_3 + \cdots} \end{aligned}\right\} \quad (A)$$

ich w_1 represents the weight of one part and $\bar{x}_1$, $\bar{y}_1$, $\bar{z}_1$, are the coordinates of nter of gravity. Any one of the above expressions may be read as, "the of the moments divided by the sum of the forces."

in formulas (*A*) the weights are replaced by volumes, areas, surfaces, or hs of lines, they determine the coordinates of the centroids, or centers of ty, of the volumes, areas, surfaces, or lines. These centroids are often l the centers of gravity of the corresponding magnitudes.

nter of gravity of part of a body remaining after certain parts have taken away is determined by the rule: The moment of the remainder of a with respect to any plane, equals the moment of the whole body minus oments of the parts taken away.

40. Formulas

ntroids of lines and areas. CIRCULAR ARC (Fig 117). Centroid is e axis of symmetry; its distance from the center is $\bar{x} = \frac{r \times c}{s} = \frac{r \sin \alpha}{\alpha}$, ig in radians. For a semicircle, $\bar{x} = 2r \div \pi$; for a quadrant, $\bar{x} = 2r\sqrt{2} \div \pi$. IGLE. Centroid of a triangle is at the intersection of the medians; its

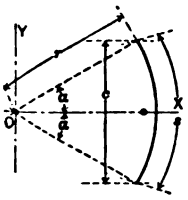


Fig 117

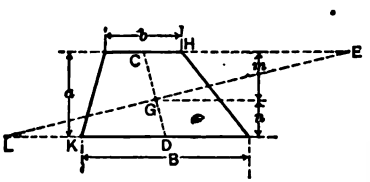


Fig 118

ndicular distance from any side equals one-third the altitude measured that side. TRAPEZOID (Fig 118). Centroid lies on the median *CD* (join- le middle of the parallel sides), at a point *G* located by following equations:

$$m = \frac{a\,(2B + b)}{3\,(B + b)}; \quad n = \frac{a\,(B + 2b)}{3\,(B + b)}$$

ocate the centroid graphically, make $HE = B$, $= b$. Draw *LE*, and the median *CD*; they in- ct at *G*, the centroid. QUADRILATERAL (Fig . Divide each side of the quadrilateral *ABCD* thirds, and through the third points draw *ef*, m, and *no*. These lines form a parallelogram, liagonals of which intersect at *G*, the centroid of quadrilateral. SECTOR OF A CIRCLE (Fig 120). roid is on the axis of symmetry; its distance the center is $\bar{x} = (2rc) \div (3s)$. For a semicircle, $4r \div 3\pi$. For a quadrant $\bar{x} = 4r\sqrt{2} \div 3\pi$; and

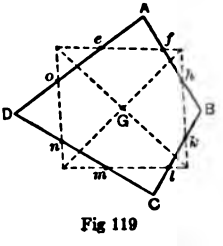


Fig 119

the distance from each bounding radius = $4r \div 3\pi$. SEGMENT OF A CIR (Fig 121). Centroid is on the axis of symmetry; its distance from the ce is $\bar{x} = (c^3 \div 12A)$, in which A is the area of the segment = $\frac{r^2}{2}(2\alpha - \sin$ α being in radians.

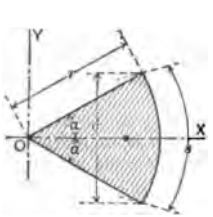


Fig 120

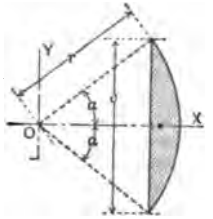


Fig 121

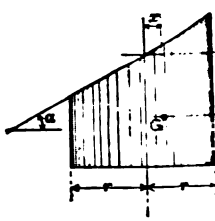


Fig 122

Centers of gravity of volumes. RIGHT CIRCULAR CYLINDER (Fig 1 Base is normal to the axis; top makes an angle α with base; mean height radius of base = r. If $\bar{y}$ is the distance of the center of gravity G from base, and $\bar{x}$ is the distance to the right of the vertical center line,

$$\bar{x} = \frac{r^2 \tan \alpha}{4h}; \text{ and } \bar{y} = \frac{h}{2} + \frac{r^2 \tan^2 \alpha}{8h}$$

FRUSTUM OF A CIRCULAR CONE. Let R = radius of the larger base and r ra of the smaller base; a = the altitude. Distance of the centroid of the cu surface from large base is $\frac{a(R+2r)}{3(R+r)}$ and from the smaller base $\frac{a(2R-}{3(R-}$ Distance from the center of gravity of the solid frustum to larger bas $\frac{a(R^2+2Rr+3r^2)}{4(R^2+Rr+r^2)}$. CONE AND PYRAMID. Centroid of the surface (base cluded) is on a line joining the apex with the centroid of the perimeter o base, at a distance of two-thirds its length from the apex. Center of gravi the solid cone or pyramid is on the line joining the apex with the centroid o base, at a distance of three-fourths its length from the apex. FRUSTUM A PYRAMID. If the frustum has regular bases, let R and r be lengths of the of the larger and smaller bases, and h the altitude. Distance of the centro the surface (bases excluded) from the larger base is $\frac{h(R+2r)}{3(R+r)}$. For any let A and a be the areas of large and small bases and h the altitude; the dis of center of gravity from the larger base is $\frac{h(A+2\sqrt{Aa}+3a)}{4(A+\sqrt{Aa}+a)}$. Sim Centroid of the surface of any zone of a sphere is midway between the Distance of the center of gravity of a solid segment from the base is $\frac{h(4r-}{4(3r-}$ where r = radius of the sphere, and h = height of the segment. For a hemis this distance is $3r \div 8$. Distance of center of gravity of a solid secto center of the sphere = $\frac{3}{8}(1+\cos\alpha)r = \frac{3}{8}(2r-h)$, where α = half the angle tended by the sector. ELLIPSOID. Choose 3 coordinate axes x, y, and z; let and c denote the semi-lengths of the corresponding axes of the ellipsoid. coordinates of the center of gravity of one octant of the solid are $\bar{x} = 3a \div$ $\bar{y} = 3b \div 8$, and $\bar{z} = 3c \div 8$.

MOMENTS OF INERTIA OF AREAS AND MASSES

41. Areas

oment of inertia of a plane area with respect to a given line is the sum of products obtained by multiplying each element of area by the square of its nce from the line. The moment is called polar if the axis is perpendicular e plane area. The common symbol for moment of inertia is I; and a cript denotes the axis to which it refers. Let I = the moment of inertia t any axis; A the area of the figure and r the distance of any element of from the given axis, then $I = \int r^2\,dA$. RADIUS OF GYRATION k, of any figure, is the square root of the moment of inertia divided by the area of igure, thus $k = \sqrt{\frac{I}{A}}$, or $I = Ak^2$. PARALLEL AXIS THEOREM. $I = \bar{I} + Ad^2$, hich $\bar{I}$ is the moment of inertia with respect to an axis through the centroid I is moment of inertia about a parallel axis at distance d.

roduct of inertia J, of an area with respect to a pair of rectangular axes, e sum of the products obtained by multiplying each element of area by coordinates. Thus $J_{xy} = \int xy\,dA$. Product of inertia may be positive, tive, or zero, depending on the distribution of the area with respect to the . Moment of inertia is always positive and never zero. PARALLEL AXIS REM. $J = \bar{J} + A\bar{x}\bar{y}$, in which $\bar{J}$ is product of inertia of area A with respect to a pair entroidal axes, J is the product of inertia respect to a set of parallel axes, and $\bar{x}$ and e the distances from the parallel axes to the roid of the area. For a rectangle, $J_{xy} = A^2 +$ or a triangle, $J_{xy} = A^2 + 8$ (Fig 123). J is for an axis of symmetry and any line perlicular to that axis. The customary unit for both moment and product of ia is biquadratic inches = in^4. The principal axes of inertia at a point are e about which the moments of inertia are the greatest and the least; these will always be rectangular and the product of inertia for them is zero.

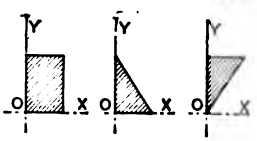


Fig 123

42. Oblique Axes and Least Radius of Gyration

blique axes. Let I_x, I_y, and J_{xy} represent moments and products of inertia a pair of rectangular axes at a point of an area, I_u the moment of inertia for any oblique axis through that point and α its angle with the X axis; then, $I_u = I_x \cos^2\alpha + I_y \sin^2\alpha - J_{xy} \sin 2\alpha$. For principal axes, $\tan 2\alpha = 2J_{xy} + (I_y - I_x)$.

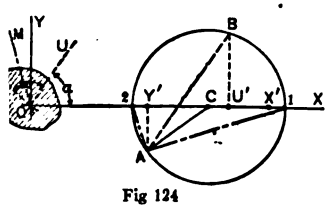


Fig 124

The equation for I_u is interpreted by the inertia circle as follows: Suppose I_x, I_y, and J_{xy} given for the shaded area in Fig 124. To convenient scale, plat OX' and OY' to esent I_x and I_y, and $Y'A$ to represent J_{xy} (downward if negative and up- l if positive). Center C is midway between X' and Y'. With CA a

radius, describe inertia circle. To find I_u, draw chord AB parallel to OU; draw perpendicular BU'. OU' (to scale) $= I_u$, and BU' (to scale) $= J_{uv}$. OM, parallel to A 2, is axis of least I; and a parallel to A 1, through is axis of greatest I. O 2 (to scale) is the value of least $I = I_2$; and O 1, v of greatest $I = I_1$. Least radius of gyration is measured perpendicular to OM and $= \sqrt{I_2 \div \text{area}}$. If principal moments of inertia (greatest and least a point are equal, ($I_1 = I_2$), then I for any oblique axis through the point same value; that is, $I_u = I_1 = I_2$.

43. Table of Plane Figures

I = Moment of Inertia. k = Radius of Gyration.

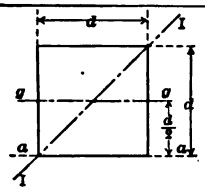


Fig 125. Square

$$I_g = \frac{d^4}{12} = I_1 \qquad I_a = \frac{d^4}{3}$$

$$k_g = \frac{d}{\sqrt{12}} = 0.289\,d = k_1 \qquad k_a = \frac{d}{\sqrt{3}} = 0.577$$

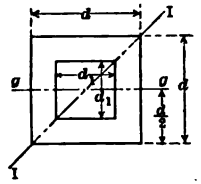


Fig 126. Hollow Square

$$I_g = \frac{d^4 - d_1^4}{12} = I_1$$

$$k_g = \sqrt{\frac{d^2 + d_1^2}{12}} = k_1$$

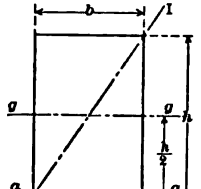


Fig 127. Rectangle

$$I_g = \frac{bh^3}{12} \qquad I_a = \frac{bh^3}{3} \qquad I_1 = \frac{bh^3}{6\,(b^2 + h^2}$$

$$k_g = \frac{h}{\sqrt{12}} = 0.289\,h \qquad k_a = \frac{h}{\sqrt{3}} = 0.577\,h$$

$$k_1 = \frac{bh}{\sqrt{6\,(b^2 + h^2)}}$$

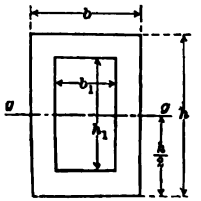


Fig 128. Hollow Rectangle

$$I_g = \frac{bh^3 - b_1h_1^3}{12}$$

$$k_g = \sqrt{\frac{bh^3 - b_1h_1^3}{12\,(bh - b_1h_1)}}$$

Table of Plane Figures (*Continued*)

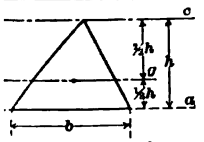


Fig 129. Triangle

$$I_g = \frac{bh^3}{36} \qquad I_a = \frac{bh^3}{12} \qquad I_c = \frac{bh^3}{4}$$

$$k_g = \frac{h}{\sqrt{18}} = 0.236\,h \qquad k_a = \frac{h}{\sqrt{6}} = 0.408\,h$$

$$k_c = \frac{h}{\sqrt{2}} = 0.707\,h$$

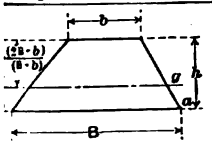
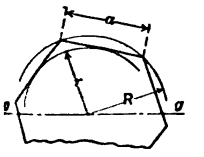


Fig 130. Trapezoid

$$I_g = \frac{h^3\,(B^2 + 4\,Bb + b^2)}{36\,(B + b)} \qquad I_a = \frac{h^3\,(B + 3\,b)}{12}$$

$$k_g = \frac{h}{6\,(B + b)}\sqrt{2\,(B^2 + 4\,Bb + b^2)}$$

$$k_a = \frac{h}{\sqrt{6}}\sqrt{\frac{(B + 3\,b)}{(B + b)}}$$



g 131. Any Complete Regular Polygon

A = Area of polygon.
g–g is any axis through center and in plane of polygon.
Axis p–p is perpendicular to plane of polygon at center.

$$I_g = \frac{A}{24}\,(6\,R^2 - a^2) = \frac{A}{48}\,(12\,r^2 + a^2)$$

$$I_p = \frac{A}{12}\,(6\,R^2 - a^2) = \frac{A}{24}\,(12\,r^2 + a^2)$$

$$k_g = \sqrt{\frac{(6\,R^2 - a^2)}{24}} = \sqrt{\frac{(12\,r^2 + a^2)}{48}}$$

$$k_p = \sqrt{\frac{(6\,R^2 - a^2)}{12}} = \sqrt{\frac{(12\,r^2 + a^2)}{24}}$$

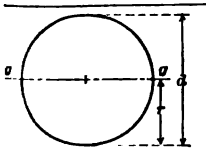


Fig 132. Circle

Axis p–p is perpendicular to circle at center.

$$I_g = \frac{\pi d^4}{64} = \frac{\pi r^4}{4} \qquad I_p = \frac{\pi d^4}{32} = \frac{\pi r^4}{2}$$

$$k_g = \frac{d}{4} = \frac{r}{2} \qquad k_p = \frac{d}{\sqrt{8}} = \frac{r}{\sqrt{2}}$$

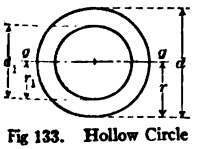


Fig 133. Hollow Circle

Axis p–p is perpendicular to hollow circle at center.

$$I_g = \frac{\pi}{64}\,(d^4 - d_1^4) = \frac{\pi}{4}\,(r^4 - r_1^4)$$

$$I_p = \frac{\pi}{32}\,(d^4 - d_1^4) = \frac{\pi}{2}\,(r^4 - r_1^4)$$

$$k_g = \tfrac{1}{4}\sqrt{d^2 + d_1^2} = \tfrac{1}{2}\sqrt{r^2 + r_1^2}$$

$$k_p = \sqrt{\frac{(d^2 + d_1^2)}{8}} = \sqrt{\frac{(r^2 + r_1^2)}{2}}$$

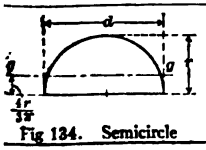


Fig 134. Semicircle

$$I_g = \frac{d^4\,(9\,\pi^2 - 64)}{1\,152\,\pi} = 0.00686\,d^4 = 0.110\,r^4$$

$$k_g = \frac{d\,\sqrt{(9\,\pi^2 - 64)}}{12\,\pi} = 0.132\,d = 0.264\,r$$

Table of Plane Figures (*Continued*)

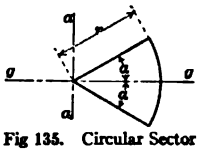


Fig 135. Circular Sector

A = Area of sector $= r^2\alpha$, α expressed in radians.

$$I_g = \frac{Ar^2}{4}\left(1 - \frac{\sin\alpha \cdot \cos\alpha}{\alpha}\right)$$

$$I_a = \frac{Ar^2}{4}\left(1 + \frac{\sin\alpha \cdot \cos\alpha}{\alpha}\right)$$

$$k_g = \frac{r}{2}\sqrt{1 - \frac{\sin\alpha \cdot \cos\alpha}{\alpha}}$$

$$k_a = \frac{r}{2}\sqrt{1 + \frac{\sin\alpha \cdot \cos\alpha}{\alpha}}$$

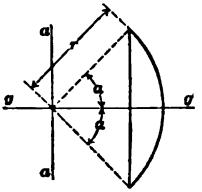


Fig 136. Circular Segment

A = Area of segment $= r^2(\alpha - \sin\alpha \cdot \cos\alpha)$, α expressed in radians.

$$I_g = \frac{Ar^2}{4}\left(1 - \frac{2}{3}\,\frac{\sin^2\alpha \cdot \cos\alpha}{\alpha - \sin\alpha \cdot \cos\alpha}\right)$$

$$I_a = \frac{Ar^2}{4}\left(1 + \frac{2\sin^3\alpha \cdot \cos\alpha}{\alpha - \sin\alpha \cdot \cos\alpha}\right)$$

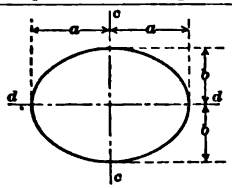


Fig 137. Ellipse

$$I_d = \frac{\pi}{4}ab^3 \qquad I_c = \frac{\pi}{4}ba^3$$

$$k_d = \frac{b}{2} \qquad k_c = \frac{a}{2}$$

44. Masses

Moment of inertia of a mass, with respect to an axis, is the algebraic s of the products of each element of mass and the square of its distance from t axis. $I = \Sigma mr^2$, in which m is any element of mass and r its distance from t axis. The common engineer's unit is slug-ft² (Art 53).

Radius of gyration is the distance from the axis to a point at wh the entire mass were concentrated, the moment of inertia would not be chan If k is the radius of gyration, and M the total mass, then $I = Mk^2$; or $k = \sqrt{I}$ and is measured in ft. Value of k is always between the least and gr values of r.

Parallel axis theorem. Let $\bar{I}$ be the moment of inertia with respect t axis through the center of gravity, and I the moment of inertia for any allel axis at a distance d; then $I = \bar{I} + Md^2$, and $k^2 = \bar{k}^2 + d^2$.

Principal axes and moments of inertia. The integrals $\int yz\,dm$, $\int zx$ and $\int xy\,dm$ are called products of inertia. For an oblique axis, through origin with direction angles, α, β, and γ,

$$I = I_x\cos^2\alpha + I_y\cos^2\beta + I_z\cos^2\gamma - 2J_{xy}\cos\alpha\cos\beta - 2J_{yz}\cos\beta\cos\gamma - 2J_{zx}\cos\gamma\cos\alpha$$

t each origin, there is one set of rectangular axes for which the three products of ia vanish, and with respect to two of which the moments of inertia are least and test of all moments of inertia with respect to axes at that origin. The axes are called cipal axes, and the moments of inertia with respect to them are called principal mo- ts of inertia. If a body has three planes of symmetry, at right angles to each other, lines of intersection are principal axes at the point of intersection. If a body has planes of symmetry, their intersection is a principal axis for every point on it as a, and the other two axes are in the planes of symmetry. If a body has one plane mmetry, any perpendicular to the plane is a principal axis at the point where the pierces the plane. At the origin, the x-axis is a principal axis if $J_{xy} = 0$ and $J_{xz} = 0$; -axis is a principal axis if $J_{xy} = 0$ and $J_y = 0$; and the z-axis is a principal axis if $= 0$ and $J_{yz} = 0$.

45. Formulas for Masses

in circular lamina, of radius a and mass m, with respect to a diameter, $ma^2 + 4$. For axis perpendicular to face of lamina at its center, $I = ma^2 + 2$.

lid of revolution of volume V and mass M, with respect to its axis,

$$I = \left[\pi M \int y^4\,dx\right] \div 2\,V$$

respect to an axis at right angles to its geometrical axis,

$$I = \left[\pi M \int \left(\frac{y^4}{4} + x^2y^2\right)dx\right] \div V$$

in straight rod of length l and mass M. Axis perpendicular to rod at r, $I = Ml^2 \div 12$; at end, $I = Ml^2 \div 3$. Axis inclined to rod at angle α: ıter, $I = Ml^2 \sin^2\alpha \div 12$; at end, $I = Ml^2 \sin^2\alpha \div 3$.

ctangular prism of dimensions a, b, c. Axis through center of gravity, el to edge c, $I = M\,(a^2 + b^2) \div 12$. Axis through center of base bc and el to edge c, $I = M\,(4\,a^2 + b^2) \div 12$.

ht circular cylinder of length h and base radius a; for its geometrical $I = Ma^2 \div 2$. Axis through center of gravity, parallel to base, $I = M$ $+ h^2) \div 12$. Axis coinciding with diameter of base, $I = M\,(3\,a^2 + 4\,h^2) \div 12$.

llow circular cylinder of height h and radii R and r; about axis of cylin- $= M\,(R^2 + r^2) \div 2$. Axis through center of gravity, perpendicular to axis inder, $I = M\,(3\,R^2 + 3\,r^2 + h^2) \div 12$.

ptical cylinder, length h and semi-axes a and b; for its geometrical $I = M\,(a^2 + b^2) \div 4$. Axis through center of gravity, parallel to minor b, of elliptical section, $I = M\,(3\,a^2 + h^2) \div 12$.

ıere of radius a, about a diameter, $I = 2\,Ma^2 \div 5$.

ht circular cone, of height h and base radius a. About its axis, $I = 3\,Ma^2$ Axis through vertex, perpendicular to geometrical axis, $I = 3\,M\,(a^2 + 4\,h^2)$

psoid of axes $2\,a$, $2\,b$, and $2\,c$. About the axis $2\,b$, $I = M\,(a^2 + c^2) \div 5$.

ht rectangular pyramid, of height h and sides of base a and b, about **trical axis,** $I = M\,(a^2 + b^2) \div 20$. Axis through vertex perpendicular to **trical axis** and parallel to b side of base, $I = M\,(a^2 + 12\,h^2) \div 20$. Axis **h center** of gravity, parallel to b edge of base, $I = M\,(4\,a^2 + 3\,h^2) \div 80$.

:ular ring of circular section, radius of section $= r$, and center of **ı at distance** R from axis of ring. About axis of ring, $I = M\,(4\,R^2 + 3\,r^2)$ About **axis through center of gravity, perpendicular to axis** of ring, $(4\,R^2 + 5\,r^2) \div 8$.

KINEMATICS

46. Rectilinear Motion

A particle is in motion with respect to surrounding objects if its position w respect to them is continually changing. Displacement of a particle is its cha in position. If A is the position of a particle at a time t_1, and B its positio later time t_2, its displacement in the time interval $t_2 - t_1 = \Delta t$ is the vector $\overline{AB}$ matter whether path is straight or curved. Velocity of a particle is its time r of displacement. Acceleration of a particle is its time rate of change of velo

Rectilinear motion of a particle. Let s be distance measured along path of a particle, s_1 the distance from origin at time t_1, s_2 the distance later time t_2, $\Delta s = s_2 - s_1$ = displacement in time interval $\Delta t = t_2 - t_1$. T AVERAGE VELOCITY = $\Delta s \div \Delta t$. If the position changes at a uniform r actual velocity at any time = $\Delta s \div \Delta t$. For every case, INSTANTANEOUS VE ITY = $v = \dfrac{ds}{dt} = \text{limit}_{\Delta t \doteq 0} \left(\dfrac{\Delta s}{\Delta t}\right)$. UNIT OF VELOCITY is any distance unit divi by any time unit. SPEED is the name for magnitude of velocity, and d not refer to its sense or linear direction. If s is not known in terms of displacements for several time intervals, beginning or ending at the instant question, are known, an appr value of v can be obtained fro consideration of average spee

$s_2 - s_1 = \Delta s$, feet	Δt, seconds	$\Delta s \div \Delta t$, ft per sec
28.8 − 3.73 = 25.07	2 to 6 = 4	6.27
20.83 − 3.73 = 17.10	2 to 5 = 3	5.70
13.87 − 3.73 = 10.14	2 to 4 = 2	5.07
8.1 − 3.73 = 4.37	2 to 3 = 1	4.37

For example, determine the s at time $t = 2$ sec, having giv accompanying 4 sets of observa ues for Δs and Δt.

Column 3 is computed from the first two. As $\Delta t \doteq 0$, the value of aver velocity last column appears to approach a limit of 3.6 ft per sec. A value of this limit may determined graphically by platting time intervals along the horiz (Fig 138), and c sponding values of aver velocity vertically. The curve $ABCD$, "faired" to meet the ordinate through O at E (zero time interval), determines $v = OE = 3.6$ ft per sec.

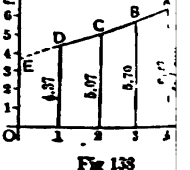


Fig 138

Let v_1 = velocity at time t_1, v_2 = velocity at a later time t_2, $\Delta v = v_2 - v_1$ = change in velocity in time interval Δt; then AVERAGE ACCELERATION = $\Delta v \div \Delta t$. If velocity changes at a uniform rate, the actual acceleration at any time $\Delta v \div \Delta t$. For every case, INSTANTANEOUS ACCELERATION

$$= a = \frac{dv}{dt} = \frac{d^2s}{dt^2} = \text{limit}_{\Delta t \doteq 0}\left(\frac{\Delta v}{\Delta t}\right)$$

UNIT OF ACCELERATION is any velocity unit divided by any time unit. unit used herein is feet per second per second, or ft per sec^2. If s or v are known in terms of t, but the placement or velocity record available, an approx value can be obtained from a consi tion of average accelerations the displacement record or available, values of v must first obtained as in the above numer example.

t, sec	v, ft per sec	Δt, sec	Δv, ft per sec	$\Delta v \div \Delta t$, ft per sec^2
2	3.6	2 to 6 = 4	4.8	1.2
3	5.1	2 to 5 = 3	3.9	1.3
4	6.4	2 to 4 = 2	2.8	1.4
5	7.5	2 to 3 = 1	1.5	1.5
6	8.4			

For example, determine the acceleration at time $t = 2$ sec, having given the five set values for t and v as shown in first two columns of accompanying table.

umns 3 and 4 are computed from the first two. s in last column are computed from third and fourth. = 0, the value of aver acceleration appears to ap- 1.6 ft per sec² as a limit. A value of this limit may ermined graphically by platting time intervals along orizontal (Fig 139), and corresponding values of aver ration vertically. The curve *ABCD*, "faired" to meet dinate through *O* at *E* (zero time interval), determines *E* = 1.6 ft per sec².

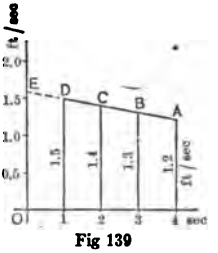


Fig 139

is given in terms of t, v and a are determined fferentiation as indicated above. If a is given ms of t, v and s are determined by integration. ormulas are

$$= \int_{t_1}^{t_2} a\,dt;\quad \int_{s_1}^{s_2} ds = \int_{t_1}^{t_2} v\,dt;\quad \int_{v_1}^{v_2} v\,dv = \int_{s_1}^{s_2} a\,ds;\quad \int_{t_1}^{t_2} dt = \int_{v_1}^{v_2} \frac{1}{a}\,dv = \int_{s_1}^{s_2} \frac{1}{v}\,ds$$

ich s_1, v_1, t_1, are simultaneous values. For uniform acceleration, a = con- $v = at + v_o$, $s = \frac{1}{2}at^2 + v_o t + s_o$, v_o being initial velocity and s_o initial dis- . For bodies falling from rest in vacuum, $v_o = 0$, $s_o = 0$, and $a = g =$ 32.2 ft per sec², $s = \frac{1}{2}gt^2$, $v = gt = \sqrt{2gs}$.

47. Motion Graphs

tion graphs. DISTANCE-TIME (*s-t*) GRAPH (Fig 140). Any ordinate ents value of s at corresponding time. Slope of tangent represents speed $= \frac{ds}{dt}$. CB must be read to distance scale and AB to time scale; then speed $v = CB \div AB$. SPEED-TIME (*v-t*) GRAPH (Fig 141). Any ordinate represents value of v at corresponding time. Slope of tangent represents $a = \frac{dv}{dt}$. FE is read to speed scale and DE to time scale; then $a = FE \div DE$. Area between curve and time axis, limited by any two ordinates, represents displacement in corresponding time interval. $\Delta s =$ of area × product of scale numbers = area × $m \times n$. If area is below axis it is considered minus. ACCELERATION-TIME (*a-t*) GRAPH. Any ordi-

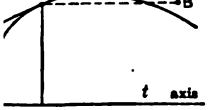


Fig 140

represents value at corresponding Area between and time axis, d by any two ites, represents e in speed in cor- ding time inter- Δv = sq in of area duct of scale numbers.

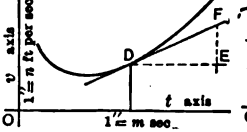


Fig 141

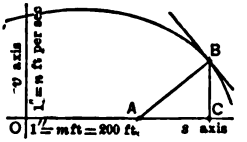


Fig 142

SPEED-DISTANCE (*v-s*) GRAPH (Fig 142), for recti- or curved path. The subnormal at any point B (line AC) represents ntial component of acceleration at corresponding instant. If m = distance number and n = speed scale number, then $a = AC$ (in) × $m^2 \div n$. (In Fig $m = 200$, $n = 30$.)

ple harmonic motion. If a point P moves in a circular path of radius r at uni- peed, its projection on any diameter has simple harmonic motion. Radius r is called

AMPLITUDE. PERIOD is time required for the projection to go from one end of diam to other and back. FREQUENCY is number of periods per unit time.

When $t = 0$, let P be at P_0 (Fig 143). ϵ is called the lead angle (lag, if negative). simple harmonic motion (S H M) of V in the vertical diam, $y = r \sin(\theta + \epsilon) = r \sin(\omega t +$ in which $\omega = \frac{d\theta}{dt}$ = radians per unit time.

$$v_y = r\omega \cos(\theta + \epsilon) = r\omega \cos(\omega t + \epsilon) = \omega x$$
$$a_y = -r\omega^2 \sin(\theta + \epsilon) = -r\omega^2 \sin(\omega t + \epsilon) = -\omega^2 y$$

For S H M of H in horiz diam:

$$x = r \cos(\theta + \epsilon) = r \cos(\omega t + \epsilon); \qquad v_x = -r\omega \sin(\theta + \epsilon) = -r\omega \sin(\omega t + \epsilon) =$$
$$a_x = -r\omega^2 \cos(\theta + \epsilon) = -r\omega^2 \cos(\omega t + \epsilon) = -\omega^2 x$$

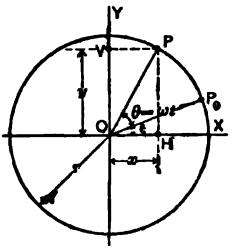


Fig 143

Crank and connecting-rod mechani Problem is to find expressions for the ve and acceleration of any point in the cross as A in Fig 144. Let $c = r \div l$, n = rev per (assumed constant), ω = radians of angle

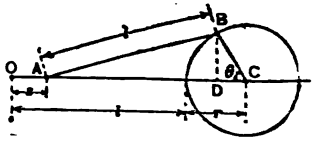


Fig 144

scribed by crank per second, and s = distance of A from its extreme left p tion, all distances expressed in ft. Then,

$$s = (l + r) - l(1 - c^2 \sin^2\theta)^{\frac{1}{2}} - r\cos\theta$$

$$v = r\omega\left(\sin\theta + \frac{c \sin 2\theta}{2(1 - c^2\sin^2\theta)^{\frac{1}{2}}}\right); \qquad a = r\omega^2\left(\cos\theta + \frac{c\cos 2\theta + c^2\sin^4\theta}{(1 - c^2\sin^2\theta)^{\frac{3}{2}}}\right.$$

The above formulas are exact; close approximations are:

$$s = r(1 - \cos\theta) + \tfrac{1}{4}cr(1 - \cos 2\theta); \qquad v = r\omega(\sin\theta + \tfrac{1}{2}c\sin 2\theta)$$
$$a = r\omega^2(\cos\theta + c\cos 2\theta)$$

Exact and approximate formulas for a give same values for extreme r and left positions of point A.

48. Curvilinear Motion

Curvilinear motion of a particle. The definitions of displacement velocity, given at the beginning of this article, apply to curvilinear mo Velocity is a vector quantity. If s is distance measured along the curved p the magnitude of velocity (speed) at an instant $= \frac{ds}{dt}$; the linear direct the velocity is tangent to the path at the instantaneous position of the par and the sense of velocity corresponds to the direction of motion of the par at the instant.

Acceleration for curvilinear motion is time rate of change of velocity a vector quantity. The velocity vector changes in magnitude and dir In Fig 145, let A, B, C represent positions of particle P in its curved p s, distance along the path, and v_1, v_2, v_3, velocity vectors at A, B, C. Plot v ity vectors $O'A'$, $O'B'$, $O'C'$, etc, from any origin O' to represent the velo

, *B*, *C*, etc. The curve $A'B'C'$, drawn through the ends of the vectors, is l a hodograph for the motion. For every position of *P* in its path, there corresponding position P' in odograph; and P' describes ce s' on hodograph, while scribes distance s on path. r $O'P'$ represents the velocity

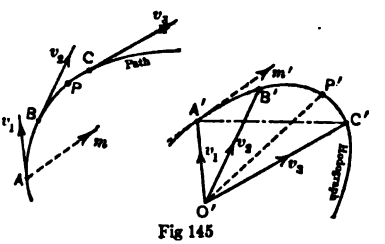


Fig 145

In time Δt, *P* moves from *C*, its velocity changes from to $O'C'$, and the velocity te is $A'C'$. AVERAGE ACRATION for interval Δt = or $A'C' \div \Delta t$), and it has irection of the chord $A'C'$. NTANEOUS ACCELERATION of *P* at $A = a$ = limit of aver acceleration as proaches zero.

$$a = \lim_{\Delta t \doteq 0}\left(\frac{\text{vector } A'C'}{\Delta t}\right) = \lim_{\Delta t \doteq 0}\left(\frac{\text{arc } A'B'C'}{\Delta t}\right) = \frac{ds'}{dt} = \text{speed of } P'$$

idograph. The direction of a is along the tangent $A'm'$, and as P' is movlockwise, the sense is as indicated by arrow at m'. Hence acceleration is Am, parallel to $A'M'$ and $= \frac{ds'}{dt}$. Unit is any velocity unit divided by ime unit.

otion graphs. Speed may be computed from a distance-time graph in ame way as for rectilinear motion. Tangential component (see Art 49) celeration may be computed from the speed-time graph; and the distance ay be obtained from the area under the curve.

49. Components of Velocity and Acceleration

mponents of velocity and acceleration of a particle for any curved (not a plane curve). The position of the particle *P*, being defined by its coates x, y, z, the axial components of velocity are $v_x = \frac{dx}{dt}$, $v_y = \frac{dy}{dt}$, $v_z = \frac{dz}{dt}$.

ltant velocity $v = \sqrt{v_x^2 + v_y^2 + v_z^2}$, and its direction cosines are $\cos\theta_x = \frac{v_x}{v}$, $_y = \frac{v_y}{v}$, $\cos\theta_z = \frac{v_z}{v}$. Axial components of acceleration are:

$$a_x = \frac{dv_x}{dt} = \frac{d^2x}{dt^2}; \qquad a_y = \frac{dv_y}{dt} = \frac{d^2y}{dt^2}; \qquad a_z = \frac{dv_z}{dt} = \frac{d^2z}{dt^2}.$$

ltant acceleration $a = \sqrt{a_x^2 + a_y^2 + a_z^2}$; and its direction cosines are:

$$\cos\phi_x = \frac{a_x}{a}; \qquad \cos\phi_y = \frac{a_y}{a}; \qquad \cos\phi_z = \frac{a_z}{a}$$

e path is a plane curve, $v_z = 0$, $a_z = 0$. The tangential and normal comnts of acceleration are $a_t = \frac{dv}{dt} = \frac{d^2s}{dt^2}$, and $a_n = \frac{v^2}{\rho}$, ρ being the radius of ature. Resultant acceleration is $a = \sqrt{a_t^2 + a_n^2} = \sqrt{a_x^2 + a_y^2}$.

50. Translation

anslation of a rigid body is a motion such that each straight line in it ins fixed in direction. The paths of all particles of the body are exa

alike, straight or curved (not necessarily plane curves); the velocities of particles at an instant are the same, and their accelerations at an instant the same. For these reasons, it is customary to use the expressions, "velo of the body" and "acceleration of the body."

51. Rotation

Rotation of a rigid body is a motion such that one line of the body, or its extension, remains fixed. The fixed line is the axis. The plane that the mass center perpendicular to the axis is the plane of rotation. The pa

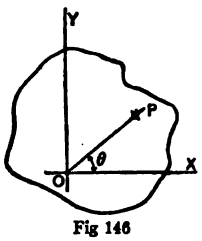


Fig 146

of all particles are circles with centers on the a Since all lines of the body parallel to the plane rotation sweep out equal angles in equal times, it customary to describe rotation by the behavior of radial line. In Fig 146 let θ be the angle from the axis to the radial line OP. $\Delta\theta = \theta_2 - \theta_1$ is the ANGU DISPLACEMENT of the body in the time $\Delta t = t_2 - t_1$. is expressed in any angular unit. ANGULAR VELO is the time rate of angular displacement. Aver angular velocity $= \dfrac{\theta_2 - \theta_1}{t_2 - t_1} = \dfrac{\Delta\theta}{\Delta t}$. The limit of average, as Δt approaches zero, is the instantan angular velocity. The symbol for angular velocity is ω, and $\omega = \lim_{\Delta t \to 0}\left(\frac{\Delta}{\Delta}\right.$ $= \dfrac{d\theta}{dt}$. The unit is any angular unit divided by any time unit, such as rad per sec, radians per min, rev per sec, or rev per min. The sign depends on numerator of the fraction, or the way in which θ is changing. ANGU ACCELERATION is the time rate of angular velocity. Average angular accele tion $= \dfrac{\omega_2 - \omega_1}{t_2 - t_1} = \dfrac{\Delta\omega}{\Delta t}$. The limit of this average, as Δt approaches zero instantaneous angular acceleration. The symbol for angular acceleration α, and $\alpha = \lim_{\Delta t \to 0}\left(\dfrac{\Delta\omega}{\Delta t}\right) = \dfrac{d\omega}{dt} = \dfrac{d^2\theta}{dt^2}$. The sign of α depends on the numera of the fraction, or on the way in which ω is changing.

Relations between v and a of any point P, of a rotating body and ω and α the body. Let r = radius of path of P. Then

$$v = r\omega, \qquad a_t = r\alpha, \qquad a_n = r\omega^2, \qquad a = r\sqrt{\alpha^2 + \omega^4}$$

Sense of v must agree with sense of ω. Sense of a_t must agree with se of α. Sense of a_n is always toward the axis.

52. Plane Motion

Plane motion of a rigid body is a motion such that each particle of body moves in a plane at a constant distance from a fixed plane, through center of gravity, called the plane of motion. Each line of the body paralle plane of motion turns through the same angle θ in the same time inter ANGULAR DISPLACEMENT of the body in time interval Δt is change of ang position of any line in the plane of motion. ANGULAR VELOCITY is time-rat angular displacement. ANGULAR ACCELERATION is time-rate of angular ve ity. The expressions, units, and rules of signs for plane motion are the s as those for rotation about a fixed axis (Art 51). Angular displacem $\Delta\theta = \theta_2 - \theta_1$; angular velocity is $\omega = \dfrac{d\theta}{dt}$; angular acceleration is $\alpha = \dfrac{d\omega}{dt} =$

lane motion may be traced by giving the history of the movement of one point of dy (called a base point) in its own curved path, and a description of the rotation of dy about the selected base point. The point selected e should be one for which the motion is readily spe-

In the case of a wheel rolling along a straight path, ater would be selected as a base point. VELOCITY r POINT P of the body, at an instant, with respect xed point O, is the vector sum of the velocity of oint A, with respect to O, and of velocity of P with to A due to rotation about A. Thus (Fig 147) O d point, A the moving base point, and P any other of the body at distance r from A; v_1 is velocity of A spect to O, and $v_2 = r\omega$ is velocity of P with respect to A. Resultant velocity of P spect to $O = v$; or

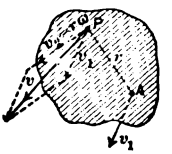


Fig 147

$$v_{P \text{ to } O} = v_{P \text{ to } A} + v_{A \text{ to } O}$$

ERATION OF ANY POINT P, with respect to a fixed point O, at an instant, has two ents; one is that of the base point A with respect to O, and the other that of P spect to base A. Acceleration of P with respect to A is rotational, and is con-tly replaced by its tangential and normal components, $a_t = r\alpha$ and $a_n = r\omega^2$. Then ant acceleration of P, with respect to O, is the vector sum of $r\alpha$, $r\omega^2$, and accel-n of A with respect to O. Thus (Fig 148) a_1 is acceleration of A to O, and accele-of P to A is resultant of a_t and a_n. Acceleration of P to O is a = vector sum of and a_n.

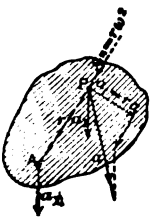


Fig 148

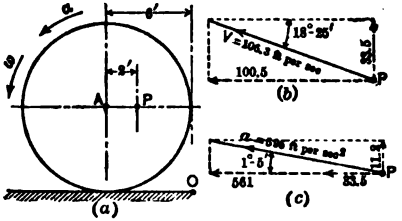


Fig 149

mple. A wheel of 6 ft radius rolls along a straight horiz path, and at a certain t the point P, 2 ft from center of wheel, is in the position shown in Fig 149a. At stant $\omega = 16.75$ radians per sec and $\alpha = 5.6$ radians per sec^2. Determine the y and acceleration of point P with respect to fixed point O at the specified instant. ION. Select center A as base point. From relation between v and a of any of a rotating body, and ω and α of the body (Art 51),

$v_A = r\omega = 2 \times 16.75 = 33.5$ ft per sec, vertically upward.

$v_O = r\omega = 6 \times 16.75 = 100.5$ ft per sec, horizontally toward left.

refore, $v_{P \text{ to } O} = 106.3$ ft per sec, upward to left, at 18° 25′ to horizontal (Fig 149b).

$a_{PO} = r\alpha = 6 \times 5.6 = 33.5$ ft per sec^2, horizontally toward left.

$= r\alpha = 2 \times 5.6 = 11.2$ ft per sec^2, vertically upward.

$= r\omega^2 = 2 \times (16.75)^2 = 561$ ft per sec^2, horizontally toward left.

refore, $a_{P \text{ to } O} = 595$ ft per sec^2, upward to left, at 1° 5′ to horizontal (Fig 149c).

tantaneous axis. For a body having plane motion, there is always one point in in its extension), at each instant, for which the velocity with respect to A is equal pposite to velocity of A with respect to O, that is, its velocity is zero at the instant. oint Q is called the INSTANTANEOUS (or instant) CENTER of rotation, and a line through pendicular to the plane of motion, is called the INSTANTANEOUS AXIS. Since Q is for the instant, the resultant velocities of all points at the instant are purely rota-about the instant axis. The instant center is the intersection of two lines drawn ny two points, C and D, in the plane of the motion, perpendicular to their ve

ties. If the velocity of the point C is known, ω for the body is determined by di v_C by the distance of C from Q, or by r_C. The velocity of any other point E is ω perpendicular to the radius r_E.

The position of Q in the body (or in its extension) is continually changing; its loc a line (usually curved) fixed in the body and moving with it, called the BODY CENTR The locus of the positions of Q in the fixed plane of motion is a line (usually curved) c the SPACE CENTRODE. The plane motion may be considered as produced by the rol without slipping, of the body centrode upon the space centrode.

KINETICS

53. Force and Mass Units

Mass is the quantity of matter in a body; it means substance as meas by a beam scale. Standards of mass are the pound and kilogram; they certain pieces of metal preserved in London and Paris respectively. mass of a body does not change with locality.

A force F acting on a particle of mass m gives it an acceleration a parallel to the f and the relation between the quantities is given by the equation $F = Kma$, in w K is a constant. It is customary to use a KINETIC SYSTEM OF UNITS, one for which K For such a system, unit force gives to unit mass unit acceleration. The $c\,g\,s$ syste units is a kinetic system; in it force is measured in dynes, mass in gm, and acce in cm per sec^2. The system is called an absolute system, because the units are

Units. The English speaking engineers' unit of force is the pull of the ea on 1 lb of mass. This amount of pull is called ONE POUND OF FORCE. A EARTH PULL varies with the elevation of the pound of mass, POUND FOR not an absolute unit. But, the range of variation is so slight, that the involved by assuming it constant is negligible in most computations engineers' corresponding mass unit has no standard name, but will here called a SLUG; it is the mass in which an acceleration of 1 ft per sec^2 is prod by a force of 1 lb. 1 slug of mass = about 32.2 lb of mass. In any comp tion that involves mass with pounds force, feet distance, and seconds time. should be reduced to slugs. Slugs of mass = wt of body (lb) ÷ 32.2 (ap value of acceleration due to gravity. In the following problems, the numer value of this acceleration is taken = 32).

54. Motion of Mass Center and Translation

For any particle of mass m

$$\sum \begin{pmatrix}\text{components in any } x \text{ direction of}\\ \text{all forces acting on the particle}\end{pmatrix} = \begin{pmatrix}\text{mass of particle times its } x \text{ comp}\\ \text{of acceleration}\end{pmatrix}$$

or, more briefly, if R_x represents the component of the resultant in the x direct

$$R_x = \Sigma F_x = ma_x$$

For any body, rigid or non-rigid,

$$\sum \begin{pmatrix}\text{components in any } x \text{ direction of}\\ \text{all external forces, pairs of in-}\\ \text{ternal forces excluded}\end{pmatrix} = \begin{pmatrix}\text{total mass of system times } x \text{ com}\\ \text{nent of acceleration of mass cent}\end{pmatrix}$$

or, more briefly, $$\Sigma F_x = M\bar{a}_x$$

This is known as the equation of the motion of the mass center of a ba system of bodies.

Rigid body having motion of translation. Velocities of all partic the same, and accelerations of all particles are the same. Resultant

d forces is a single force $R = Ma$, acting through the mass center, parallel acceleration,

$$\left(\begin{array}{l}\text{components in any } x \text{ direction of}\\ \text{applied forces including weight}\\ \text{of body}\end{array}\right) = \left(\begin{array}{l}\text{mass of body times the } x \text{ component of}\\ \text{acceleration}\end{array}\right)$$

ore briefly, $$R_x = \Sigma F_x = Ma_x \qquad (C)$$

$$\sum\left(\begin{array}{l}\text{moments about any } x\text{-axis of ap-}\\ \text{plied forces, including weight}\\ \text{of body}\end{array}\right) = \left(\begin{array}{l}\text{moment of resultant about same}\\ x\text{-axis}\end{array}\right)$$

$$\Sigma M_x = Ma_x \times arm \qquad (D)$$

lowing method of procedure is advisable in solving a problem: (*a*) Make tch of the body and place arrows upon it to represent all forces (including ant at mass center), indicating unknown forces or distances by letters. ume for each body if a system of bodies is under discussion. (*b*) Write as resolution equations (*C*) and moment equations (*D*) as there are unknown tities. (*c*) Solve the simultaneous equations to determine unknown quan-

ample 1. A rectangular box (Fig 150), 2 by 2 by 8 ft high, weighing 480 lb, is upon a rough flat-car. If the car is running on a straight level track, what acceleration will cause the box to tip? If coeff of friction = 0.22, will the box slip or tip first? SOLUTION. If tipping about edge O is impending, normal reaction is in position shown by N. By equation (*D*), $\Sigma M_O = -480 \times 1 = -\frac{480}{32} \times a \times 4$; hence $a = 8$ ft per sec². If slipping impends, N acts through some point O' at distance x from center line. $a_y = 0$; hence $\Sigma F_y = 0 = N - 480$, shows $N = 480$ lb. Limiting friction = 0.22 N = 105.6 lb. By equation (*C*), $\Sigma F_x = 105.6 = \frac{480}{32} \times a$; hence, $a = 7.04$ ft per sec² to cause slipping. As this value of a is smaller than the first one, slipping will occur before tipping. x may be found from $\Sigma M_{O'} = -4 \times Ma$.

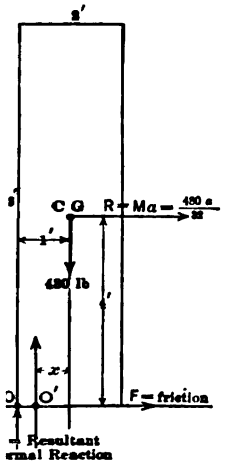


Fig 150

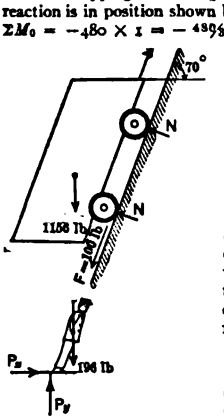


Fig 151

Example 2. A mine car (Fig 151), weighing 960 lb starts up a 70° slope, with an acceleration of 6 ft per sec², and is ed at top with an acceleration of 4 ft per sec². A 196-lb man stands in the car. mine starting and stopping tensions in the cable, if the frictional resistance amounts 100 lb parallel to track. Also find reaction of car on man's feet for starting and stop-

SOLUTION:

starting, man and car: $\Sigma F_x = T - 100 - 1156 \sin 70° = \frac{1156}{32} \times 6 = 216.9$; $T = 1402.9$ lb.

stopping, man and car: $\Sigma F_x = T - 100 - 1156 \sin 70° = \frac{1156}{32} \times (-4) = -144.5$; hence, $T = 1041.5$ lb.

n only, starting: $\Sigma F_x = P_x = \frac{196}{32} \times 6 \cos 70° = 12.57$ lb to right. $\Sigma F_y = P_y - 196 = \frac{196}{32} \times 6 \sin 70° = 34.55$ lb; hence, $P_y = 230.55$ lb up. $P = 231$ lb up to right, gle of 3° 7' to vertical.

Man only, stopping: $\Sigma F_x = -P_x = \frac{196}{32} \times (-4) \cos 70° = 8.38$ lb to left. ΣF
$+P_y - 196 = \frac{196}{32} \times (-4) \sin 70° = 23$ lb; hence, $P_y = 196 - 23 = 173$ lb up.
173.1 lb up to left, at angle of 2° 46′ to vertical.

55. Rotation and Plane Motion

Rigid body having motion of rotation about a fixed axis. Angular velo
$= \omega$, angular acceleration $= \alpha$, and moment of inertia of body about axi
rotation $= I$. The usual equation of motion is

$$\sum \begin{pmatrix} \text{moments about axis of rotation} \\ \text{of all external forces including} \\ \text{weight of body} \end{pmatrix} = \begin{pmatrix} \text{moment of inertia about axis of rota} \\ \text{tion times angular acceleration} \end{pmatrix}$$

or, more briefly,

$$\Sigma M = I\alpha$$

Equation (E) is used to determine one unknown quantity, as a force, arm, α, or I, according to the problem. In solving problems dealing wi

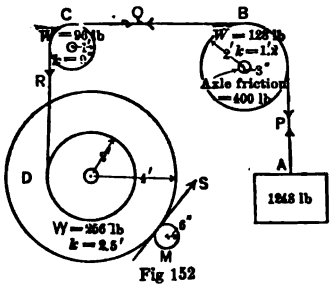


Fig 152

connected system, in which cer
bodies have motion of translation
others rotation, (C) and (E) may
used as simultaneous equations.

Example 3. Motor M (Fig 152) is ge
to hoisting drum D to raise and l
weight A. Assume A descending at s
of 15 ft per sec, and determine tange
force S, exerted by motor, that will st
in 30 ft at a uniform rate. Also, deter
tensions P, Q, and R during the mo
Neglect mass of motor armature, and ne
all friction except that of axle on B (s
and sizes are not selected to conform
practice, but to illustrate the metho
writing and using equations). SOLUTION. Determine acceleration of A by for
given under Rectilinear motion (Art 46).

$$\int_{v_1}^{v_2} v\,dv = \int_{s_1}^{s_2} a\,ds; \quad \left[\frac{v^2}{2}\right]_{15}^{0} = a\left[s\right]_{0}^{30}; \quad -\frac{(15)^2}{2} = a \times 30; \quad a = 3.75 \text{ ft per sec}^2 \text{ up}$$

By equations given under Angular acceleration (Art 51), $3.75 = a_A = 2a_B = a$
$2\alpha_D$. By (C) and (E) of this Art, equations of motion for each body are:

$$P - 1\,248 = \frac{1\,248}{32} \times 3.75 = 146.25$$

$$2 \times Q - 2 \times P + 400 \times \frac{1}{4} = \frac{128}{32}(1.2)^2 \times \frac{3.75}{2} = 10.8$$

$$1 \times R - 1 \times Q = \frac{96}{32}\left(\frac{3}{4}\right)^2 \times 3.75 = 6.33$$

$$4 \times S - 2 \times R = \frac{256}{32}\left(2.5\right)^2 \times \frac{3.75}{2} = 93.75$$

Solution of these simultaneous equations determines $S = 701.4$ lb, $P = 1\,354.1$
$Q = 1\,349.7$ lb and $R = 1\,356$ lb. The required motor torque is $S \times r = 701.4 \times$
$0.5 = 351$ ft-lb.

An axis of rotation which does not pass through the mass center must
held by forces (exerted by bearings) to keep it from shifting position. The
bearing reactions depend upon the weight of the body, the manner in wh
the mass of body is distributed about the axis, the applied forces, the angu
velocity ω, and the angular acceleration α. Generally, the resultant of
applied forces for such a body is not a single force, but a single force at a sele

a and a couple. Selecting the origin on the axis of rotation, the axial ponents of the single force and axial components of the couple are given by ollowing six equations:

$$\left.\begin{aligned} \Sigma F_x &= M\bar{x}\alpha - M\bar{x}\omega^2 & \Sigma M_x &= -\alpha\int xy\,dm - \omega^2\int yz\,dm \\ \Sigma F_y &= 0 & \Sigma M_y &= I_y\alpha \\ \Sigma F_z &= -M\bar{x}\alpha - M\bar{x}\omega^2 & \Sigma M_z &= -\alpha\int yz\,dm + \omega^2\int xy\,dm \end{aligned}\right\} \qquad (H)$$

ese equations, the axis of rotation is fundamentally the Y-axis; $\bar{x}$, $\bar{y}$, and the instantaneous coordinates of the mass center; ΣF_x, ΣF_y, ΣF_z are ums of components of all applied forces in the axial directions; ΣM_x, ΣM_z are the sums of moments of all applied forces about the axes; and onvention of signs for moments of forces, sense of α, and measurement of $= \frac{d^2\theta}{dt^2}$ is as follows:

rotation about Z-axis from $+X$ to $+Y$ is positive,
rotation about X-axis from $+Y$ to $+Z$ is positive,
rotation about Y-axis from $+Z$ to $+X$ is positive.

itions (H) are simultaneous at each instant. They are used more often termine the forces exerted by the bearings on the axle, than to determine esultant.

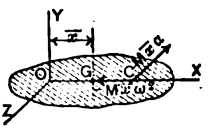


Fig 153

ecial Case (Fig 153). Choose the X-axis through the center and let XZ be a plane of symmetry of a geneous body. The resultant is a single force in the of symmetry having the Z component $-M\bar{x}\alpha$ and component $-M\bar{x}\omega^2$ acting at point C. $\overline{OC} = \frac{k^2_y}{\bar{x}}$, ing radius of gyration about Y-axis. If $\bar{x} = 0$, the result- ecomes a couple, in the XZ-plane, of moment $= \Sigma M_y = I_y\alpha$. If $\alpha = 0$ and $\bar{x} \neq 0$, sultant $= -M\bar{x}\omega^2$, in the sense CO. If $\alpha = 0$ and $\bar{x} = 0$, the resultant vanishes.

entrifugal force, body rotating about fixed Y-axis. Any particle of mass oves in a circular path of radius r. The resultant of all forces acting ne particle has a normal component $= mr\omega^2$ and a tangential component $r\alpha$. The component $mr\alpha$ increases or decreases the speed of the particle; component $mr\omega^2$ continually changes the direction of the linear velocity. resultant of such forces for all the particles of the body is equivalent to esultant specified by equations (H). If ω is constant and $\alpha = 0$, the re- nt force acting on a particle to make it rotate in its circular path is $mr\omega^2$ rd the axis, and is called CENTRIPETAL FORCE. CENTRIFUGAL FORCE for the cle is equal and opposite to centripetal force, and is exerted by the particle upon its neighboring particles, or upon the axis of rotation. CENTRIFUGAL RESULTANT for a body is the resultant of centrifugal forces of all its particles. Generally, this resultant is not a single force; it may be computed from equations (H) by making $\alpha = 0$ and reversing senses of resultant force and couple.

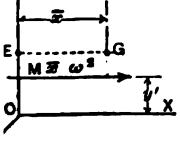


Fig 154

In the three following cases the centrifugal resultant is a single force: Case (*a*) Fig 154. Body has a plane of symmetry, XY, containing axis of rotation and mass center. Centrifugal resultant is a single force $= M\bar{x}\omega^2$, in XY plane, perpendicular to Y-axis, at a distance $\bar{y}' = \frac{\int xy\,dm}{M\bar{x}}$ from the XZ plane, and having sense EG. Case (*b*). Body has an axis of symmetry parallel to axis of rotation. Case (*c*). Body has plane of symmetry

perpendicular to axis of rotation. In cases (*b*) and (*c*), centrifugal resultant is a ... force = $M\bar{x}\omega^2$, acting through mass center outward and perpendicular to axis ... tion ($\bar{x}$ specified in Fig 154).

Center of percussion. Let Fig 155 represent a prismatic bar suspended on a h... zontal *Y*-axis at *O*. *G* is the mass center. If a force *P*, parallel to *X*-axis, is app... the body, the axle reaction at *OD* will generally be inclined t... *Z*-axis at some angle $\pm\beta$, the angle depending on the distance ... from the axis of rotation. If $k = k_z^2 \div \bar{x}$, in which k_z is rad... gyration about *Z*-axis, *P* will cause no *x* component of axle re... that is β will be zero, and the point *C*, where the action line ... intersects *OG*, is the center of percussion. In impact testing ... chines, heavy pendulums are used to deliver blows, and pr... design requires the striking point to coincide with the cent... percussion, in order to avoid shock to the axle and detrim... vibration of the pendulum itself.

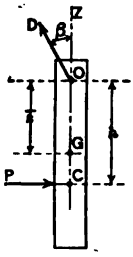


Fig 155

Body having plane motion. This motion has been ... scribed as translation of a base point in the plane of m... and rotation about the base point (Art 52). Choose ... mass center as base point and origin of coordinates ... the *XY* plane as plane of motion. Resultant of all ap... forces may be described as a single force and a couple in the plane of m... The couple is computed from the equation $\Sigma M_z = I_z\alpha = Mk_z^2\alpha$, in which Σ ... represents sum of moments of applied forces about *Z*-axis. I_z and k_z ... moment of inertia and radius of gyration about the *Z*-axis. The single ... acts through the mass center, and its components are given by $R_x = \Sigma F_x = M\bar{a}_x$, $R_y = \Sigma F_y = M\bar{a}_y$, $R_z = \Sigma F_z = 0$. ΣF_x, ΣF_y, and ΣF_z, are sums of co... ponents of all applied forces, and $\bar{a}_x$, $\bar{a}_y$ are components of acceleration of m... center.

56. Work, Energy and Power

Work of a constant force is the product of the force and the effectiv... placement of its application point. Effective displacement of applica... point is the component of the displacement parallel to the force. In Fig 156, the work of force *F* as application point describes path *AB* $= F \times AC$. Since $F\ (AB\cos\alpha) = (F\cos\alpha)\,AB$, the work is also equal to displacement of application point times component of force parallel to the displacement. Work of a variable force is $\int_{s_1}^{s_2} F\cos\alpha\,ds = \int_{s_1}^{s_2} F_t\,ds$, in which *F* is the variable force, *ds* is the elementary length of path, α is angle between force and element *ds*, and F_t is tangential component of *F*. The SIGN OF WORK is plus if force and effective displacement have same sense ... the sign is negative if they differ in sense.

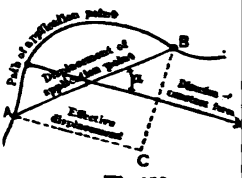


Fig 156

Unit of work is unit force times unit dist... 1 erg = 1 dyne cm. Work done by a body a... a force is equal and opposite to work d... the force on the body.

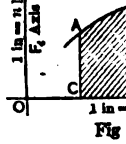


Fig 157

Work diagram (Fig 157). Plot values of F_t a... nates, corresponding values of *s* as abscissas dra... *AB* through ends of ordinates. Area *ABDC* t... equals work, in ft-lb, done by F_t over distance ...

Work of gravity on a body in any motion equals product of weight and ch... height of the center of gravity. Work of a central force *F* (one always directed t...

ed point), in any displacement of its application point, is $-\int_{r_1}^{r_2} F\,dr$, in which r_1 r_2 are the distances of the application point from the center at the beginning and of the displacement.

ork done by a pair of equal, opposite and colinear forces, for any motion of their cation points, A and B. If force on A acts from B toward A, work of the pair $\int_{r_1}^{r_2} F\,dr$; if force on A acts from A toward B, work of the pair $= -\int_{r_1}^{r_2} F\,dr$. In formulas, r_1 and r_2 are the initial and final distances between A and B. Work of a e T on a body for an angular displacement of θ radians $= T\theta$.

ower is time-rate of doing work. Let P = power and w = work; $w = \int_t ds$, and $P = \frac{dw}{dt} = F_t \frac{ds}{dt} = F_t v$, v being speed of application point of force

If P equals power (rate of doing work) of the steam in a locomotive, R = resistance (grade, air, rolling, etc), M, v, and a_t being mass, speed, and leration of train respectively, then $P = Rv + Mva_t$; $P = 0$, for $v = 0$. Power torque $T = T\omega$, ω being angular velocity of rotating body. UNIT of ER is any work unit divided by time unit, such as ft-lb per sec, ft-lb per or erg per sec. (10^7 ergs per sec = 1 watt). One horsepower = 550 ft-lb sec = 33 000 ft-lb per min.

nergy. A body possesses energy when it can do work against forces ap- to it. The amount of energy possessed at any instant is the amount of the body can do in changing to some standard state. KINETIC ENERGY E) of a body is energy possessed by virtue of its velocity, and the standard is zero velocity. K E of a particle is one half its mass times square of velocity. K E of a body in translation = $0.5\,Mv^2$. K E of a rotating $= 0.5\,I\omega^2 = 0.5\,Mk^2\omega^2$, I, k, and ω being moment of inertia, radius of tion, and angular velocity about axis of rotation. K E of a body having e motion = $0.5\,I\omega^2 = 0.5\,Mk^2\omega^2 = 0.5\,M\bar{v}^2 + 0.5\,\bar{I}\omega^2$, in which I and k refer stantaneous axis, $\bar{v}$ = velocity of mass center and $\bar{I}$ is moment of inertia t axis through mass center perpendicular to plane of motion. UNIT of K E me as unit of work, usually ft-lb. In computing K E, engineers use mass ugs, v in ft per sec, ω in radians per sec, and k in ft.

rinciple of work and kinetic energy. Total work of the applied forces ng on any body, or on any system of connected bodies, equals the change he kinetic energy of the body, or bodies. Work done = Δ K E. Change in gy in translation = $0.5\,M\,(v_2^2 - v_1^2)$, v_1 and v_2 being initial and final velocities. Change in energy in rotation = $0.5\,I\,(\omega_2^2 - \omega_1^2) = 0.5\,Mk^2\,(\omega_2^2 - \omega_1^2)$, ω_1 ω_2 being initial and final angular velocities. In plane motion, change in is

$$\Delta \text{ K E} = 0.5\,I\,(\omega_2^2 - \omega_1^2) = 0.5\,Mk^2\,(\omega_2^2 - \omega_1^2)$$
$$= 0.5\,M\,(\bar{v}_2^2 - \bar{v}_1^2) + 0.5\,\bar{I}\,(\omega_2^2 - \omega_1^2) = 0.5\,M(\bar{v}_2^2 - \bar{v}_1^2) + 0.5\,M\bar{k}^2\,(\omega_2^2 - \omega_1^2)$$

which I and k refer to instantaneous axis, $\bar{I}$ and $\bar{k}$ to a parallel axis through s center, and $\bar{v}$ is velocity of mass center.

xample. In example 3, Art 55 (Fig 152): (*a*) Use principle of work and K E to constant torque M necessary to stop body A in 30 ft, from a downward veloc of 1 per sec; (*b*) Find tension in cord P; (*c*) Find power of motor for above value of que, when veloc of A is 10 ft per sec. SOLUTION:

Work of gravity is positive = $1\,248 \times 30 = +37\,440$ ft-lb
Work of axle friction is negative = $-400 \times 3\frac{9}{2} \times \frac{1}{4} = -1\,500$ ft-lb
Work of motor is negative = $-M\theta = -M \times 60 \times 0.5 = -120\,M$ ft-lb
Total work = $-120\,M + 35\,940$
Final K E = 0, and change is a loss, or negative.

For D, $\Delta K E = \frac{1}{2} M k^2 (\omega_2^2 - \omega_1^2) = -\frac{1}{2} \times 285\frac{5}{32} \times 2\frac{3}{4} \times (13\frac{1}{2})^2 = -1\,308$ ft-lb
For C, $\Delta K E = \frac{1}{2} M k^2 (\omega_2^2 - \omega_1^2) = -\frac{1}{2} \times 9\frac{5}{32} \times 9\frac{1}{16} \times (15)^2 = -189.8$ ft-lb
For B, $\Delta K E = \frac{1}{2} M k^2 (\omega_2^2 - \omega_1^2) = -\frac{1}{2} \times 125\frac{5}{32} \times (1.2)^2 \times (13\frac{1}{2})^2 = -162$ ft-lb
For A, $\Delta K E = \frac{1}{2} M \quad (v_2^2 - v_1^2) = -\frac{1}{2} \times 124\frac{5}{32} \times (15)^2 = -4\,387.5$ ft-lb
Total $\Delta KE = -6\,147.3$ ft-lb

(*a*) Since $W = \Delta K E$, $\quad -120 M + 35\,940 = -6\,147.3$; **hence,** $M = 351$ ft-lb
(*b*) When $v_A = 10$ ft per sec, ω of motor $= 40$ radians per sec

Power = torque $\times \omega = 351 \times 40$ ft-lb per sec $= 351 \times 40 \div 550 = 25.5$ hp
(*c*) To find tension in P, apply principle of work and K E to body A alone.

Work $= (-P + 1\,280)\,30 = -30P + 37\,440$
$\Delta K E = \frac{1}{2} M (v_2^2 - v_1^2) = \frac{1}{2} 124\frac{5}{32} (0 - 15^2) = -4\,387.5$ ft-lb

Then, $\quad -30P + 37\,440 = -4\,387.5$; $\quad$ hence, $P = 1\,394.3$ lb

57. Impulse and Momentum

Linear impulse of a constant force F for time $t = F \times t$. If the force var in magnitude and direction, the impulse is computed from axial compone of impulse; and the axial components of impulse are found by taking the ti integrals of axial components of the force. The three axial component impul are $\int_{t_1}^{t_2} F_x\,dt$, $\int_{t_1}^{t_2} F_y\,dt$, $\int_{t_1}^{t_2} F_z\,dt$; the resultant of these is the impulse of t force F. Impulse is a vector quantity; hence the impulse of the force equ the square root of the sum of the squares of the components. The direct cosines of the resultant vector are determined in the usual manner (see resultan of noncoplanar concurrent forces or resultant velocity in curvilinear motio Unit of impulse is unit force times unit time, or lb-sec in engineers' units.

Angular impulse of a force about a line for a time interval dt is the prod of the moment of the force about the line and the time dt. If T represents t moment or torque of the force, the angular impulse for the time interval $t_2 - t_1$ $\int_{t_1}^{t_2} T\,dt$. Unit of angular impulse is unit moment times unit time, or ft-lb in engineers' units.

Sign of impulse. Impulse of a force tending to increase velocity of body to which the force is applied is positive; that which tends to decr velocity is negative.

Linear momentum of a particle is the product of its mass and velo It is a vector quantity and has the sense and direction of the velocity. U of momentum is unit mass times velocity and is equivalent to uni impulse. Linear momentum of a is the resultant, or vector sum, of momentums of its particles. In motion the linear momentum of a b is $M\bar{v}$, M being mass of the body $\bar{v}$ the velocity of its mass center engineers' units this is expressed in ft per sec or lb (force)-sec.

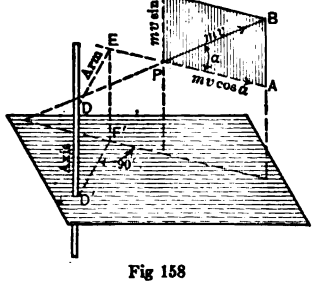


Fig 158

Angular momentum of a parti about an axis is the moment of its m mentum about that axis. In Fig 1 let mv = momentum of particle P solve the momentum into compon parallel and perpendicular to the axis. DE is perpendicular distance fro axis to line AP. The momentum of $P = mv \cos \alpha \times DE$. The angular m

tum of a body about an axis is the algebraic sum of the angular momen- of its particles. The angular momentum of a rotating body about the axis tation is $I\omega = Mk^2\omega$, I and k being moment of inertia and radius of gyration t the axis of rotation and ω the angular velocity. Engineers use I in slug in radians per sec, M in slugs and k in ft; the unit of the quantity is ft² per sec or lb (force)-ft-sec.

rinciple of impulse and momentum for a body. The change in the ponent of linear momentum parallel to any axis x in the time $t_2 - t_1$ equals algebraic sum of the components of the impulses of the applied forces par- to the axis in the same time, or, more briefly, $\Delta (M\bar{v}_x) = \sum \int_{t_1}^{t_2} F_x \, dt$. change in the angular momentum about any axis y in the time $t_2 - t_1$ equals algebraic sum of the angular impulses of the applied forces about the axis e same time, or, more briefly, $\Delta (I_y\omega) = \sum \int_{t_1}^{t_2} T_y \, dt$. This component of r momentum and the angular momentum remain constant if no forces are ied.

ample. In example 3, Art 55 (Fig 152), assume the time required to stop A is 4 sec. rinciple of impulse and momentum to determine tensions P and Q. SOLUTION:

r A, $$\Delta (M\bar{v}_x) = \sum \int_{t_1=0}^{t_2=4} F_x \, dt$$

$^{1249}/_{32}$ (0 − 15) = −585 = +1 248 × 4 − P × 4; hence, P = 1 394.3 lb

r B, $$\Delta (I_y\omega) = \sum \int_{t_1=0}^{t_2=4} T_y \, dt$$

$^{29}/_{32}$ (1.2)² (0 − 7.5) = −43.2 = −Q × 2 × 4 + 1 394.3 × 4 − 400 × $^{3}/_{32}$ × 4;

Q = 1 349.7 lb

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SECTION 37

CHEMICAL NOTES AND TABLES

Table 1. The Chemical Elements (*a*)

Element	Symbol	Atom wt (*b*)	Valence	Element	Symbol	Atom wt (*b*)	Valence
Aluminum	Al	27.1	*iii*	Mercury	Hg	200.6	*i, ii*
Antimony	Sb	120.2	*iii, v*	Molybdenum	Mo	96.0	*iv, vi*
Argon	A	39.88	*o*	Nickel	Ni	58.68	*ii, iii*
Arsenic	As	74.96	*iii, v*	Nitrogen	N	14.01	*iii, v*
Barium	Ba	137.37	*ii*	Osmium	Os	190.9	*iv, viii*
Bismuth	Bi	208.0	*iii, v*	Oxygen	O	16.0	*ii*
Boron	B	11.0	*iii*	Palladium	Pd	106.7	*ii, iv*
Bromine	Br	79.92	*i*	Phosphorus	P	31.04	*iii, v*
Cadmium	Cd	112.40	*ii*	Platinum	Pt	195.2	*ii, iv*
Calcium	Ca	40.07	*ii*	Potassium	K	39.1	*i*
Carbon	C	12.005	*iv*	Radium	Ra	226.0	*ii*
Cerium	Ce	140.25	*iii, iv*	Selenium	Se	79.2	*ii, iv, vi*
Chlorine	Cl	35.46	*i*	Silicon	Si	28.3	*iv*
Chromium	Cr	52.0	*iii, vi*	Silver	Ag	107.88	*i*
Cobalt	Co	58.97	*ii, iii*	Sodium	Na	23.0	*i*
Columbium	Cb	93.5	*iii, v*	Strontium	Sr	87.63	*ii*
Copper	Cu	63.57	*i, ii*	Sulphur	S	32.06	*ii, iv, vi*
Fluorine	F	19.0	*i*	Tantalum	Ta	181.5	*iii, v*
Glucinum	Gl	9.1	*ii*	Tellurium	Te	127.5	*ii, iv*
Gold	Au	197.2	*i, iii*	Thallium	Tl	204.0	*i, iii*
Helium	He	4.0	*o*	Thorium	Th	232.4	*iv*
Hydrogen	H	1.008	*i*	Tin	Sn	118.7	*ii, iv*
Iodine	I	126.92	*i*	Titanium	Ti	48.1	*ii, iv*
Iridium	Ir	193.1	*iv, viii*	Tungsten	W	184.0	*ii, iv, vi*
Iron	Fe	55.84	*ii, iii*	Uranium	U	238.2	*iv*
Lanthanum	La	139.0	*iii*	Vanadium	V	51.0	*iii, v*
Lead	Pb	207.2	*ii, iv*	Ytterbium	Yb	173.5	*iii*
Lithium	Li	6.94	*i*	Yttrium	Yt	88.7	*iii*
Magnesium	Mg	24.32	*ii*	Zinc	Zn	65.37	*ii*
Manganese	Mn	54.93	*ii, iv*	Zirconium	Zr	90.6	*iv*

(*a*) Omitting 22 elements of small interest to engineers. (*b*) Figures for 1916; ... on oxygen = 16.

uping of elements based on similarity in chemical properties (same ons as in Table 1):

HYDROGEN
CHLORINE, bromine, iodine, fluorine
OXYGEN, sulphur, selenium, tellurium
NITROGEN, phosphorus, arsenic, antimony, bismuth
CARBON, silicon
BORON
ALKALIES: lithium, sodium, potassium (ammonium, NH_4)
ALKALINE EARTHS: barium, strontium, calcium
MAGNESIUM, glucinum, zinc, cadmium, mercury
SILVER, copper, gold
ALUMINUM, yttrium, lanthanum, ytterbium, thallium
IRON, nickel, cobalt
MANGANESE
CHROMIUM, molybdenum, tungsten, uranium
TIN, lead, titanium, zirconium, cerium, thorium
VANADIUM, tantalum
PLATINUM, palladium, osmium, iridium

compute percentage composition from a formula. Two examples:

CALCIUM PHOSPHATE $Ca_3(PO_4)_2$

..	3×40.07	$= 120.21 =$	38.7%
..	2×31.04	$= 62.08 =$	20.0
..	$2 \times 4 \times 16$	$= 128.00 =$	41.3
ula-weight,		$= 310.29 =$	100.0

KAOLIN $H_2Al_2(SiO_4)_2, H_2O$

H.....	2×1.008	$= 2.016 =$	0.8%
Al.....	2×27.1	$= 54.2 =$	20.9
Si.....	2×28.3	$= 56.6 =$	21.8
O.....	$2 \times 4 \times 16$	$= 128.0 =$	49.5
H_2O	$2.016 + 16$	$= 18.016 =$	7.0
Formula-weight		$= 258.832 =$	100.0

letermine formula from analysis: divide percentage of each element by nic weight, and reduce quotients to least common denominator. Example:

Phosphorus.....	21.82% ÷ 31.04 = 0.703 = 1	$= Na_2HPO_4$
Hydrogen.....	0.71% ÷ 1.01 = 0.703 = 1	
Sodium.....	32.43% ÷ 23.00 = 1.407 = 2	
Oxygen.....	45.02% ÷ 16.00 = 2.814 = 4	
	99.98%	

putations from analysis will not always work out with mathematical preci- it small errors can be compensated according to the fundamental law that ts always combine in simple proportions represented by whole numbers.

rmining formula of a mineral from its analysis is complicated by st few natural substances are "chemically pure"; certain constituents neral from one locality may be found, in same mineral from a different , to be replaced wholly or partly by another element. First convert age of minor element to equivalent percentage of preponderating element group, equivalents being directly proportional to atomic weights; then e ratio of elements (or molecules) in same manner as above. Example:

sis of a garnet:

.....	39.85%
O_3.....	22.07
O_3.....	1.13
O.....	0.68
).....	36.31
	100.04

$Fe_2O_3 : Al_2O_3 = (2 \times 55.84 + 3 \times 16 = 159.68) : (2 \times 27.1 + 3 \times 16 = 102.2) = 1.13 : x$. $x = 0.72$; that is, 1.13% of Fe_2O_3 is equivalent to 0.72% of Al_2O_3. Similarly, $MgO : CaO = (24.32 + 16 = 40.32) : (40.07 + 16 = 56.07) = 0.68 : x$. $x = 0.94\%$ CaO.

SiO_2.....	$39.85 + (28.3 + 32)$	$= 0.660 = 3$
Al_2O_3.....	$(22.07 + 0.72) \div 102.2$	$= 0.223 = 1$
CaO.....	$(36.31 + 0.94) \div 56.07$	$= 0.664 = 3$

Type formula: $3\,CaO \cdot Al_2O_3 \cdot 3\,SiO_2$

Table 2. Composition, etc, of More Important Industrial Compounds

(For composition of minerals, see Sec 1, Tables I–VIII; and Sec 28, Table 1 gases, see Sec 14, Table 4. For additional organic and inorganic compounds, see Nostrand's "Chemical Annual," Merck's "Index," or U S Pharmacopoeia).

Name	Formula	Mol wt	Sp gr
Acetic acid	$HC_2H_3O_2$	60.03	1.06
Alcohol (grain)	$C_2H_5{\cdot}OH$	46.05	0.785
" (wood)	$CH_3{\cdot}OH$	32.03	0.791
Alumina	Al_2O_3	102.2	3.86
Alum, ammonium	$Al_2(SO_4)_3{\cdot}(NH_4)_2SO_4{\cdot}24\ H_2O$	906.95	1.64
" potassium	$Al_2(SO_4)_3{\cdot}K_2SO_4{\cdot}24\ H_2O$	949.06	1.76
Ammonia	NH_4OH	35.03	0.88
Am chlor (sal-ammoniac)	NH_4Cl	53.50	1.52
Ammonium nitrate	NH_4NO_3	80.05	1.73
" sulphate	$(NH_4)_2SO_4$	132.14	1.77
Arsenious oxide (white arsenic)	As_2O_3	395.84	3.74
Barium carbonate	$BaCO_3$	197.37	4.27
" sulph (blanc fize)	$BaSO_4$	233.44	4.40
Boric acid	H_3BO_3	62.02	1.43
Calcium acetate (acet of lime)	$Ca(C_2H_3O_2)_2{\cdot}H_2O$	176.13	...
" carbide	CaC_2	64.07	2.22
" carbonate	$CaCO_3$	100.07	2.85
" oxide (quicklime)	CaO	56.07	3.30
" hydroxide (slaked lime)	$Ca(OH)_2$	74 09	2.08
" phosphate (phos of lime)	$Ca_3(PO_4)_2$	310.29	3.18
" sulph (plaster paris)	$CaSO_4$	136.14	2.96
Carbon tetrachloride	CCl_4	153.84	1.58
" disulphide	CS_2	76.14	1.29
Cupric arsenite (paris green)	$CuHAsO_3$	187.54	...
" oxide	CuO	79.57	6.37
" sulphate (bluestone)	$CuSO_4{\cdot}5\ H_2O$	249.72	2.28
Ferric oxide	Fe_2O_3	159.68	5 18
Ferrous oxide	FeO	71.84	..
" sulph (copperas)	$FeSO_4{\cdot}7\ H_2O$	278.02	1.90
Hydrochloric acid	HCl	36.47	...
Hydrogen peroxide	H_2O_2	34.02	...
Lead acetate (sugar of lead)	$Pb(C_2H_3O_2)_2{\cdot}3\ H_2O$	379.20	2.50
" carbonate (white lead)	$2PbCO_3{\cdot}Pb(OH)_2$	775.31	...
" monoxide (litharge)	PbO	223.10	9.37
" oxide (red lead)	Pb_3O_4	685.30	9.10
" sulphate....	$PbSO_4$	303.17	6.23
Magnesium carbonate	$MgCO_3$		
" sulph (epsom salt)	$MgSO_4{\cdot}7\ H_2O$		
Manganese dioxide	MnO_2		
Mercuric chlor (corrosive sublimate)	$HgCl_2$		
Nitric acid	HNO_3		
Phosphoric acid	HPO_3		
Potassium carbonate (potash)	$K_2CO_3{\cdot}2\ H_2O$		
" chlorate	$KClO_3$		
" chloride	KCl		
" chromate	K_2CrO_4		
" cyanide	KCN		
" ferricyanide	$K_3Fe(CN)_6$		
" ferrocyanide	$K_4Fe(CN)_6{\cdot}3\ H_2O$		
" hydroxide (caustic potash)	KOH		
" nitrate (saltpeter)	KNO_3		
" permang'n'te	$KMnO_4$		
" sulphate	K_2SO_4		
Silver nitrate (lunar caustic)	$AgNO_3$		
Sodium borate (borax)	$Na_2B_4O_7{\cdot}10\ H_2O$		
" carbonate (soda)	Na_2CO_3		
" bicarbonate	$NaHCO_3$		
" chloride (salt)	$NaCl$		
" cyanide	$NaCN$		
" hydroxide (caustic soda)	$NaOH$		
" nitrate (Chile saltpeter)	$NaNO_3$		
" silicate (water glass)	$Na_2Si_4O_9$		
" sulphate (Glauber salts)	$Na_2SO_4{\cdot}7\ H_2O$		
" sulphite	$Na_2SO_3{\cdot}7\ H_2O$		
" thiosulphate	$Na_2S_2O_3{\cdot}5\ H_2O$		
Sulphur dioxide	SO_2		
Sulphuric acid	H_2SO_4		
Tartaric acid	$H_2C_4H_4O_6$		
Tin chloride	$SnCl_4$		
Vanadium oxide	V_2O_5		
Zinc chloride	$ZnCl_2$		
" oxide (zinc white)	ZnO		
" sulphate	$ZnSO_4{\cdot}7\ H_2O$		

* Table 4.

3. **Solubilities of Certain Salts in Water at 18°** (Professor Alexander Smith)

K	Na	Li	Ag	Ba	Sr	Ca	Mg	Zn	Pb
32.95	35.86	77.79	0.0_216	37.24	51.09	73.19	55.81	203.9	1.49
3.9	5.42	13.3	0.0_410	1.7	3.0	5.4	5.1	9.2	0.05
65.86	88.76	168.7	0.0_41	103.6	96.52	143.3	103.1	478.2	0.598
4.6	6.9	12.6	0.0_46	2.9	3.4	5.2	4.6	9.8	0.02
137.5	177.9	161.5	0.0_535	201.4	169.2	200.0	148.2	419.0	0.08
6.0	8.1	8.5	0.0_71	3.8	3.9	4.8	4.1	6.9	0.0_92
92.56	4.44	0.27	195.4	0.16	0.012	0.0016	0.0076	0.005	0.07
12.4	1.06	0.11	13.5	0.0_292	0.001	0.0_32	0.0_314	0.0_35	0.003
30.34	83.97	71.43	213.4	8.74	66.27	121.8	74.31	117.8	51.66
2.6	7.4	7.3	8.4	0.33	2.7	5.2	4.0	4.7	1.4
6.6	97.16	313.4	12.25	35.42	174.9	179.3	126.4	183.9	150.6
0.52	6.4	15.3	0.6	1.1	4.6	5.3	4.7	5.3	3.16
6.38	36.67	152.5	0.59	0.8	30.0	85.17	42.86	58.43	1.3
0.38	2.2	8.20	0.025	0.02	0.9	2.3	1.5	1.8	0.03
7.62	8.33	80.43	0.004	0.05	0.25	0.25	6.87	0.83	0.002
0.35	0.4	3.84	0.0_314	0.001	0.0_257	0.007	0.26	0.02	0.0_43
42.9	116.4	12.04	0.01	3.7	0.77	0.17	0.001	0.0_25	0.01
18.0	21.0	5.0	0.001	0.22	0.063	0.02	0.0_32	0.0_45	0.0_34
11.11	16.83	35.64	0.55	0.0_223	0.011	0.20	35.43	53.12	0.0041
0.62	1.15	2.8	0.020	0.0_410	0.0_26	0.015	2.8	3.1	0.0_13
63.1	61.21	111.6	0.0025	0.0_338	0.12	0.4	73.0	.....	0.0_42
2.7	3.30	6.5	0.0_215	0.0_415	0.006	0.03	4.3	.....	0.0_25
30.27	3.34	7.22	0.0035	0.0086	0.00146	0.0_256	0.03	0.0_26	0.0_315
1.6	0.24	0.69	0.0_32	0.0_338	0.0_326	0.0043	0.0027	0.0_34	0.0_35
108.0	19.39	1.3	0.003	0.0023	0.0011	0.0013	0.1	0.0047	0.0_21
5.9	1.8	0.17	0.0_21	0.0_311	0.0_47	0.0_213	0.01	0.0_23?	0.0_43

upper figure of each pair is the number of grams of the anhydrous salt held in by 100 c c of water; the lower is the molar solubility, *i e*, the number of moles lar weight expressed as grams) contained in 1 liter of saturated solution. The s for small solubilities have been abbreviated; thus $0.0_44 = 0.0000004$.

e 4. **Specific Gravity of Sulphuric Acid at 15° C,°** Compared with water at 4° C (Lunge)

Deg Baumé	Deg Twaddell	100 parts of c p acid contain			
		SO_3	H_2SO_4	60° Bé acid	50° Bé acid
0.0	0	0.07	0.09	0.12	0.14
6.7	10	6.02	7.37	9.44	11.79
13.0	20	11.71	14.35	18.39	22.96
18 8	30	17.07	20.91	26.79	33.46
24.0	40	22.30	27.32	35.01	43.71
28.8	50	27.29	33.43	42.84	53.49
33 3	60	31.99	39.19	50.21	62.70
37 4	70	36.58	44.82	57.43	71.71
41 2	80	40.91	50.11	64.21	80.18
44.8	90	44.92	55.03	70.52	88.05
48.1	100	48.73	59.70	76.50	95.52
50 0	106	51.04	62.53	80.13	100.00
51.2	110	52.46	64.26	82.34	102.82
54 1	120	55.93	68 51	87.79	109.62
56.9	130	59.45	72.82	93.29	116.51
59.5	140	63.00	77.17	98.89	123.47
60 0	142	63.70	78.04	100.00	124.86
61 8	150	66.58	81.56	104.52	130.49
64.2	160	70.94	86.90	111.35	139.06
65.2	165	74.29	91.00	116.61	145.60
65 5	166	75.19	92.10	118.02	147.36
65 7	167	76.27	93.43	119.72	149.49
65.9	168	78.04	95.60	122.51	152.96

* To reduce to 15° sp gravities observed at other temperatures, for each deg above or below 15° add to or subtract from the sp gr observed:

0.0006	with acids to	1.17	
0.0007	" "	from 1.17 to	1.45
0.0008	" "	" "	1.58
0.0009	" "	" "	1.75
0.0010	" "	" "	1.84

Table 5. Comparison of Thermometer Scales

$$\frac{C}{5} = \frac{F - 32°}{9} = \frac{R}{4}$$ (See also Sec 39, Art 6)

Cent	Fahr	Re	Cent	Fahr	Re	Cent	Fahr	R
−273	−459.4	−218.4	+ 55	+ 131	+ 44	+1 300	+2 372	+1
−200	−328	−160	60	140	48	1 400	2 552	1
−150	−238	−120	65	149	52	1 500	2 732	1
−100	−148	− 80	70	158	56	1 600	2 912	1
− 50	− 58	− 40	75	167	60	1 700	3 092	1
− 40	− 40	− 32	80	176	64	1 800	3 272	1
− 30	− 22	− 24	85	185	68	1 900	3 452	1
− 20	− 4	− 16	90	194	72	2 000	3 632	1
− 17.8	0	− 14.2	95	203	76	2 100	3 812	1
− 10	+ 14	− 8	100	212	80	2 200	3 992	1
0	32	0	200	392	160	2 300	4 172	1
+ 5	41	+ 4	300	572	240	2 400	4 352	1
10	50	8	400	752	320	2 500	4 532	2
15	59	12	500	932	400	2 600	4 712	2
20	68	16	600	1 112	480	2 700	4 892	2
25	77	20	700	1 292	560	2 800	5 072	2
30	86	24	800	1 472	640	2 900	5 252	2
35	95	28	900	1 652	720	3 000	5 432	2
40	104	32	1 000	1 832	800	3 100	5 612	2
45	113	36	1 100	2 012	880	3 200	5 792	2
50	122	40	1 200	2 192	960	3 300	5 972	2

Boiling and melting temperatures, see Sec 39, Art 11.

Coefficient of expansion, see Sec 39, Art 8.

Specific gravities, OF MINERALS, see Sec 1, Art 3, and Tables I-VIII. 28, Table 1; Sec 31, Art 5. Of STONE, EARTH, AND STRUCTURAL MATE see Sec 43, Table 9. Of GASES, see Sec 14, Table 4.

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SECTION 38

ELEMENTS OF HYDRAULICS

BY

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1. Definitions

dromechanics deals with the mechanics of fluids. Fluids embrace LIQUIDS ically incompressible, and the molecules of which are not fixed in position) ASES (compressible and expand spontaneously, volume depends on press mp, and molecules are free). Hydromechanics comprises HYDROSTATICS, g with fluids at rest and HYDRODYNAMICS, with fluids in motion. HY- ics treats of the hydrostatics and hydrodynamics of water, but the term n restricted to hydrodynamics only, or the mechanics of water in motion.

principles and formulas applying to water apply also to other liquids, but are ed by VISCOSITY, or internal friction. This is absent in a perfect liquid, but is t to some extent in water and to a large extent in viscous liquids like syrups, in which article requires a shearing force to move it over adjacent particles. It is important hydrodynamics.

w of the principles of hydrostatics were established by Archimedes (250 B C), other [illegible] (1640), Torricelli and Pascal (1650). Hydrodynamics is a very modern

... requires the study not only of the fundamental l... experiments. Marriotte ... and Newton ... Bernoulli and Chezy in the 18th century, but t... accuracy by numerous experimenters who have ...

Properties of Water

... and quantity of air present in sol... press of 1 atmos, or 1°C decrease in vol...

... at 62.5 lb per cu ft. or 8.355 lb per U S ... lb in some mineral springs; salt w... ... is said to weigh 73: of Great Salt La... ... ft. Wt of distilled water is greatest ... boiling point (59.84 lb).

... barometer (14.7 lb per sq in or 1 atmos... boils at 212.2° F. Boiling point is l... ... press of 20 in (= 192.4° F), water ... barometer drops 1 in per 1 000 ft of eleva... observed in it at different temp. At 60° F th... wt in 100 000 of water.

... units and constants for hydraulic computa... = 62.5 lb.
... lb.
... per sec; $\sqrt{2g} = 8.02$, $1 \div 2g = 0.0...$
... $R = 2.304$ ft head.
... $\pi/6 = 0.049$ sq ft, etc.
... = a flow of 1 cu ft per sec for 12 hr ...

HYDROSTATICS

Transmission of Pressure

... fluid, neglecting effect of gravity, ... transmitted undiminished in all direct... ... ft per sec = velocity of s... ... acts with the same force on al... ... at right angles to the surface.

... stand that great forces could thus be pro... the total press increases in proportio... ... A light press exerted on the wa... ... inserted in a barrel may burst th... ... principle was applied by Bramah (17... HYDRAULIC PRESS (Fig 1). It ... large and small, fitted with pis... ... a pipe through which water pa... ... to other. A force P, applied to th... ... transmitted through the liquid and p... ... press p on the large piston.

... $F = W = PD^2 \div d^2$. If ordinary l... ... packing is used, the area of leather in ... ... the actual area is πDC, and the press is ... ... friction resistance is $\pi\pi DCp$. The rel...

$$\frac{W}{P} = \frac{D^2}{d^2}\left(\frac{1 - 4\mu\frac{C}{D}}{1 + 4\mu\frac{c}{d}}\right).$$

iency $= \left(1 - 4\mu\frac{C}{D}\right) \div \left(1 + 4\mu\frac{c}{d}\right)$. μ is variable, but may be taken at 0.12 to r well-oiled packing.

4. Pressure Produced by Gravity

sidering a liquid divided into layers, the press in each layer is: (*a*) pro- nal to its depth, (*b*) proportional to wt of the liquid, (*c*) the same at all in each layer, (*d*) same in all directions at any depth, (*e*) independent of id shape of containing vessel. Also water surface is normal to the result- the forces acting.

s (*a*) and (*b*), above, are practically self-evident. $P = wAh$, where P = total press, = wt of 1 cu ft of water, lb; A = area, sq ft; h = head, ft. If $p = P \div A$ =

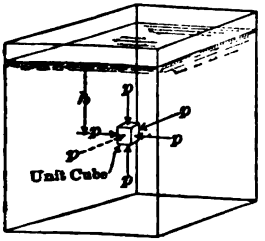


Fig 2

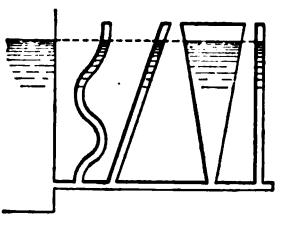


Fig 3

ess, then p in lb per sq ft $= wh = 62.5\,h$, or in lb per sq in $= 0.434\,h$, or h in ft = oes p in lb per sq in. (*c*) and (*d*) follow from Pascal's law (see Fig 2). (*e*) is shown 3, where the bottoms of vessels are at same level, and unit pressures must be same would take place.

5. Total Normal Pressure and Center of Pressure

al normal press, or press acting at right angles to surface $= P_n = wAh_{cg}$; × **area** of surface × distance of center of gravity of the surface below water This applies to any surface plane, curved or 1. Thus the total normal press against the ace of dam shown in Fig 5*a* is $P_n = 62.5 \times h \div 2) = \frac{1}{2}\,wh^2$ lb per lin ft.

al pressure in any direction $= P' = P_n \cos\theta$ h_{cg}, where θ = angle between normal and the lirection. That is, the total press in any direc- total normal press on a projection ($= A'$) per- ular to that direction. In Fig 5*d*, total horiz , $P_h = wh^2 \div 2$, the same as the normal press t a vertical projection (Fig 5*a*).

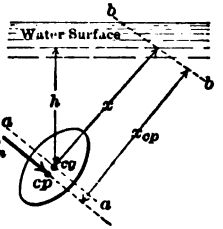


Fig 4

ter of pressure (*cp*) of a submerged surface is the point of application resultant press. For a submerged rectangle the center of press is $\frac{2}{3}\,h$ the water surface, when the top of the rectangle is at the surface (Fig 5*a*) eral the distance $x_{cp} = I \div S$, where I is the moment of inertia of the

face and S its statical moment (= area × distance of cg from axis) both wit reference to an axis situated at intersection of the plane of the surface with th water level. x_{cp} is the perpendicular distance from this axis to the cp.

In applying this formula use is made of the well-known relation $I = I' + Ax^2$, whe I = moment of inertia about any axis, I' = that about an axis through cg of surfac

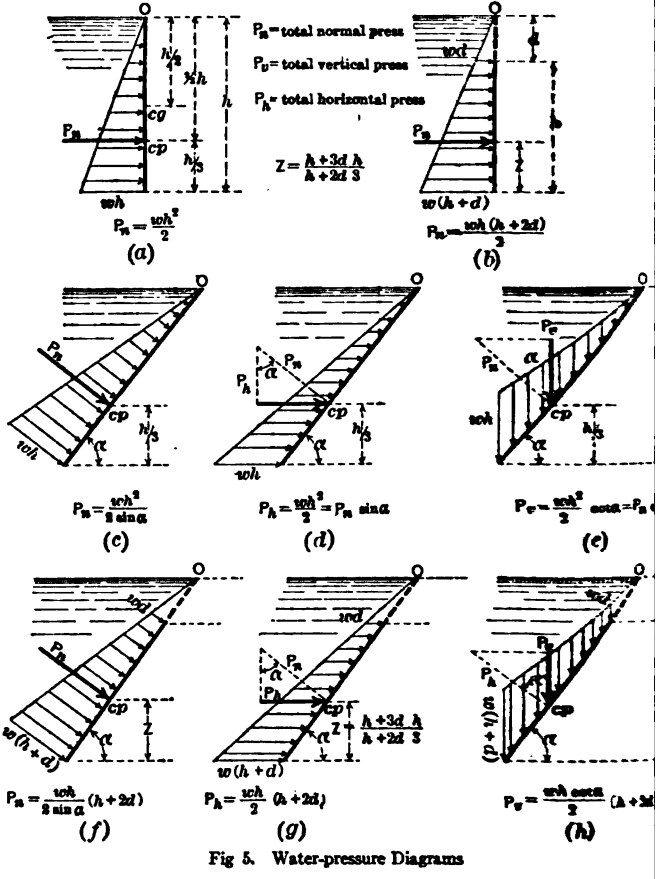


Fig 5. Water-pressure Diagrams

A = area of surface, and x = distance between axes. Thus for circular area of Fig 4 is axis through cg, bb is water surface axis, $A = \pi r^2$, $I' = \pi r^4 + 4$; hence, $x_{cp} = (\pi r^4 + \pi r^2 x^2) + \pi r^2 x = (r^2 + 4x) + x$, or when the circle is vertical $= (r^2 + 4h) + h$.

It is seen that the center of press is lower than the center of gravity, and that the tance between them decreases as the head increases. Practically they may be taken coincident, when the head exceeds 3 or 4 times the vertical dimension of the surface.

6. Pressure against Dams and Gates

ig 5 shows graphically the variation in normal, horiz, and vertical press, total
s and centers of press for rectangular surfaces of unit width. The press at
er surface is 0, and at depth $h = wh$ in all
ctions; hence the variation is uniform and
ltant press acts through the cg of the pres-
figure.

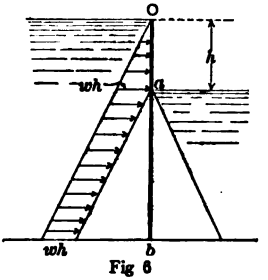


Fig 6

case of press from both sides (Fig 6), the resultant
, represented by arrows and found by drawing the
ss triangles, is uniform in intensity and = wh for
depth below the water surface a. It depends only
he difference in levels h. Thus a gate anywhere
ween a and b would be subjected to the same result-
ress, independent of its depth below a. The point
plication of the total resultant press against the
is best found by taking the difference in moments
e 2 press triangles about b.

7. Pressure in Pipes and Tanks

Hoop tension. Considering the vertical section of pipe in Fig 7*a* as a unit
1) in height, subjected to a uniform internal press of p lb per sq in, this press
luces tension in the pipe = T (Fig 7*b*), which tends to rupture the pipe longi-
tudinally. From the principle that the total
press in any direction = total normal press on a
projection perpendicular to that direction, $2T = pd$, or T (lb per lin in of pipe) = p (lb per sq in) $\times r$ (radius of pipe, in). If f = allowable stress in shell, lb per sq in, then required thickness t (in) = $pr \div f = 0.434\, hr \div f$, where h = head in ft.

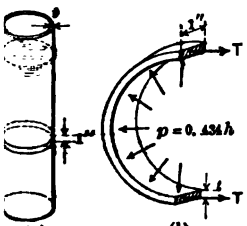


Fig 7. Pressure in Pipes and Tanks

Example. Head, 400 ft; diam, 3 ft; riveted steel pipe; joint efficiency, 70%; f = 14 000 lb per sq in. $t = 0.434 \times 400 \times 18 \div (14\,000 \times 0.70) = 0.32$, or $\frac{3}{8}$ in. This formula holds only for thin shells. The limit is usually put at $t = \frac{1}{20}\, d$. For thick shells, see *Proc* Am Soc Mech Engrs, 1912. Also the press p will only be uniform for a vertical pipe, but may be so
a for a small pipe under a large head. For large pipes, besides the allowance for
nal press, other factors must be taken into account (Art 13). Same principle is used
signing the hoops or bands for wood-stave pipes or water tanks, or for the reinforcing
in reinforced concrete tanks (Art 12).

Flotation and Loss of Weight in Water. (See Merriman, Treatise Hydraulics; also Sec 1, Art 3, and Sec 31, Art 5.)

HYDRODYNAMICS

9. Flow through Orifices and Nozzles

Torricelli's theorem (1644). The theoretical velocity of flow from an orifice is the
e as that acquired by a body falling freely through a height equal to the head of water
he orifice. That is, theoretically the velocity of flow in ft per sec from an orifice
$\sqrt{2gh}$, where g = acceleration of gravity, ft per sec per sec, and h = head on orifice, ft.
formula represents a theoretical limit which can not be exceeded by the actual velocity.

ig 8*a* shows the STANDARD OR SHARP-EDGED ORIFICE. The contracted vein,
ENA CONTRACTA, of a jet issuing from a standard orifice, is due to the fact
t the particles of water converge as they approach the orifice and
e for a short distance beyond it. The contracted section occurs at a

from the orifice of about 0.5 the diam of the orifice. Its diam is about 0.5 that of the orifice and its area about 0.62, this latter figure being the coeff contraction = c'. Experiments show that the average velocity at the contracted section is about 0.98 (= c_1 = coeff of velocity) times the theoretical velocity

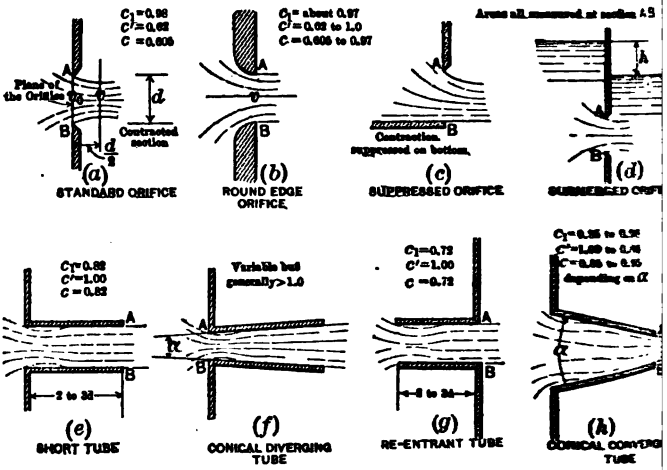


Fig 8. Orifices and Short Tubes

hence the velocity in ft per sec in the contracted section = $c_1\sqrt{2gh}$, the aver being $0.98\sqrt{2gh}$.

Since the discharge, in ft per sec, flowing from the orifice = Q = area × velocit the theoretical discharge = $av = a\sqrt{2gh}$, where a = area of orifice in sq Practically $Q = c'ac_1\sqrt{2gh} = ca\sqrt{2gh}$, where $c = c'c_1$ = COEFFICIENT OF DI CHARGE, the average value of which is 0.605. Theoretically the shape of or he on the orifice has no effect. But experiments show that c is greater for low hea than high, greater for rectangles than squares, and greater for squares th circles, and varies from 0.59 to 0.63 or more. Averages: for rectangular ori 0.604, for circular 0.597. For accurate work use the tabulated values given text books on Hydraulics. The preceding figures apply to the contracted s tion. A fundamental law of hydrodynamics, the EQUATION OF CONTINUIT is that for steady flow equal masses or weights of a fluid must pass all cr sections in equal times. This reduces to equal volumes for water, since it practically incompressible. Hence, the discharge in the plane of the orifice m equal that at the contracted section = $ca\sqrt{2gh}$, or since $v = Q \div a$, the velo in the plane of the orifice = $v_0 = ca\sqrt{2gh} \div a = c\sqrt{2gh} = c'v$.

In these formulas the HEAD ON THE ORIFICE should be taken as the head o center of press, but if the head is greater than 2 or 3 times the vertical dimensi of the orifice, the head on the center of gravity of the orifice may be used. F a circular orifice the head on the center of press = $(r^2 + 4h) \div h$, where r radius and h = head on cg. For rectangular orifices a convenient formula small heads is $Q = c\,\frac{2}{3}\,b\sqrt{2g}\left(h_2^{\frac{3}{2}} - h_1^{\frac{3}{2}}\right)$, where b = breadth, h_2 = depth to bo tom and h_1 = depth to top of orifice. This allows for difference between cp and

oss of head in an orifice = $h' = (1 - c_1^2)\, h = (1 + c_1^2 - 1)\, v^2 \div 2g = (1 + c_1^2 - 1)\, v_0^2 \div 2g$. For a standard orifice, $c_1 = 0.98$, $c' = 0.62$ and $h' = 0.04$. $h = 0.041\, v^2 \div 2g = 0.11\, v_0^2 \div 2g$.

ound-edge orifices (Fig 8*b*). By rounding the inner edge of an orifice c' may have value between 0.62 and 1.0, depending on how closely the rounding corresponds to the our of the jet. An orifice for measuring discharge should not be round-edged, because ncertainty in its value of c, but the increase in Q due to rounding may often be taken ntage of. Aver value of c for round-edge orifice = 0.8.

uppression of contraction occurs when the orifice edges are nearer than 3 times the : dimension of the orifice from sides or bottom of tank. Preventing the jet from con- ting increases c' and hence the discharge. K (coeff of discharge for a suppressed ce) $= c\,(1 + 0.15\,n)$ for rectangular and $= c\,(1 + 0.13\,n)$ for circular orifice; where coeff for standard orifice with full contraction and nP = portion of perimeter having ontraction, P being the entire perimeter.

ubmerged orifices. EFFECTIVE HEAD is the difference in level between the 2 water ices, and the action of the orifice is independent of its depth below the lower water ice. c is slightly smaller than for free discharge into air and is usually taken as 0.60.

'elocity of approach. In above formulas h is taken as constant, but this occur only when the inflow Q is constant. The water then approaches the ice with an initial velocity, and the effective head is $H = h + h_v$, where $h_v = \div 2g$ and v_a = velocity of approach. For a standard orifice,

$$v = c_1 \sqrt{\frac{2gh}{1 - c^2(a/A)^2}} \text{ and } Q = a \sqrt{\frac{2gh}{(1/c)^2 - (a/A)^2}}$$

re a = area of orifice and A = that of tank. When $a \div A$ is 0.2 or less, the r due to neglecting v_a is negligible. In the case of water approaching a large ice in a canal or conduit, the formula for a rectangular orifice under small d is $Q = c\, \tfrac{2}{3}\, b \sqrt{2g}\, [(h_2 + h_v)^{3/2} - (h_1 + h_v)^{3/2}]$, which is the basis for the ncis weir formulas with v_a.

ime required to empty any vessel of uniform horiz cross-section is t (sec) $= 2A\,(\sqrt{H} - \sqrt{h}) \div [ca\sqrt{2g}]$, where A = cross-sectional area of vessel, a = ice area, and H = initial and h = final head on orifice.

low under pressure may be estimated by the same formulas given above, pressure being first reduced to equivalent head.

xample. Find Q when an orifice 3 in diameter is first opened in a tank containing 3 ft r with 50 lb per sq in gage air pressure on top. Effective head $= 3 + 50 \times 144 \div$ $= 118$ ft. $Q = 0.60 \times (0.785 \div 16) \times 8.02 \times \sqrt{118} = 2.52$ cu ft per sec.

or fluids other than water the corresponding wt per cu ft is used to determine the tive head, and c must also be modified. Thus, in preceding example, if the fluid is cury, $c = 0.62$ and wt = 850 lb per cu ft, or $h = 3 + 50 \times 144 \div 850 = 11.5$, and 0.80 cu ft sec. Same formula is used to find the discharge of air from an orifice, the f being about 0.60.

hort tubes are short sections of cylindrical or tapering pipe, used for connecting two els or other special purposes. Fig 8 shows some common forms, with their coeffs.

lozzles are of the smooth or ring type. The former is the commoner. The

harge is given by $q = 29.83\, D^2 \sqrt{\dfrac{p}{\left(\frac{1}{c}\right)^2 - \left(\frac{D}{d}\right)^4}}$, where q = discharge, gal per

, p = press at the nozzle, lb per sq in, c = coeff of discharge = c_1 for a smooth zle, since there is no contraction, and $c' = 1$. (For smooth nozzle, c

from 0.990 for a 1-in to 0.997 for a 6-in nozzle.) D = diam of nozzle, in, and diam of pipe. If q is required in cu ft per sec, use 0.0665 instead of 29.83. T formula allows for velocity of approach, which is sometimes considerable. I is measured by a Pitot gage held in the nozzle (Art 19), $q = 29.83\ cd^2\sqrt{p_1}$, wh p_1 = press recorded on gage in lb.

10. Flow Over Weirs and Dams

Contracted weir (Fig 9) with end contractions is a rectangular notch in upper edge of a vertical wall. The length of notch is less than that of the ch nel, and, that the end contracti be complete, there should be a di ance of not less than 3 H on the si and the crest should be at least from bottom of channel. If the is the full width of the channel traction is suppressed on the ends it is known as a SUPPRESSED WEIR ing no end contractions. There many weir formulas. The rational formula is $Q = c\ \frac{2}{3}\ b\sqrt{2g}\ H^{\frac{3}{2}}$, where a constant or coeff of discharge, b = breadth, and H = head in ft. It is deri from the orifice formula by letting the head on the top of the orifice = 0. Fra uses $Q = 3.33\ bH^{\frac{3}{2}}$ for a suppressed weir (Table 1), and for a contracted we

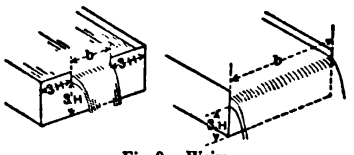


Fig 9. Weirs

Table 1. Discharge in Cu Ft per Sec per Ft of Length of Thin-edge Weirs from $Q = 3.33\ bH^{\frac{3}{2}}$.

Head H, ft	0.00	0.01	0.02	0.03	0.04	0.05	0.06	0.07	0.08	0.09
0.0	0.000	0.003	0.009	0.017	0.027	0.037	0.049	0.062	0 075	0 [illegible]
0.1	0.105	0.121	0.138	0.156	0.174	0.193	0.213	0.233	0.254	0 [illegible]
0.2	0.298	0.320	0.344	0.367	0.391	0.416	0.441	0.467	0 493	0 [illegible]
0.3	0.547	0.575	0.603	0.631	0.660	0.689	0.719	0.749	0 780	0 [illegible]
0.4	0.842	0.874	0.906	0.939	0.972	1.005	1.039	1.073	1 107	1 [illegible]
0.5	1.177	1.213	1.248	1.285	1.321	1.358	1.395	1.433	1 471	1 [illegible]
0.6	1.548	1.586	1.626	1.665	1.705	1.745	1.785	1 826	1 867	1 [illegible]
0.7	1.950	1.992	2.034	2.077	2.120	2.613	2.206	2.250	2 294	2 [illegible]
0.8	2.383	2.428	2.473	2.518	2.564	2.610	2.656	2.702	2.749	2 [illegible]
0.9	2.843	2.891	2.938	2.986	3.035	3.083	3.132	3.181	3 231	3 [illegible]
1.0	3.330	3.380	3.430	3.481	3.532	3.583	3.634	3.686	3.737	[illegible]
1.1	3.842	3.894	3.947	4.000	4.053	4.107	4.160	4.214	4 268	4 [illegible]
1.2	4.377	4.432	4.487	4.543	4.598	4.654	4.710	4 766	4.822	4 [illegible]
1.3	4.936	4.993	5.050	5.108	5.165	5.223	5.281	5.340	5 398	5 [illegible]
1.4	5.516	5.575	5.635	5.694	5.754	5.814	5.874	5 935	5 996	6 [illegible]
1.5	6.118	6.179	6.240	6.302	6.364	6.426	6.488	6.551	6 613	6 [illegible]
1.6	6.739	6.803	6.866	6.930	6.994	7.058	7.122	7.186	7 251	[illegible]
1.7	7.381	7.446	7.512	7.577	7.643	7.709	7.775	7.842	7.908	[illegible]
1.8	8.042	8.109	8.176	8.244	8.311	8.379	8.447	8.515	8.584	[illegible]
1.9	8.721	8 790	8.859	8.928	8.998	9.068	9.137	9.207	8 278	[illegible]

considering the effect of end contractions as reducing effective length of the w by 0.1 H on each end, $Q = 3.33\ (b - 2\ H)\ H^{\frac{3}{2}}$. For Miner's Inch see Art 1

Velocity of approach affects weir discharge, and for accurate work sh kept below 1 or 2 ft per sec. The effect of v_a can be allowed for by adding t actual head H on the weir $1.5\ h_v$, which is obtained from Table 2, using average velocity of approach v_a as an argument.

Table 2. Head h_v Due to Velocity of Approach v_a

0 4	0.6	0.8	1.0	1.2	1.4	1.6	1.8	2.0	2.2
0.002	0.005	0.010	0.015	0.022	0.030	0.040	0.050	0.062	0.075
2 4	2.6	2.8	3.0	3.2	3.4	3.6	3.8	4.0	.....
0.089	0.105	0.122	0.140	0.150	0.179	0.201	0.213	0.248	.....

amples. (*a*) Suppressed weir, no velocity of approach, $b = 4$ ft, $H = 1.73$ ft. Find
From **Table I**, Q per ft of length $= 7.577$ and total $= 7.577 \times 4 = 30.31$ cu ft sec.
Contracted weir, same as above. Q per ft of length $= 7.577$, but effective length $- 0.2\ H = 4 - 0.35 = 3.65$ ft, and total $Q = 27.65$ cu ft per sec.
Contracted weir as above, with average velocity of approach of 2 ft per sec. From e 2, $h_v = 0.062$ and $1.73 + 1.5 \times 0.062 = 1.82$, which from Table 1 gives 8.176 in ft., or a total of $8.176 \times 3.65 = 29.80$ cu ft per sec.

obtain accurate results, the weir must be constructed in a standard manner, similar at used in obtaining the experimental coefficients: the notch must have sharp edges as standard orifice; crest must be level sides vertical; length should be between d $3\ H$; velocity of approach should be ! (area of cross-section of channel of ach should be at least $6\ bH$); sides crest should be $3\ H$ from sides and om of channel; head should be meas- accurately, 4 to 6 ft upstream, to elim- effect of slope in the water surface rd the weir (for common work a stake be driven in the bed of the channel of ach, Fig 10, and cut off level with the crest, H being obtained by measuring water level on a graduated rod held on this stake. For accurate work a hook gage ld be used); for suppressed weirs there must be free access of air to the under side e overflowing sheet of water or NAPPE.

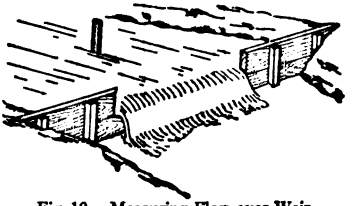
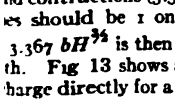


Fig 10. Measuring Flow over Weir

mple. Design a contracted weir to measure the flow of a stream 6 ft wide by 2 ft , the aver velocity of which, obtained roughly by a float, was 3 ft sec. Q = approx ft per sec. For a depth of 1 ft Q (Table 1) $= 3.33$ cu ft per sec per ft of length, and equired length would be $36 + 3.33$, or 11 ft, which is too long (see above). By further using Table 2, it is found that a weir 8 ft long, with $H = 1.25$ ft, will give the required atio about 1 : 6.4). Crest of the weir must be about 4 ft ($3\ H = 3.75$) above bottom tream and width of channel of approach at least 16 ft ($8 + 2 \times 3 \times 1.25 = 15.5$). e channel is narrower when flooded, it should be widened or a suppressed weir used.



11. Triangular Weir

Triangular weir (Fig 11) is convenient for small quantities of water. It should have sharp inner corners, sides of equal slope, and bottom angle should be a right angle. For this weir $Q = 2.53\ H^{5/2}$, H being measured from bottom of notch.

rapezoidal or Cippoletti weir (Fig 12) saves time in computations, as a e may be made or a stick graduated to read the discharge directly, as for rtangular suppressed weir. Sides are sloped to make the discharge through triangular end portions equal to the loss due nd contractions $(3.33 \times 0.2 \times H \times H^{3/2})$. End es should be 1 on 4, as shown. Formula $3.367\ bH^{3/2}$ is then used, b being the bottom th. Fig 13 shows a stick graduated to read harge directly for a trapezoidal weir 1 ft long 1).

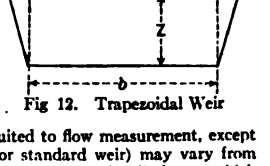
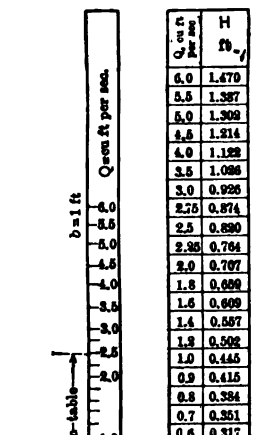


Fig 12. Trapezoidal Weir

pillway dams are a type of weir, but are seldom suited to flow measurement, except roximately, as the coeff in Francis formula (3.33 for standard weir) may vary from to 4.2. A sloping upstream face on the dam generally increases the discharge, which

is also modified by width and shape of crest and inclination of downstream face. In signing spillways, the flood flow Q is approximately known, the breadth b is usually by the flood width of channel below the dam, and the able flood height H over the spillway may be computed Francis formula. As it is difficult to determine the charge, an ample factor of safety is desirable, and the used should be small enough to be on safe side (3.3 is mon coeff for masonry spillways)



Q cu ft per sec	H ft.
6.0	1.470
5.5	1.387
5.0	1.309
4.5	1.214
4.0	1.122
3.5	1.036
3.0	0.926
2.75	0.874
2.5	0.830
2.25	0.764
2.0	0.707
1.8	0.659
1.6	0.609
1.4	0.557
1.2	0.502
1.0	0.445
0.9	0.415
0.8	0.384
0.7	0.351
0.6	0.317
0.5	0.280
0.4	0.242
0.3	0.200
0.2	0.152
0.1	0.096

Fig 13. Measuring Stick for Weirs

11. Flow in Pipes and Open Channels

Bernoulli (1738) stated the law of steady flow. "... any section of a tube or pipe, under steady flow with friction, the press head plus the velocity head is equal to the hydrostatic head that obtains when there is flow." This law, together with the equation of continuity (Art 9) and a formula for loss of head in friction is fundamental in determining flow conditions in pipes.

Loss of head by friction in pipes is inconsiderable in short tubes, and is provided for in the coeff of discharge; for long pipes it is important and as the length increases, assumes most of the available head. The proposed formulas for loss of head in friction are so numerous, and some so complicated, that the engineer may be at a loss which to use. All reduce practically to one of the two forms given below; their chief value is not in their form, but in the fact that coefficients have been determined from experiments on certain kinds of pipes with use of a particular form of formula, so that in designing a given pipe the equation for which the best coefficients are available should be used.

Experimental laws of friction head. The loss of head: (a) is directly proportional to the length l of the pipe; (b) is inversely proportional to the diam d; (c) increases nearly as the square of the velocity v; (d) increases with the roughness of the interior surface; and is independent of the pressure. These laws are expressed by $h_f = f \frac{l}{d} \cdot \frac{v^2}{2g}$, where $2g$ is introduced so that h_f is expressed in terms of velocity head, which is convenient. Experiments show that the loss is not exactly proportional to v^2. For low velocities the loss is due to shearing action and varies simply as v. When critical velocity is reached, beyond which the flow is not smooth but turbulent, the loss being of an impact as well as shearing nature, and the variation expressed by v^2 (approx). Reynolds gives for the critical velocity, $v = 0.0...$ where v = ft per sec and d = diam in ft. Since the loss is not accurately expressed by v^2 and the exponent of v should really vary, two types of formula have been developed: first, those which reduce to the above form, where v is squared and the variation is allowed for by making f vary with v and also with d as found in experiment; second, the so-called exponential formulas, in which f is constant and the exponents of v and d are varied to suit aver conditions.

Other losses occur, but can generally be disregarded. Thus the ENTRANCE LOSS = $m \frac{v^2}{2g}$, where $m = 1$ for an inward projecting pipe and 0.5 for a pipe set flush with reservoir or tank wall. CURVATURE also causes a loss, that is least for a 90° bend if radius is about $3d$. For a 90° bend in 6-in pipe, the loss is nearly equal to that in a length of straight pipe, and for 30-in pipe in a length of 40 ft. For intermediate sizes loss varies roughly with diam.

LOSS IN EXPANSION from a small pipe with velocity V_1 to large pipe with velocity V_2 is $(V_1 - V_2)^2 \div 2g$. LOSS IN CONTRACTION from large to small pipe may be taken

$+ 2g$, where V = velocity in small pipe. Loss in gates and valves is uncertain, f full gate is about the same as in a length of straight pipe of 6 d.

draulic gradient. The conditions of flow in a pipe line may be clearly i by a graphical construction. Thus, Fig 14 shows a uniform diam pipe, urging freely in the air at With no flow the hydro- head at $A = h$. With k (from Bernoulli's the- $= h_p$, the press head, the velocity head, and, lering friction, $+ h_f$, or ad lost in pipe friction,

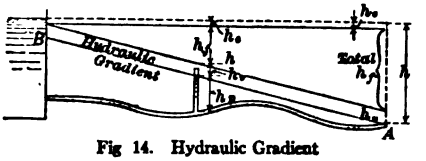


Fig 14. Hydraulic Gradient

Since there is no press head at A when there is flow, $h = h_v + h_f$, and for ondition this is true if h be taken as head causing flow. The lost head for ipe is made up of 2 parts: that lost in entrance h_e, and that lost in friction id, as friction loss varies directly with length and velocity head is constant the pipe diam is uniform, the different losses may be represented by the ical construction shown. Line AB is the hydraulic gradient, and repre- the height to which water would rise in a pressure tube inserted in the pipe y point. Fig 15 shows the hydraulic gradient for a compound pipe, neglect- ntrance loss and velocity head.

elocity of discharge may be obtained from the above formulas. In case of shown in Fig 14, with no bends, gates or valves, $h = h_v + h_f + h_e = \frac{v^2}{2g} + \frac{v^2}{2g} + 0.5\frac{v^2}{2g}$, or solving for $v = \sqrt{2gh \div \left(1.5 + f\frac{l}{d}\right)}$. Usually the velocity and entrance loss are so small compared to friction loss that the entire head y be assumed as consumed in friction, and $v = \sqrt{2gh \div f\frac{l}{d}}$. Discharge $= 6.30\sqrt{hd^5 \div fl}$. Diameter for a given discharge and head is $d = 0.4789 \div h)^{1/5}$. Though these formulas are fundamental they are seldom used in orm, as proper value of friction factor f is less well known than for other forms. ew smooth pipe $f = 0.02$, while for old pipe it may be 0.04 or even higher. sual formulas for flow are given below, following explanation of terms used.

ydraulic radius, or hydraulic mean depth $R = A \div p$, where A = area ss-section of flow and p = wetted perimeter, or length of boundary of cross- n in contact with water. For a circular pipe flowing full, $A = \pi d^2 \div 4$ and d, or $R = d \div 4$. For a rectangular flume 6 ft wide and 3 ft deep, $A = 18$, 2, and $R = 1.5$ ft. Hydraulic radius is a term that can be used with both and closed channels and hence is preferable to d.

pe of the hydraulic gradient S is expressed in various ways, usually as the sine of the of the hydraulic gradient and hence practically $= h \div l$, or the available head divided gth of pipe or channel. It is sometimes given in ft per mile, per cent, or ft per 1 000.

ezy formula (1775) is the most widely used formula in hydraulic work. probably the best available for designing canals, ditches, and large conduits pipes, while the Hazen-Williams formula (see below) is well suited to com- tions for smaller pipes. Chezy gives $v = C\sqrt{RS}$, where v = mean velocity, r sec, R = hydraulic radius, ft, S = sine of slope of hydraulic gradient $= h \div l$, C is a coeff depending on roughness of the pipe or channel, etc. By substi- g $d \div 4$ for R and $h \div l$ for S, this formula is evidently identical wi- above, except that f is substituted for the term $8g \div C^2$.

Kutter's formula. Value of C in the Chezy formula varies with smoothness of the channel and with R and S. Various attempts have been made to ... a formula for C, the best known being that of Ganguillet and Kutter (... ("Flow of Water in Rivers and other Channels." Transl by Hering and Trautwine, Wiley, N Y.)

$$C = \frac{\frac{1.811}{n} + 41.65 + \frac{0.00281}{S}}{1 + \frac{n}{\sqrt{R}}\left(41.65 + \frac{0.00281}{S}\right)}$$

where R and S are as above and n is a factor depending only on the roughness of pipe or channel. Kutter gave the following values for n: 0.009 for well-planed timber; 0.010 for neat cement; 0.011 for cement with ¼ sand; 0.012 for unplaned timber; 0.013 for ashlar and brickwork; 0.015 for unclean surfaced sewers and conduits; 0.017 for rubble masonry; 0.020 for canals in very firm gravel; ... for canals free from stones and weeds; 0.030 for canals with some stones and weeds; ... for canals in bad order.

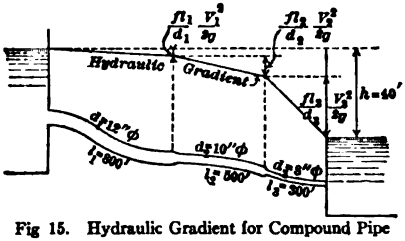


Fig 15. Hydraulic Gradient for Compound Pipe

Kutter's formula is purely empirical, but has been so widely accepted that several ... have been devised to facilitate its application. The most satisfactory is a diagram such as the accompanying, which solves the Chezy formula using the Kutter formula for C.

The diagram, Fig 16, by K. R. Kennison, (*Eng News*, June 20, 1912), was published with explanation in extended form, 1913. Example 1. A wood-stave pipe is 6 ft diam and 2 miles long. Available head, 10 ft. If n is 0.011, what are Q and v? $R = \frac{6}{4} = 1.5$. $S = 0.95$ ft per 1 000. Enter lower part of diagram at A, with $R = 1.5$, and ... (as shown by arrows in Fig 16a) to intersection with "n" line for $n = 0.011$ (interpolated) and run vertically upward to intersection with "S" line in upper part, where the velocity (=5.7 ft per sec) is read on the vertical scale. $Q = 0.785 \times 6^2 \times 5.7 = 161$ cu ft per sec. Example 2. An ordinary riveted steel pipe, 4 ft diam and 5 000 ft long, is to carry 60 cu ft per sec. $n = 0.016$. What head is required? $R = 1.0$ ft, $A = 0.785 \times 4^2 = 12.55$ sq ft, hence $v = 60 \div 12.55$ or 4.8 ft per sec. Start at A as before, running upward from the intersection with the "n" line and locating a point corresponding to B in Fig 16a on this vertical line and on a horiz line representing $v = 4.8$. The position of B with respect to the slope or "S" lines gives the required slope = 2.5 ft per 1 000, or a total head of 12.5 ft. For other problems see Art 15. (Note that when s is small several lines are given for each slope, to be used for different values of n.)

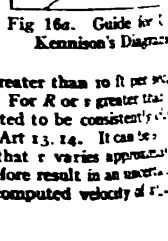
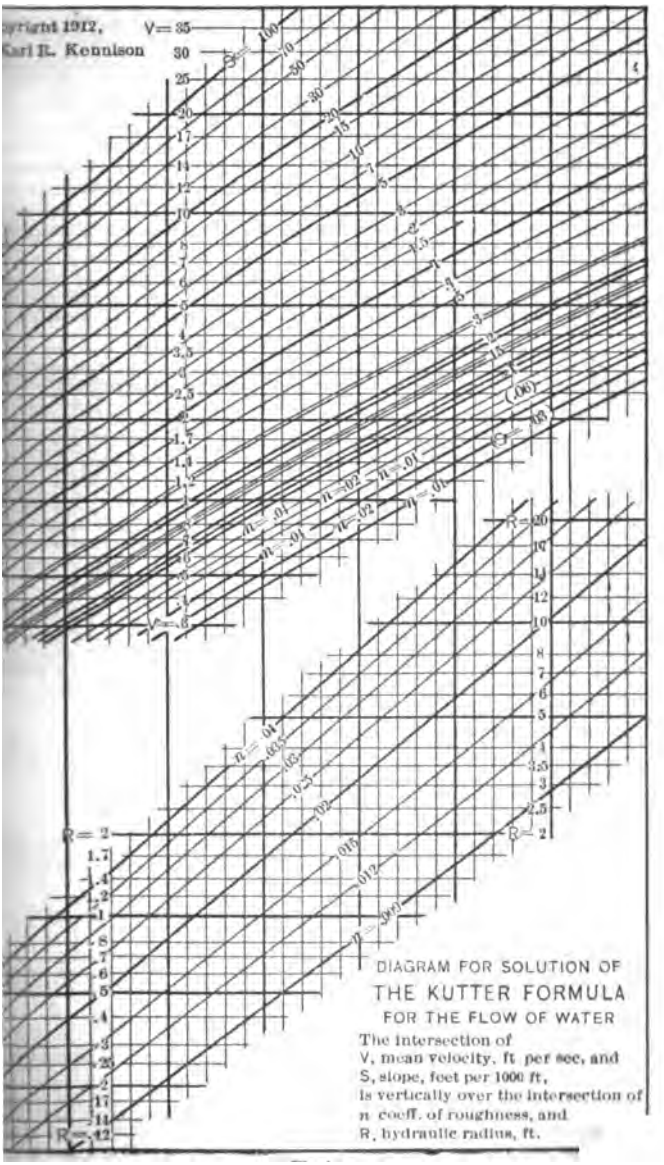


Fig 16a. Guide for ... Kennison's Diagram

Limits of application. Kutter's formula is best used for velocities between 1 and 6 ft per sec, but is fairly reliable up to 10 ft. For hydraulic radii greater than 10 ft, velocities greater than 10 ft per sec, slopes flatter than 1 in 10 000, it should be used with caution. For R or v greater than ... it is unreliable. Results from the formula must not be expected to be consistently ... than 5%. For values of n recommended for use in design see Art 13. 14. It can be ... that S varies as n^2 for values of R greater than 1 ft, and that v varies approximately inversely as n. An uncertainty of x% in selecting n will therefore result in an uncertainty in a computed slope of twice this amount, or $2x$%, and in a computed velocity of ...


Copyright 1912, Carl R. Kennison
DIAGRAM FOR SOLUTION OF
THE KUTTER FORMULA
FOR THE FLOW OF WATER
The intersection of
V, mean velocity, ft per sec, and
S, slope, feet per 1000 ft,
is vertically over the intersection of
n coeff. of roughness, and
R, hydraulic radius, ft.

Hazen-Williams formula (Hydraulic Tables. Wiley, N Y, 1905) is an ponential formula, $v = 1.318\,cR^{0.63}S^{0.54}$, based on following reasoning Ch formula may be written $v = CR^{0.5}S^{0.5}$, and if its exponents agree perfectly w the facts the value of C would depend on roughness only. But, the values the exponents must vary with different surfaces, diameters, slopes, etc. Has Williams formula uses exponents representing aver conditions; the value of which varies with different surfaces and sections, but has a smaller range than is selected from experience, not computed by a formula. The quantity c is introduced simply to make the value of c about the same as that of the Kut or Chezy constant C. Hence, Hazen-Williams formula is probably less accur than the Chezy; but, since in most computations it is simply a question of sele ing a pipe of some standard size, great accuracy is not necessary.

Hazen-Williams advise following values of c: for very best C-I pipe, 140; good new pipe, 130; tuberculated C-I pipe, 80 to 110; designing C-I pipe, 100; new riveted s pipe, 110; ordinary W-I pipe, 100; lead, brass, tin pipe, 140; smooth wood pipe, 1

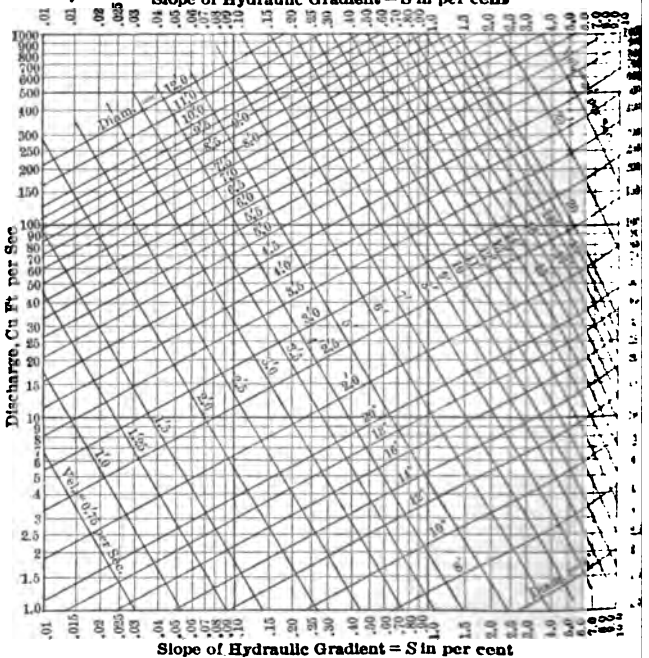


Fig 16b. Diagram for Solving Hazen-Williams Formula

brick sewers, 100; vitrified pipe, 110; smooth clean masonry, 140; slime-coated masor 130; ordinary good masonry, 120.

The diagram, Fig 16b (S. D. Bleich, *Eng Rec*, Nov 30, 1907), is convenient for s Hazen-Williams formula. It is constructed for $c = 100$; for other values of c m the required discharge by $100 + c$, and the resulting value of q must be used to get size of pipe and velocity. Then multiply this velocity by the ratio $c \div 100$ to get

alue. **Example 1.** A C-I pipe is to carry 10 cu ft per sec, length 1 000 ft, available head Using $c = 100$, find required size? $S = 1$ ft per 100, and running vertically from this in the diagram to the intersection with the horiz line representing a discharge of 10 cu ft c, the intersection is found to be slightly below the 18-in diam line, indicating that an pipe is needed, and the mean velocity is found to be slightly less than 6 ft per sec. ple 2. A wood pipe is 4 000 ft long, and is to discharge 60 cu ft per sec with 20-ft ble head. Find d and v. In this case $c = 120$. Multiply Q by 100 ÷ 120, giving $Q =$ d enter the diagram with this value and a slope $S = 0.5$ ft per 100, as before. Hence, 8 in and $v = 6.5$ ft per sec. Actual v will be $6.5 \times 1.20 = 7.8$ ft per sec.

mpound pipes. Algebraic expressions can be derived giving exact relations een discharge and head for compound pipes, but it is simpler to solve these ems by determining size of a single pipe that will be equivalent to the given ination; that is, such a size as will give same loss of head for a given discharge.

example, find the discharge from the compound pipe in Fig 15. Take $c = 100$ in -Williams formula. Assume any convenient and reasonable Q, say 3 cu ft per sec. Hazen-Williams diagram find for the 8, 10, and 12-in pipes the required grades or head in per cent, viz, 5, 1.8, and 0.7 for this Q, which, multiplied by the respective s, give 15, 9, and 5.6 ft, or a total of 29.6 ft necessary head. As the total length is ft, this represents an aver loss of 1.85 ft per 100, and the uniform pipe which would discharge of 3 cu ft per sec with this hydraulic gradient would be (from diagram) 9.8 in diam. Actual available head is 40 ft ($S = 2.5\%$), and Q for a 9.8-in pipe his head is 3.5 cu ft per sec, the required value.

nch pipes, as shown in Fig 17*a*, may be solved similarly. First find size of uniform o replace *BCDE*, as above. This is found to be 8.8 in. Now the loss of head be- *B* and *E* must be the same by outes, *BE* and *BCDE*. Assume asonable loss of head, say 15 ft, ind discharge by each route. e *BE* pipe (10 in, 600 ft long), .5 ft per 100, and from diagram .7 cu ft per sec. For the 8.8-in 200 ft long, $S = 1.25$ and $Q =$ The total discharge = 5.5 cu ft c, and the diam of pipe 600 ft hich will deliver 5.5 cu ft per der a head of 15 ft is found he diagram to be 11.5 in. The m is thus reduced to that in 5, which when computed as for compound pipes, assuming f 6 cu ft per sec, gives uniform f 11.9 in, the discharge for 2 400 hich under a 50-ft head is 5.2 per sec.

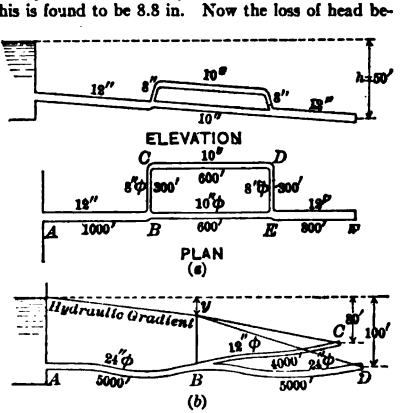


Fig 17. Branch Pipes

17*b* shows another problem in pipes. It may be solved alge- lly, but more easily by trial. A portion y of the head up to the fork must be used up in causing a discharge through at will just equal the sum of the discharges through *BC* and *BD* under the remaining $80 - y$ and $100 - y$. Value of y is best found by trial. From Fig 17*b*, since lengths id *BD* are same, it is evident that y must be more than 50, and by a few trials it is that $y = 58$ will give satisfactory results, the total discharge being 23 cu ft per sec = 2.6 and *BD* = 20.5).

pen channels. The usual formula for flow is the Chezy, with the Kutter given above. In open channels or conduits, not flowing under pressure, the ulic gradient coincides with water surface. R may be obtained from cross- ns, and is generally about equal to mean depth of channel. Right-angle bends sually considered as causing a loss equal to $v^2 \div 2g$, the velocity head, which e small, since velocities are very low. For design of canals, see Art

DESIGN

12. Kinds of Pipe

Standard sizes. In computing the necessary diam for a pipe no great de of refinement is ordinarily necessary, as it reduces to a question of selecti standard size. The relative discharge capacity of pipes vary as the five ha power of the diam (about) and is shown below, using a 4-in pipe as 1. An pipe will carry 6.5 ÷ 3, or over twice as much as a 6-in pipe, hence no very a rate computations are necessary to determine which is required.

Diam.....	4	6	8	10	11	14	16	20	24	30	
Relative capacity...	1	3	6.5	12	20	30	43	80	130	235	3

Cast-iron pipe. Standard sizes are 4, 6, 8, 10, 12, 14, 16, 18, 20, 24, 30, 3 48, 54, and 60 in, usually made in 12-ft lengths, with bell and spigot (Fig 1 flanged joints (Sec 41, Art 7). The latter is use pump connections, etc, or wherever it may be occas ally necessary to remove sections of pipe. Bell and s is common for water and gas mains. The joint is m by first packing with oakum, then pouring a collar of around the bell, which is afterwards calked tight (T.

Fig 18. Standard C-I Pipe, Bell and Spigot Joint

Cement joints are commonly used for gas mains. Pipe is made in standard and thicknesses (Table 3). For large orders any desired thickness is obtain

C-I pipe is cast in 12-ft lengths, bell down, and cleaned and dipped in hot tar and ba oil for a protective coating. C-I pipe is subject to formation of tubercles of rust or inside (particularly when used for impure water or water containing free CO_2), seriously affects its carrying capacity, but usually without greatly imparing its str Probably Hazen-Williams formula (Art 11) is the best to use in design, and while the of c may be as high as 140 for new clean pipe, about 100 must be used if the carr capacity is to be maintained over a period of years. Care should be taken in layin pipe that "bell holes" are dug for the joints, so that the pipe will have a firm suppor its entire length and not rest on the ends; for large sizes the back fill should be well ta under and around the pipe and for a depth of 2 or 3 ft over it, to relieve the pipe excessive external loading. Ashes in refill should be avoided.

C-I pipe has an aver life of 40 to 80 years or more. Cost varies considerably, avera between $26 to $30 a ton. Special shapes, as Ys, Ts, curves, blow-off branches average $60 a ton.

Wrought-iron pipe was formerly made in considerable quantities, but has been pr cally replaced by mild steel (frequently sold as wrt iron). The relative durabi of two materials has long been a matter of dispute, but since in 1916 only 1 or 2 U S make a real W-I pipe, there is seldom opportunity to choose between them.

Steel pipe is of several forms: welded, ordinary riveted, spiral rivete lock bar. BUTT OR LAP-WELDED PIPE is made of mild steel, from ¼ to 15 in. inal diam (actual internal diam varies. Sec 41, Art 7).

It is used for steam and water connections in mechanical work, for interior p water supply, gas, heating, etc, and for supply pipe lines and pumping. It is the st pipe for most service, when the required size is 10 or 12 in or less, except street supply mains. Quite recently the manufacture of lap-welded pipe of larger sizes ha developed; it is now made up to 72 in diam, and is used in water-power work heads, where the saving in metal due to its greater joint strength and perfectly s interior make it more economical than riveted steel pipe.

Riveted steel pipe is used chiefly in large sizes and for high heads. made of curved sheets, the longitudinal edges of which are joined usually double-riveted lap joint, although in larger sizes butt joints double or t riveted with single or double cover plates are used.

Table 3. Standard Thicknesses and Weights of Cast-iron Bell and Spigot Pipe (U S C-I Pipe & Foundry Co)

Class A 100-ft head, 43 lb press			Class B 200-ft head, 86 lb press			Class C 300-ft head, 130 lb press			Class D 400-ft head, 173 lb press			Approx lb lead per joint, 2 in thick	Approx lb hemp per joint
Thickness, in	Lb per		Thickness, in	Lb per		Thickness, in	Lb per		Thickness, in	Lb per			
	Ft	L'th		Ft	L'th		Ft	L'th		Ft	L'th		
.39	14.5	175	0.42	16.2	194	0.45	17.1	205	0.48	18.0	216	6.00	0.18
.42	20.0	240	0.45	21.7	260	0.48	23.3	280	0.52	25.0	300	7 50	0.21
.44	30.8	370	0.48	33.3	400	0.51	35.8	430	0.55	38.3	460	10.25	0.31
.46	42.9	515	0.51	47.5	570	0.56	52.1	625	0.60	55.8	670	13.25	0.44
.50	57.1	685	0.57	63.8	765	0.62	70.8	850	0.68	76.7	920	16.00	0.53
.54	72.5	870	0.62	82.1	985	0.68	91.7	1 100	0.75	100.0	1 200	19.00	0.61
.57	89.6	1075	0.66	102.5	1230	0.74	116.7	1 400	0.82	129.2	1 550	22.00	0.81
.60	108.3	1 300	0.70	125.0	1 500	0.80	143.8	1 725	0.89	158.3	1 900	30.00	0.94
.64	129.2	1 550	0.75	150.0	1 800	0.87	175.0	2 100	0.96	191.7	2 300	33.80	1.00
.67	150.0	1800	0.80	175.0	2 100	0.92	208.3	2 500	1.03	229.2	2 750	37.00	1.25
.76	204.2	2450	0.89	233.3	2800	1.04	279.2	3 350	1.16	306.7	3 680	44.00	1.50
.88	291.7	3500	1.03	333.3	4000	1.20	400.0	4800	1.37	450.0	5 400	54.25	2.06
.99	391.7	4700	1.15	454.2	5450	1.36	545.8	6550	1.58	625.0	7 500	64.75	3.00
10	512.5	6150	1 28	591.7	7100	1.54	716.7	8600	1.78	825.0	9 900	75.25	3.62
.26	666.7	8000	1.42	750.0	9000	1.71	908.3	10900	1.96	105.0	12 600	85.50	4.37

Class E 500-ft head, 217 lb press			Class F 600-ft head, 260 lb press			Class G 700-ft head, 304 lb press			Class H 800-ft head, 347 lb press			Approx lb lead per joint	Approx lb hemp per joint
Thickness, in	Lb per		Thickness, in	Lb per		Thickness, in	Lb per		Thickness, in	Lb per			
	Ft	L'th		Ft	L'th		Ft	L'th		Ft	L'th		
.58	42.5	510	0.61	44.3	531	0.65	48.1	577	0.69	50.5	606	21.9	0.22
.66	60.9	731	0.71	66.8	802	0.75	72.3	868	0.80	76.1	913	28.2	0.28
.74	86.9	1043	0.80	92.8	1114	0.86	101.4	1 217	0.92	107.3	1 288	34.5	0.34
.82	114.6	1375	0.89	122.8	1474	0.97	136.2	1 634	1.04	144.4	1 733	40.8	0.40
.90	145.6	1747	0.99	158.8	1905	1.07	175.1	2 101	1.16	187.5	2 250	47.1	0.46
.98	180.7	2168	1.08	196.5	2358	1.18	218.0	2 616	1.27	233.8	2 805	53.4	0.52
1.07	221.8	2662	1.17	239.3	2872	1.28	268.2	3 218	1.39	287.8	3 453	59.7	0.57
1.15	265.8	3190	1.27	287.3	3448	1.39	321.8	3 862	1.51	345.8	4 149	66.0	0.65
1 31	359.1	4309	1.45	392.3	4707	1.75	479.8	5 758	1.88	510.6	6 127	79.4	0.78
1.55	530.9	6371	1.73	588.8	7065	....	.....	....	....	....	....	122.9	0.93
1.80	738.1	8857	2.02	821.0	9852	....	.....	....	....	....	....	146.7	1.11

s up to 30 in diam are sometimes made of single sheets in 12 or 15-ft lengths. Larger re made with single-riveted lap joints. Field joints between the lengths are made eral ways. For low pressure and pipes up to 40 in diam, a slip joint may be used. ve is placed inside and attached to one end of the pipe, which is covered with burlap vas coated with red lead or asphaltum and then driven into the end of the next length eld by wire ties. For higher pressures, the pipe ends are riveted to forged steel s, connected by bolts and gaskets. Fig 19 shows several kinds of joints. Bolted are also used, which not only act as expansion joints but also allow a slight deflection ermit straight pipe to be laid on easy curves. Lengths of large pipe are usually d together in the field. For design of riveted joints see Sec 43, Art 29.

iral-riveted pipe is frequently used for smaller heads and in sizes up to 42 ec 43, Art 29).

s made in 15-ft lengths galvanized, and 30 or 40 ft, asphalt coated. Each length is ingle sheet, rolled with overlapping edges forming a single-riveted spiral seam. This an efficient joint against bursting, and the seam is said to be the strongest part

pipe. In large sizes it is not made thicker than 0.25 in, and though used for orda water or low-head pipe lines, finds its best application in construction work where its s strength as a b saves necessity of quent support, for its strength, the lightest steel made.

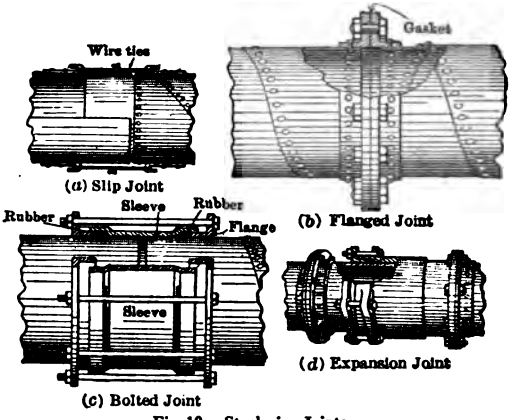


Fig 19. Steel-pipe Joints

Lock-bar pi a form of steel in which a 100 ficient joint is tained by bendi plate of steel t proper form. edges, first u are inserted in open grooves "lock bar" or H-section, whi closed over the set edges by a draulic press.

Life of steel pipe depends on so many factors that it can be given only general terms. Unlike cast iron, which rusts more or less uniformly, steel o pits through in places. Thin steel pipes, ⅛ in or even less in thickness, lasted 15 or 20 years. A thickness not less than 0.25 in should give a use of 25 years or more.

Carrying capacity of small sizes of steel pipe may be found by the Hazen-Wi formula (Art 11). For large, riveted pipe, with rivet heads and seams projecting o side, the Chezy formula is often used with $n = 0.016$. Formula $V = 1.34 D^{0.7} H^{0.555}$ is also used, where V = veloc, ft per sec, D = diam, ft, and H = loss of head, ft per 1 000.

Wood pipe has 2 forms: continuous-stave pipe, built in place, and machine-banded pipe made in lengths in factories.

Continuous-stave pipe has been built up to 14 ft diam; it is rarely made less than 2 ft diam. It is best adapted to moderate pressures, from a minimum of 20 ft head (considered necessary to keep the wood saturated and prevent decay) to a max of 200 to 250 ft. Advantage for large diam pipe: low cost, great carrying capacity ($n = 0.009$ to 0.011 in Kutter's formula) easy transport of material, curves of moderate radius readily made, less liable to freeze than greater strength against deformation than thin steel. Staves are milled with radia to form true circular inside and outside surfaces. They have a length of 10 ft and averaging 16 ft, and are placed so as to break joint by 2 ft. Joints between ends are made by metal tongues inserted in saw kerfs (Fig 20). Hoops are usually of mild steel, in one piece for pipes up to 54 in diam. They have button head on one end cold-rolled threads for 5 in or more on other end, which is often upset to make thread

Fig 20. Continuous Wood-stave Pipe

as body of bar. Ends of hoop rest in a malleable iron shoe, and the band is cinched a hexagonal nut. All metal work is heated and dipped in a mixture of asphalt and oil, and is touched up after placing. Pipe is built by using an outside form for lower building up both sides at same time, and an inside form for upper half resting on half. Sharp curves are to be avoided. A radius of 60 times the diam is the usual m, but this has been exceeded. MATERIAL FOR STAVES is usually Redwood, Douglas Oregon yellow pine. Redwood is considered by many to be too soft. Staves are from standard thickness of lumber for each diam.

Diameter of pipe	Nominal thickness of staves
10 to 14 in	1.5 in thick, 4 in wide
16 to 48 in	2 in thick, 6 in wide
54 in	2.5 in thick, 8 in wide
60 to 72 in	3 in thick, 8 in wide

ual dimensions of staves vary with each diam. The only computation made is for ed spacing of bands. BANDS. If safe strength of band (lb) = S, then $S = fvt^2$, f = safe tensile strength, usually 15 000 lb per sq in, and r = radius of band, in. may cause failure by breaking in tension, or by crushing the wood fibers. The safe ing load on wood can in this case be taken as 650 lb per sq in. Width of bearing of on wood is taken = t, and some specifications require that band be hammered into od until this bearing is secured. Hence, safe load on band would be $S = (R + t)$ where R = radius and t = thickness of pipe, in. For economical design, diam of should be such that values of S figured from either formula are equal, or $r = 0.0138$ t). Diams of ⅜, 7/16, ½, ⅝, ¾, and ⅞ in are used. SPACING OF BANDS is determined e hoop tension due to water press, and by the tension produced by swelling of the

Using the minimum S (see above) the spacing of bands, in = $S + (pR \times et)$, p = water press, lb per sq in, and e = swelling force of the bands, which cannot l the compressive strength of wet wood and may be taken as 100 to 125 lb per sq in. OF WOOD PIPE is very variable, depending on conditions under which it is used. A pipe line, with a cover of 2 ft or more of clean soil free from alkali or vegetable r and maintained under a head of 20 ft or more, will have a longer life (25 to 50 than a line above ground. Under less favorable conditions, life is 6 to 15 years, he pipe is best supported on cradles or sills. Connections are made by inserting Ts or branches in the line, to which the wood pipe is firmly secured. Approx cost in dollars, of fir continuous-stave pipe, erected exclusive of earthwork or cradles, is:

m, in.	24	30	36	42	48	54	60	66	84
ft head	1.40	1.50	1.60	1.75	2.25	2.75	3.30	4.00	5.25
" "	1.75	1.90	2.00	2.25	2.80	3.25	4.00	5.00	6.50

achine-banded pipe is built in the factory in s of 8 to 20 ft and diam ranging from 6 to 48 in, y not over 24 in. Pipe is of staves with edges d to form tongue and groove joints, banded with galvanized steel wire, wound spirally by machine high tension. Wire is from No 8 to No 0, size pacing being regulated according to the press. teel bands are also used. The outside of the pipe vily coated with hot asphalt and tar, and rolled dust to make it better to handle and ship. The ed joint (Fig 21*a*) is for pipes up to 14 in diam. oden collar (Fig 21*b*) or wood-sleeve coupling is on larger sizes. This form has been used for supply pipes, instead of cast iron, also in hy- ic mining, and is useful for same service as con- us pipe, but where smaller sizes are required.

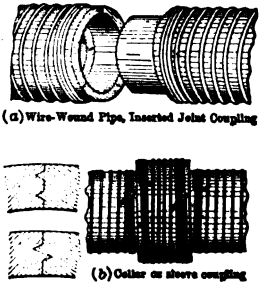


Fig 21. Machine-banded Wood Pipe

ment pipe has been used for drains, culverts, and istributing water under heads of less than 15 or . The mixture is 1 : 3 or 4 Portland cement and fine pit or bank gravel, l molds. A thin interior coating of neat cement is advisable to make the

permeable. Joints a
1 : 2 cement mortar,
mortar is removed, a

(a)

(b)

Fig. 21.

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mula $t = \frac{1}{2} d\sqrt{h \div f}$ may be used in design as a rough check, t, d, and f, in, being as for hoop tension, and h = depth of fill, ft. Wt of fill is say 100 lb per cu ft, and h ally taken at 0.5 to $\frac{2}{3}$ the total depth, to allow for arch action. The amount and ution of load is uncertain. Formula assumes the load to be uniformly distributed he upper half of the pipe, and that it is resisted by uniform press over the lower half. roduces a moment in the shell M, in-lb per in of length = $\frac{1}{16} Wd$; where W = total er linear in of pipe and d = diam, in. A concentrated load produces a positive nt $M = 0.159\ Wd$ at top and a negative moment = $0.091\ Wd$ at sides. Careless ll may cause some other mode of loading (*Bull* No 22, Univ of Ill Expt Sta, 1908).

ernal pressure, due to the pipe being placed above the hydraulic nt, or to lack of or inadequate air valves, has often caused collapse of thin pipes. It is important only in lines of this kind.

n the internal press becomes less than atmospheric, the shell is subjected to a uni- xternal press = p = difference between atmospheric (14.7 lb per sq in) and the te internal press; this causes circumferential compression in the pipe shell. The ing pressure for steel pipe, lb per sq in = p = about $50\,200\,000\,(t \div d)^2$. Air of too small a size may require a greater suction head than this amount in order to air fast enough to fill the pipe when it is emptied quickly, intentionally, or due to . It is generally more economical to provide one or several air valves of ample area summits, than to increase the pipe thickness or to provide flanges to increase its ing strength (*Eng Rec*, May 23, 1915, p 594).

ading stresses due to wt of pipe and water are important in large steel nder low heads. They are usually provided for, not by increasing the ess of shell, which is uneconomical, but by placing a concrete saddle under pe, or by riveting to the pipe at short intervals curved hoops made of with an outstanding edge.

mum moment occurs at bottom of the pipe, and is due to the water load. M, r lin ft of pipe = CwR^3, where C depends on angular length of the saddle (the angle er of pipe subtended by the saddle) and has the following approx values: for $C = 0.75$; $\alpha = 15°$, $C = 0.56$; $\alpha = 30°$, $C = 0.42$; $\alpha = 45°$, $C = 0.32$. W = ater, lb per cu ft (= 62.5) and R = radius of pipe, ft (*Eng Rec*, Jan 31, 1914, p 133). g R and α, it is possible to test a pipe for bending due to this cause, the fiber stress hell being found by the usual formula (Sec 43, Art 4), and if necessary the size and required for angle stiffeners are computed.

perature range of a pipe depends on whether it is covered and on its size. A short e of large diam full of water has a temp range nearly equal to that of the water 32° to about 80° F); a long covered pipe line of small diam has a temp range nat of the soil, which depends on depth of cover and seldom exceeds 40° F for a r. If uncovered and with low flow, a small pipe may have a range almost equal f an empty pipe. Variations in temp produce longitudinal stresses in a pipe that so as neither to contract nor expand, the direction (tension or compression) and of which depend on the temp at which the pipe was laid. These stresses may ckling, and in many exposed pipes EXPANSION JOINTS are provided at intervals top of saddles are made smooth to allow expansion and contraction without exces- ion. Fig 19*d* shows a common expansion joint. These joints are usually placed w the anchorages on exposed pipe lines, and as anchorages occur at all bends at expansion joint is required on each straight portion of the pipe, and intermediate n joints are desirable on very long stretches of straight pipe.

14. Design of Pipe Lines. Valves and Anchorages

eral pipe lines should be kept below the hydraulic gradient (Art 11). If above it siphon action is created, which is undesirable because: (*a*) it s with the action of the pipe line and usually causes decreased or uncer- harge; (*b*) the pipe at high points is under external pressure and liable se. A profile of the proposed line should therefore be carefully studied ng the hydraulic gradient.

Fig 23 shows profile of a pipe line. If a uniform diam is used from *E* to *B*, laid on face of the ground (*ECB*), and the pipe discharges freely at *B*, the hydraulic gradient run from *A* to *B*, provided the pipe is first fi and then a valve at *B* opened. Part of the near *C* will be above the hydraulic gradient, therefore subject to what is termed nega press, but which is really negative only in sense of its being less than atmospheric, 5 ft than zero. As long as *CF* is not over 34 ft, the pipe will act as a siphon, discha under the head *h*. But it is almost impo to make the pipe tight, and air will collect and finally break the suction action. The

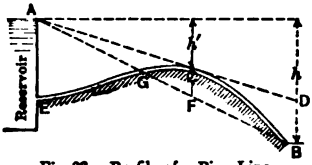


Fig 23. Profile of a Pipe Line

will then discharge at *C*, under the head *h'*, the hydraulic gradient running from *A* to *C*, a certain quantity of water *Q*, less than the remaining portion *CB*, could discharge under head $h - h'$. The portion *CB* will therefore carry off this *Q*, flowing only partly full.

This action may be avoided in several ways: (*a*) Valve at *B* may be partly closed causing the gradient to rise to *D* and acting as an orifice, the discharge of which under head *DB* just equals the discharge of pipe under head $h - DB$. This does away with negative press at *C*, but the discharge of the line is less than with siphon action. Such throttling of the valve at *B*, even if it is not opened more than the necessary amount stated, may cause the "accelerating gradient" to fall below *C* until the water gets in motion, normal conditions are reached, so that air valves should be placed at *C* if there is danger of collapse; (*b*) if constant operation is unnecessary and negative press not liable to cause collapse, the valve at *B* may be closed at intervals which puts the pipe at *C* under a press equal to head *h'*. A valve at *C* may then be opened and air allowed to escape. It would be advisable in this case to place a large air receiver at *C*, to collect the air, a tank above the pipe and not decrease the pipe section at this point; (*c*) a trench may be dug from *G* to *B*, to keep the pipe below the gradient, but this may be costly; (*d*) a compound pipe may be used, and this is often the most satisfactory method. A large pipe used from *E* to *C*, to deliver say 10% more water under the head *h'* than a smaller size, running from *C* to *B*, will deliver under the head $h - h'$.

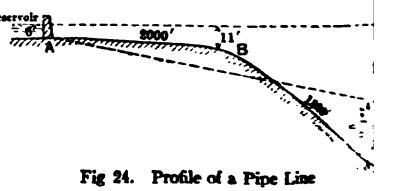


Fig 24. Profile of a Pipe Line

Example (Fig 24). Pipe line from *A* to *C* is to deliver 2 cu ft per sec at *C*, with a terminal press of 25 lb per sq in (= 25 × 2.30 = 57.5 ft head). If the pipe delivers this quantity when reservoir is empty, available head causing flow = 130 − 6 − 57.5 = 66.5 ft. For a 12-in pipe, with slope 1.75 per 100, and *C* = 100, *d* = 8.6 in (Fig 16*b*); hence, use a 12-in pipe. Using this, the gradient will be *AC'*, with siphon action present. Also a 14-in pipe is required to keep gradient above *B*, and flow at *C* must be throttled. a compound pipe for *BC*, the available head = 130 − 11 − 57.5 = 61.5 ft (slope per 100) and nearest standard size (Fig 16*b*) is 8 in. With this head an 8-in pipe will discharge 2.4 cu ft per sec. For the *AB* section the available slope (reservoir empty) is 0.25 ft per 100, and a 14-in pipe gives 2.5 cu ft per sec.

Fig 25 shows hydraulic gradient for conditions arising in pumping through pipes. (*a*) shows conditions in pumping from *A* to *B* against static head *AC*, and that the pump under total head *AD*. In a pipe line for pumping it is usually economical to keep velocity (and hence friction head) low; say between 1.5 and 3 ft per sec, depending on cost of power of pumping. In designing line *AB*, an economical velocity is computed or assumed, and diam of pipe found for required discharge. Friction head is then found from diagram.

In case *b*, if pipe *AFB* is once filled it would discharge under head *AC*, as in Fig *a*, hydraulic gradient running from *A* to *B*, provided distance *EF* does not exceed about 34 ft. If water is pumped through pipe *AFB*, the friction head is *DC* and the pump acts against head $DC - AC$, provided *FE* is as stated above. If *FE* be greater, the hydraulic gradient changes to that shown in case *c*, the pump then acting under head *AE*, and the part between *F* and *B* carrying off the discharge at *F* and flowing only partly full.

gative press is not permissible at *F*, the hydraulic gradient must be kept above *F*. in be done: I, by increasing the head against which the pump is working to head *AG*), which involves a smaller ith increase of velocity and head for the same dis- ; II, by designing a pipe n *A* and *F* that will give an nical velocity and using a om *F* to *B* that will just off this discharge, under *H*; III, by placing a valve nd throttling the flow as in . Here the gradient may sed to line *EG* and kept *F*. In this case air valves be placed at *F*. If it is to take advantage of gra- *DB* (case *b*), steel pipe is ary for the part above the nt, made thick enough to and the difference in press 3), or C-I pipe might be There must be a valve at prime the siphon.

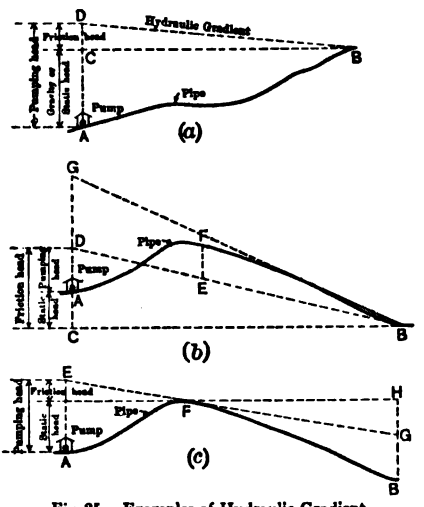
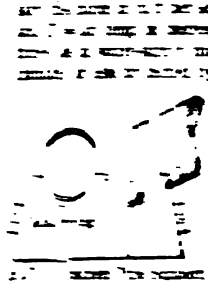


Fig 25. Examples of Hydraulic Gradient

ves. Fig 26 shows the of a pipe line, with loca- f valves. AIR VALVES, to t escape of air on filling, ntrance of air on emptying, aced at every summit and it-off valves. For a shallow reservoir best type of air inlet is a simple standpipe, above water level in reservoir. Air valves are often automatic; float valves have sed, automatically permitting escape of air collecting at summits. Air collects at mmits, and must be removed, as by ordinary gate-valves operated by hand. The ity of ample air valves to prevent formation of vacuum in steel pipe lines has been sed. BLOW-OFF VALVES are placed at low points in pipe lines for emptying the when necessary and for removal of sediment. These are gate-valves, placed on a pipe (about ⅓ the diam of main pipe) leading to discharge in a stream or waste el. SHUT-OFF OR STOP VALVES (ordinary gate-valves) are placed at end of pipe, and, g pipes, at intervals of 1 to 2 miles along the line. This permits inspection and repair section, without emptying the entire line, and in case of breakage water may be shut nearest valve, thus preventing wastage and serious damage. These valves are usually l at summits, with air valves on each side. A lock and chain are put on all valves to at tampering; on large lines the valves are in covered manholes with locks. For large diam lines, a small by-pass pipe with separate valve is added; it is left open to relieve the press while the large valve is being closed. Cost of gate-valves increases rapidly with the sizes; a saving in cost at a slight loss in head may tained by decreasing the diam of pipe at points where valves are placed by using le reducers and tapering increases. CHECK-VALVES are used mainly on pump lines, at where breakage would permit large backward flow of water, as at foot of inclines (at *A*, 5a, *b* and *c*), and on inlet pipe just outside tanks or standpipes. For small sizes a alve is used. For pipes larger than 24 in, a diaphragm or valve plate is cast in an ted section of the pipe, and a number of small valves attached to holes in the plate. all by-pass is sometimes used to avoid heavy water-hammer, caused by sudden g of valve with pulsations of pump. RELIEF (SAFETY) VALVES are occasionally used ls of long pipe lines, or wherever water-hammer is especially to be feared The g disk valves, opening outwards and held in place by springs adjusted to the

Fig 26. Profile of a Pipe Line

Freezing [illegible] gives trouble when there is continual flow. [illegible] Roche's formula is sometimes used to find [illegible] $q = 0.000045\,TA$, where A = exposed area [illegible] T below 32°. Where velocities are very [illegible] pipes must be buried (3 to 5 ft, accord[illegible] [illegible] backing.



Pipe supports are required where pipe is placed [illegible] ground surface. Wood supports are common for [illegible] pipe; for steel, concrete supports (Fig 27) are usu[illegible] Lumber between supports is roughly estimated by [illegible] ing the pipe as a continuous beam, loaded with its [illegible] plus wt of water, and using a low fiber stress.

Anchorages are necessary for exposed pip[illegible] all bends where the resultant press may tr[illegible] out of line, at valves subjected to heavy stat[illegible] water-hammer press, and on steep slopes.

[illegible] are usually anchored by wire or metal ties, carried to [illegible] [illegible] mass of concrete). Fig 28 shows a concrete anch[illegible] [illegible] and F. Pressures on the pipe [illegible] [illegible] create an outward thrust $P = [illegible]$, where a = area of pipe and h = [illegible] [illegible] also the centrifugal force acting [illegible] $= \frac{W}{g}\frac{v^2}{r}$, where W = wt in lb of water [illegible] between the end faces A and B, [illegible] and r = radius of curva[illegible] [illegible] also be necessary to take into [illegible] the resultant sliding force caused by [illegible] [illegible] and earth press against [illegible] these forces, combined [illegible] must intersect its center of gravity, [illegible] middle third of base (Sec 43, Art 15).

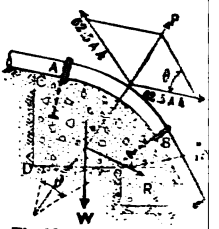


Fig 28. Concrete Ancho[illegible]

15. Design of Ditches and Canals

[illegible] the word "ditch" is used to include "canals."

Hydraulic radius. Where the cross-sec area of a channel and its grade [illegible] [illegible] the greatest hydraulic radius (least [illegible] [illegible] greatest velocity and discharge. For this re[illegible] [illegible] hydraulic radius = 0.5 the depth.

[illegible] the depth should be 0.5 the breadth; [illegible] it also gives [illegible] least cost for a flume); it also gives [illegible] [illegible] least cost for a rock cut). For a trapezoidal sec[illegible]

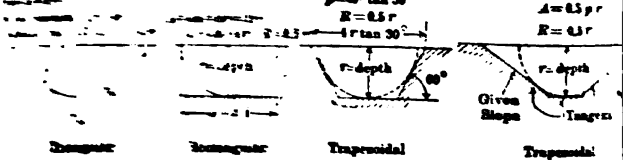


Fig 29. Cross-sections Having Most Advantageous Elements

[illegible] semicircle [illegible] half of a hexagon; or, if side slopes are assumed, the best [illegible] a circle tangents to a half circle having a radius equal to the depth (Fig 29

alined ditches in earth. The CROSS-SECTIONAL PROPORTIONS in Fig 29 not used, because they give sections that are too narrow and deep. Such ons are unstable; they will widen out on the sides and silt up in the center, are seldom economical to dig, as scrapers are generally used and a shallow s necessary. Deep narrow ditches require shoveling.

per proportions are largely a matter of judgment, depending on character of soil and of excavation. Average U S practice is to make $D = 0.5\sqrt{A}$, where D = depth, d A = area, sq ft. This rule gives following depths:

Area, sq ft.....	5	10	25	50	75	100	200	400
Depth, ft.....	1.1	1.5	2.5	3.4	4.3	5.0	7.1	10

e values are exceeded for special conditions, as on steep hill sides where a narrow, section (with lining for stability if necessary) is often economical in excavation. e water carries much silt the section may be shallower. Usual SIDE SLOPES are 1 : 1, 1, and 2 : 1. On steep hillsides in firm soil a 1 : 1 slope both for ditch section and side slope of bank is used. Any section gradually becomes rounded, the lower parts e slopes becoming flatter, the upper parts steeper. For aver loam or gravelly loam,

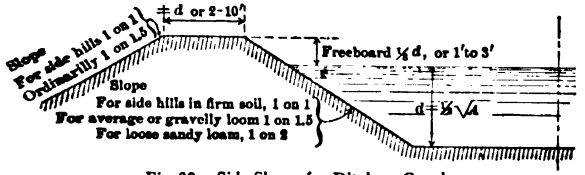

Slope
For side hills 1 on 1
Ordinarily 1 on 1.5
± d or 2-10'
Freeboard ½ d, or 1' to 3'
Slope
For side hills in firm soil, 1 on 1
For average or gravelly loom 1 on 1.5
For loose sandy loam, 1 on 2
d = ½√A

Fig 30. Side Slopes for Ditch or Canal

pe 1.5 : 1 is used; for loose sandy loam, 2.1. About 1 ft freeboard is common for , and a max of 3 ft for large ditches. Safe rule: make the freeboard ½ the depth of r above the max water level. TOP WIDTH of ditch bank may be fixed by what is re- d for its use as a road (10 to 12 ft); it ranges from about 2 ft for small, to a max of for large ditches, and is usually about equal to depth of water (Fig 30).

elocity of water must not be high enough to erode the bottom, thus deepen- he cut and making it difficult to divert water. But, there should be a cer- minimum, to prevent deposit of silt and excessive growth of water plants. rience in U S is that a velocity over 2 ft per sec will prevent deposit, and the wing max mean velocities are safe against erosion. (Note. — Bottom city is generally about 75% of the mean.)

	ft per sec
light loose sand..	1.0 to 1.5
age sandy soil....	2.0 to 2.5
age loam or allu- d soil.....	2.75 to 3.0
clay or ordinary vel.....	4.0 to 5.0
gravel or cobbles	5.0 to 6.0
lomerate, cement- gravel, soft rock.	6.0 to 8.0
rock.....10.0	to 15.0
rete, water carry- coarse sand....	7.0 to 12.0
rete, water carry- fine sand.....15.0	to 20.0

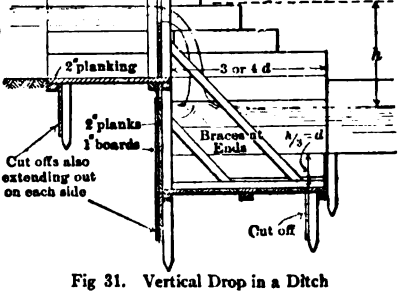


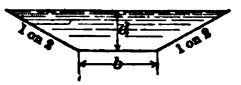
Fig 31. Vertical Drop in a Ditch

e above figures show that for es in earth the usual velocity o 3 ft per sec, which generally requires a slope of 4 to 7 ft per mile. A section ned as in the example following, and located to have the required slope; or, if

is fixed and small, it may be necessary to line the ditch to obtain the required discharge. the other hand, if a large slope is available, the ditch may be excavated at the max desi slope, and the extra head used up by constructing CHUTES or DROPS, commonly designe water-cushion or baffle spillways (Fig 31). In some cases where no supporting groun available, the ditch must be constructed on the given slope and lined to prevent erosi

Chezy formula, with C determined from Kutter's formula, is generally for ditches and flumes. Most engineers take $n = 0.025$ for aver earth actually ranges from 0.02 for firm soil, trimmed smooth with a shovel, to for rough gravelly surfaces, the aver for most irrigating ditches in Western being 0.0225.

Example. Design a ditch in earth to carry 2 500 miner's inches of water, and comp the required slope. Assume $n = 0.025$ and safe velocity = 2.5 ft per sec. Discharg



$A = bd + 2d^2$ $R = A \div (b + 4.48d)$

Fig 32. Cross-section of Ditch

2 500 ÷ 40 = 62.5 cu ft per sec, and required area = + 2.5 = 25 sq ft. Desirable depth = $0.5\sqrt{A} = 2$ With 1 : 2 side slopes, area will be $bd + 2d^2$ (Fig 32). $b = (25 - 12.5) \div 2.5 = 5$ ft. Hydraulic radius $R = A$ $(b + 4.48d) = 25 \div 16.2 = 1.54$ ft. Using the diagr (Art 11, Fig 16), the required slope is found to be 1. per 1 000 ft.

The center line of the ditch should be so located t the cut will make the required fill plus the shrinkage, except where it is advisable to b the water section entirely in cut because of danger from breaks. Breaks are especia be feared on steep hillsides, even when the sod under the fill is furrowed with a plow to g a good bond between the fill and original ground surface.

For level cuttings with 2 banks the economic cut x is computed to balance the sect (Fig 33a). For side-hill work, 1 bank only is generally required, and the "pivot-point method of locating the center line is used (Fig 33b). The section *ABC-DEF* being designed, a horiz line *LP* is drawn through the section, so that cut *LBCN* makes the fill *NDEP* plus shrinkage (usually 10%). Line *LP* is then divided into 2 parts by point *M*, equal if no allowance is made for shrinkage, or for 10% shrinkage such that $LM : MP :: \sqrt{100} : \sqrt{90}$. *M* is the pivot point, and for any slope as *AF* drawn through *N* the cut will balance the fill. *M* must thus be located on the ground as a p on a grade contour x ft above the bottom of the ditch, and the center line will be a di tance y up hill from *M*. x and y are independent of the slope of the side hill, and rem the same for any given cross-section.

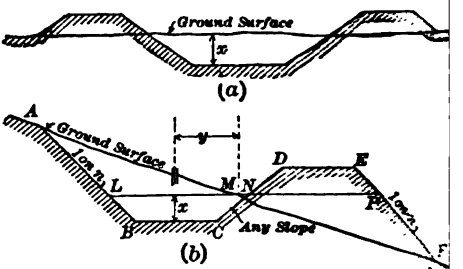


Fig 33. Balancing Cut and Fill

16. Losses in Ditches and Canals, Ditch Structures, Flumes

Seepage losses often amount to 25% or more of the water delivered at upper end of the ditch, and must be allowed for in computing the carry capacity. The loss depends on the character of the material in which the d is excavated, whether it is or is not lined, on the relation of the water level ditch to the water level of the adjacent ground, and on the age of the ditch the ditch is below groundwater level, there will be inflow from the adjoining la and vice versa. In many cases the ditch is above groundwater influence. and the seepage loss may be computed from the following table (Etcheverry):

Material	Loss, per 24 hr, per sq ft wetted perimeter	Material	Loss, per 24 hr, per sq ft wetted perimeter
pervious clay loam	0.25–0.35	Sandy loam	1.00–1.50
dinary clay loam, silt, or lava ash loam	0.50–0.75	Loose sandy soils	1.50–1.75
avelly clay loam, sandy clay oam or cemented gravel	0.75–1.00	Gravelly sandy soils	2.00–2.50
		Porous gravelly soils	2.50–3.00
		Very gravelly soils	3.00–6.00

itch linings are to prevent excessive seepage losses, or to permit higher rities than are permissible in earthy soils. Higher velocity is advisable in cases for economy, the cross-sec and seepage loss being smaller; and in cases where topography is such that supporting ground is not available combination of earth ditch and vertical drops or chutes to use up excessive

tails. About 3 gal heated heavy asphalt road oil per sq yd, applied in 2 or 3 doses and raked into the upper 2 or 3 in of the soil, will reduce seepage loss to about of value for untreated. It also prevents growth of vegetation and allows increase in ity to about 5 ft per sec, but is not permanent, requiring renewal every 2 to 4 years. PUDDLE, consisting of a 3 to 4-in layer of well-tamped clay, is as effective in stopping ge as oiling, but does not allow any higher velocity or prevent growth of vegetation. planking, nailed to sills set in sides and bottom of ditch is effective, but due to high and short life is seldom as economical as cement or concrete. CEMENT MORTAR 1 cement to 4 sand, 1 in thick, will prevent 75% of the seepage. CONCRETE lining, 4, 2 to 4 in thick, will stop 95% of the loss. The concrete is placed with or without On slopes of 1 to 1 or less, no forms are used, the lining being placed like cement alks, grooved every 8 or 12 ft to reduce width of contraction joints by making them at frequent intervals. For steeper side slopes heavier side linings are required and are used. Instead of trimming the bank to correct slope, it is best to use an out- form, filling in with wetted earth up to this form, and using an inner form for the ete. A solid foundation is important, as the concrete has little strength and will if foundation settles or pockets are washed out under it. Also, proper drainage is tial to keep material compact and dry, and to prevent heaving due to frost. Cost s from 3 or 4¢ per sq ft for cement linings 1 or 1.5 in thick, to 18 to 25¢ for 3 or 4-in ete linings.

tch structures. TURNOUTS are to distribute water from main ditch to laterals. For ditches, a wooden box will answer; for larger, where repairs would be costly, a anent vitrified or cement culvert is desirable (Fig Connections of siphons, rts, flumes and turnouts, n earth ditch must be ully made, to prevent ge at these points, with er of a washout. CHECKS hold flow-back, so as to ly a turnout with proper nt of water. They are times also used as small s (Fig 31) to decrease rade. CHUTES are long, sections on a steep grade, a water cushion at the bottom to kill high velocity. WASTEWAYS are overflow to carry off flood flows, or turnouts to a stream channel which act as blow-offs to y the ditch. They are installed on many long lines.

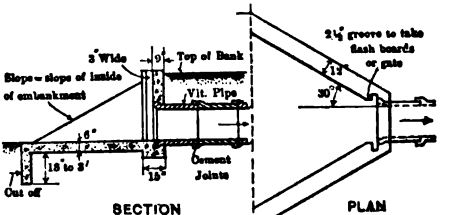


Fig 34. Turnout from Ditch

umes (Fig 35) are used to cross valleys, and on hillsides where the slopes oo steep or the soil is unsuited to an open ditch. The latter are called bench es. **Breadth** of flume box is generally twice the water depth (Art 15).

Bench flumes are often narrower, and trestle flumes wider. Freeboard (in) m be made equal to (depth of water in ft + 12) + 2. Flume lining should be o in, or better 2-in plank.

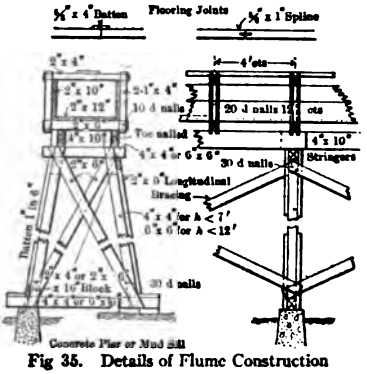


Fig 35. Details of Flume Construction

There are 2 forms of flume bot sills are placed across the stringers, flooring is laid parallel to direct flow like the sides; (*b*) flooring is tr verse and nailed to stringers. The is best and easiest to keep tight repair. Foundations and sills ar 2 × 12 or 3 × 12-in plank or con posts. Bents are spaced 12 to 15 ft ters, and stringers are designed to the load at a low unit stress. Sem cular continuous wood-stave pipe also been used for flumes; now replaced by semi-circular steel flu Several patented forms, made in R ver and San Francisco, are moderat cost, easily and quickly constructed watertight.

HYDRAULIC MEASUREMENTS

17. Measurement of Water Level

Non-recording instruments. Simplest is a graduated GAGE BOARD, or piece of sa level rod, fastened to side of tank or reservoir, and read where water level cuts the It can be read to 0.005 to 0.01 ft. If the water surface is rough, a STILL WELL (a box at the top and with a submerged opening in side) may be used and the gage placed i Fig 36 shows a variation from the ordinary form of gage board, the sloping lines allowi

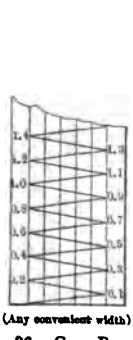


Fig 36. Gage Board

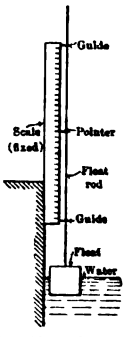


Fig 37. Float Gage

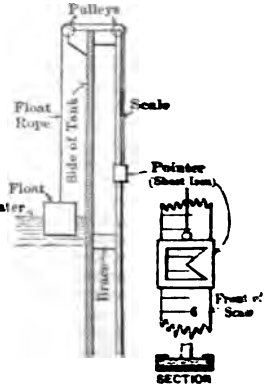


Fig 38. Float Gage for Tanks

more accurate estimate of water level. This design may be scratched with a nail side of a concrete tank before the concrete is fully set, or made by saw-cuts ⅛ in deep board. Fig 37 shows a more accurate instrument, a FLOAT GAGE consisting of a fixed and a pointer attached to a vertical rod on a float. As the reading is made on the this gage is convenient where the water surface is hard to reach. If a leveling rod

e scale, and a target vernier attached to the float rod, readings may be made to 0.001 02 ft. Fig 38 shows a float gage for a tank. By the HOOK GAGE a skilled observer can detect variations in water level of 0.0002 or 0.0003 ft. But such accuracy is seldom needed except in laboratory experimental work.

Recording gages. Fig 39 shows a simple form, which can be made by a good mechanic. The clockwork consists of an 8-day clock, with gearing to produce a revolution of the drum in any desired time. For recording weir flow, the vertical scale on the drum may be graduated to read discharge directly (Fig 13). Similar instruments, costing $75 to $150, are made by Hydro M'f'g Co, Phila, Pa; J. Friez, Baltimore, Md, and others. Fig 40 shows another recording gage. A drum *a*, containing a

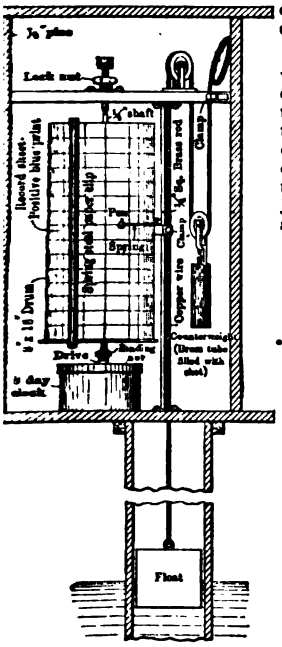


FIG 39. Recording Gage

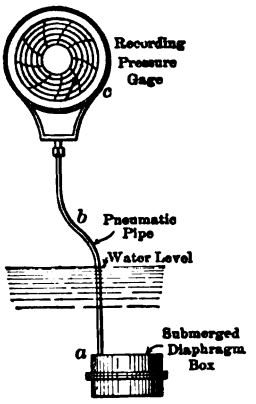


FIG 40. Bristol Recording Gage

diaphragm, is submerged in the water. Variation in water level causes variation in e of air in the drum, which is communicated by tube *b* to the recording gage *c*. ge is not interfered with by floating matter in the water, nor by ice, and is said r variations in level of ⅛ in (Bristol Co, Waterbury, Conn).

18. Measurement of Head or Pressure

lerate water press in tanks or pipes is generally measured ss tube or PIEZOMETER; higher press, by hydraulic gage, to Bourdon steam gage.

ead of a few ft a WATER TUBE is convenient, consisting of a tube attached to an opening in the pipe or tank, and having tube at its upper end. The tube is raised or lowered until the vel shows in the glass tube, and the distance of this level he datum (as center of pipe or bottom of tank) gives the head. ning in the pipe should be on a straight length of pipe, at right o its axis, and should have a sharp inner edge. The MERCURY TER (Fig 41) conveniently measures heads as high as 40 to ercury having a sp gr of 13.6. The distance *z* between the levels in the U tube, as read on the scale, is roughly 1/13 the actual head. True uals ($z \times$ sp gr of mercury) $+ z$.

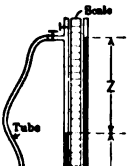


Fig 41. Mercury Piezometer

Hydraulic gages measure pressures up to 1 000 lb per sq in or 2 300 ft. **measuring DIFFERENCES IN PRESSURE** (if the press is very high) 2 gages are u

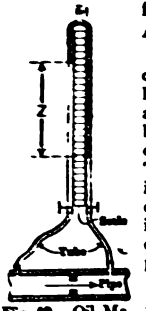


Fig 42. Oil Manometer

for very small differences, an **OIL MANOMETER** (Fig 42) (*T* A S C E, Vol 47, p 72) is used.

Kerosene oil is good. Its sp gr (which must be determined) is a 0.79. Hence the scale reading will be greater than the differen head, which is independent of height of o point of the scale above da and equals $z \times (1 - \text{sp gr of the oil})$. There may be trouble in obtaining good readings at the junction of water and oil, due to oil adhering to sides of tube. The **WATER OR AIR MANOMETER**, made by tube similar to that of Fig 42, can be used for small differences in head. Air must be pumped into the tube if the press is high. The reading is the true difference in head. Two open tubes could be used if total press is small.

Mercury manometer (Fig 43) is probably the best device for work of this kind. The true difference in head $= z \times (\text{sp gr of mercury} - 1)$.

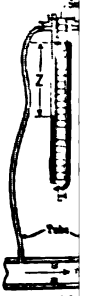


Fig 43. Me Manomet

In all these devices the length of tubes and the quantities of liquid used must be varied to suit the difference in head to be measured.

19. Measurement of Velocity and Discharge

For measuring small quantities it is best to allow the discharge for a given time to collect in a tank, where it may be measured or weighed. When this can not be done, or a permanent device is required, one of the following me must be used. Sometimes drop in hydraulic gradient can be measured, and discharge puted as in Art 11, but such results are liable to error due to uncertainty in value of the

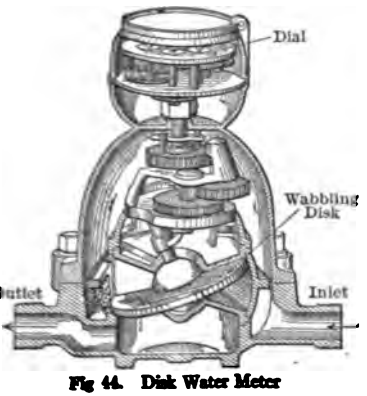


Fig 44. Disk Water Meter

Measurements in pres pipes. There are 3 method measuring water supply in o 2-in pipes, **DISK OR RING W METERS** are to be had in patterns.

The meter contains a wabbl or ring, so arranged that its mo communicated to a pin which works a train of clock wheels, the tity passing being registered on (Fig 44). This meter is "inferr and must be rated to determin discharge. Piston and other for meter are also used. All give ably accurate registration for ve flows (within say 10% for 10 g hr), but their rating varies wit and condition of meter and qual water.

Venturi meter is standard for permanent installations on pipes 3 or 4-i and over. They have been placed on the Catskill aqueduct for New Y water supply, pipe diam being about 14 ft. The meter consists simp gradual reduction in size of pipe to a throat ½ to ⅓ that of the pipe diam gradual expansion back to normal size (Fig 45). There is no obstruction flow of the water, and no moving parts.

Formula for discharge is based on the proposition that $h_1 + v_1^2 + 2g = h_2 + 2g$, and that $q = a_1v_1 = a_2v_2$. This gives $q = C \frac{a_1a_2}{\sqrt{a_1^2 - a_2^2}} \sqrt{2g(h_1 - h_2)}$, wh

a_2 = areas and h_1 and h_2 = pressure heads at the pipe and throat; g = acceleration
avity and C is a coefficient to allow for friction losses (usually taken at 0.99). A
meter is generally used to
ure $(h_1 - h_2)$, and for any
the formula reduces to $q =$
where z is the difference in
meter readings. (This meter
le by Builders Iron Foundry,
dence, R I, who also make
ling devices for recording
quantity and rate of flow.)

rough estimates of flow in
, saw dust may be intro-
at upper end and time
for passage through pipe.

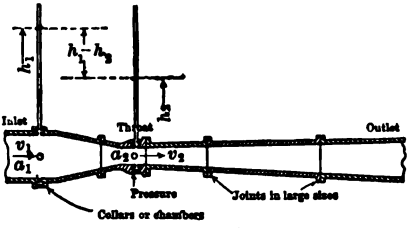


Fig 45. Venturi Meter

ot tube (1750) has been used in recent years. Fig 46 shows its principle.
urrent of water acting on the end of the curved tube causes the head in tube A to be greater than that in tube B. B measures pressure head only, and considering tube A to be a simple velocity-head proposition and the difference in levels equal to h, then $v = m\sqrt{2gh}$, where m theoretically = 1. If the action of the current against the tube is considered as impact, then $m = 0.7$. It has been shown also that, when placed so as to be subject to 2 currents of different velocities, the head is not that due to their mean, but to the mean of their squares. For high velocities, as in nozzles, m may be taken as 1 with quite accurate results. For pipes, in which velocities are low and vary as 48, the value of m is less, and for accuracy must be lined by ratings in a pipe of same diam.

Pitot Diagram)

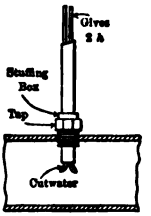


Fig 47. Pitot Tube

instrument is made by the Pitometer Co, N Y, who also make ling device. The usual form of Pitot tube (Fig 47) can be inserted in a hole tapped in a pipe, and a series of readings at different points in the diameter taken, from which the PIPE COEFF (Fig 48) is obtained. Knowing this, future measurements are made only at the center, center velocity being reduced to the mean by using the coeff. The makers advise using m =0.84.

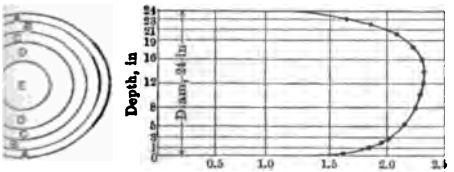


Fig 48. Diagram of Pipe Velocities

Determinations of Pipe Coeff from Fig 48

Ring	Area	Aver veloc	Discharge
A	0.502	1.57	0.788
B	0.873	1.87	1.632
C	0.698	2.05	1.431
D	0.720	2.18	1.570
E	0.349	2.24	0.781
Totals	3.142		6.202

an velocity $= \frac{6.202}{3.142}$
$= 1.97$ ft per sec.

e coeff $= \frac{\text{mean veloc}}{\text{center veloc}}$
$= \frac{1.97}{2.25} = 87.5\%$.

Measuring flow in open channels. The most accurate device is the (Art 10). Reliable data for weirs are available for a discharge up to about cu ft per sec. For greater discharge value, which is usually higher than limit of economy for the construction weirs, or where weirs are not warran or possible, floats and the current m are used. For floats a straight, uni section of the stream, at least 100 ft and on an even grade, should be sele Soundings are then taken, the stream divided into "lanes" like those in Fig If surface floats are used (small chip wood), a large number are allowed to down each "lane," and the time not the surface velocity of each course.

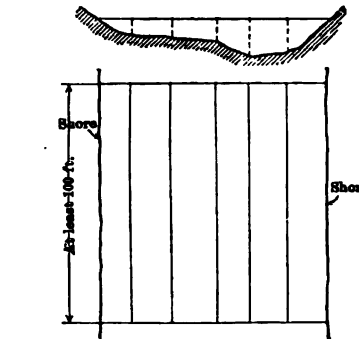


Fig 49. Soundings and "Lanes" for Measuring a Stream

Fig 50 shows variation in velocity in a tical section. To obtain the mean, the velocity must be multiplied by 0.79 to varying with roughness of bed and aver 0.86. Total discharge is found by multip the cross-sectional aver area of each lane by the mean velocity, and adding the re all the lanes. Wind affects surface floats, and shallow areas of dead water on the must be eliminated. A rough estimate of discharge can be made, with an uncertainty usually less than 20%, by timing floats down the center of a channel, thus obtaining the max surface velocity. Mean velocity in the entire cross-section is usually between 0.70 and 0.85 of this, the aver being 0.8.

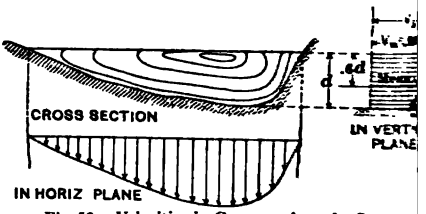


Fig 50. Velocities in Cross-section of a Stream

Submerged floats (Fig 51) are used particularly for deeper streams, and in studying tidal and other currents. By submerging the weighted float to a depth of about 0.6 (0.58 to 0.71) that of the vertical section which it floats, the mean velocity in the vertical is obtained.

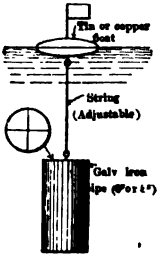


Fig 51. Submerged Float

Rod floats are hollow cylinders of tin, weighted by pebbles so as to stand vertically, and reaching nearly to the bottom channel. Francis gives $V_m = V_s (1.012 - 0.116 \sqrt{d'} + \ldots$ V_m = mean veloc in the vertical down which the rod floats, veloc V_s, d = total depth of stream, and d' = depth below S of rod. d' should not exceed 0.25 d. In practice it is difficult rod floats, as they require a very uniform channel, not obtainable except in canals and flumes.

Current meter is successfully used for measuring velocity and discharge in open channels and large pipes. It has four or more vanes mounted on a spindle, so arranged as to present always normal to direction of current, the pressure of which causes the vanes to revolve. The number of revolutions in a given time is recorded by an electric or mechanical counter, and is approx proportional to the current velocity.

The meter is first rated by moving it through still water at known velocities and noting the revolutions. If it is then submerged, by suspending it from a rod or rope, the re-

i can be counted and the velocity obtained from the rating chart. Usually the meter ld at a depth of 0.6 that of the water for obtaining mean velocity. It has been largely by the U S Geol Surv, for obtaining curves showing the discharge of a stream at rent elevations (stages) of water surface. Observers then note daily the water level, from the curves the flow is found. Two types of meter are in common use, the Price Fteley: the first for deep, rapid streams, the latter for shallow, slow streams (See Bib, "Use and Care of the Current Meter," *Trans* Am Soc C E, Vol 16, p 68).

liner's inch, as used in western U S, is the quantity of water which will in 1 min from a standard vertical orifice, 1 in sq, under a head at its center .5 in. This would equal 1.53 cu ft, but actual value varies in different dis-

s. In Cal and Mont, established by law 40 miner's inches = 1 per sec; in Col, 38.4, in Ariz, Idaho, Nev, Utah, 50 is accepted ommon agreement.

; 52 shows Foote's weir one of the cheaper de- for diverting a certain er of miner's inches from al in which flow varies. a long weir a consider- ncrease in flow in canal essary before the head ir is much increased. devices, with float and l valves, are also used intain constant head.

dule is an orifice used ing water, which under int head is to supply ain number of miner's

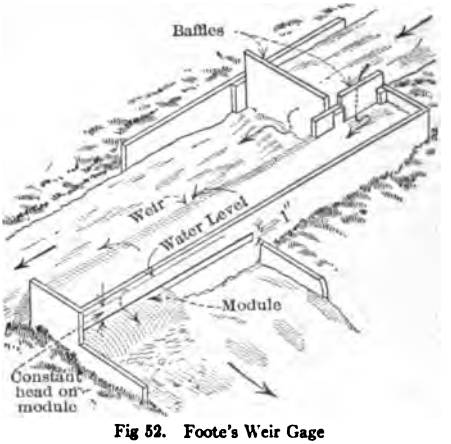


Fig 52. Foote's Weir Gage

. In Fig 52, the length of opening, in, = number of miner's inches, if head is con- As it is difficult to maintain constant head, this unit is going out of use.

WATER SUPPLY

20. Estimates for Water Supply

sumption is stated in gal per capita per day. While there are many data avail- spect ing requirements for small towns and cities, showing variation from 30 or 40 or more gal per capita aver daily consumption, it is only recently that the subject eived much attention in mining communities; the scanty information existing is unsatisfactory, as such towns have been compelled to use what has been available than the quantity desirable. Frequently the supply for mills and power plants principal requirement.

rce of supply. Having estimated the quantity required, with allowance for future e of population, a source of supply is sought and developed. This will be either round water, springs or streams. The quantity available from underground water, ed by wells or infiltration galleries, can seldom be estimated except by performance s driven to the same strata. To make sure that a spring or stream will give suffi- upply, measurements of its flow covering a number of years are necessary. In case ams, which usually have large variation in flow, a storage reservoir may be possible. e up the flood flows to supplement the natural flow during dry seasons, thus furnish- arger supply throughout the year. (Art 21.)

n pipe lines. In designing these, it is necessary to allow for variations in consu inless the water is delivered to a small distributing reservoir, having a capa

sufficient to equalize variations. The main pipe may then be designed to furnish
to the reservoir at the aver daily rate. For domestic consumption, the max daily ra
generally about 150% of the daily aver, and max hourly rate 250% of daily aver.

Distributing pipes. The size is usually determined by fire service requirements
the system is to be used for fire service without engines, a pressure of about 30 lb per s
(70 ft head) is required at the hydrants, and for an adequate stream 150 to 250 gal per
should be supplied. Generally it should be possible to concentrate 2 or 3 streams on
one point without using hose over 400 to 600 ft long; these requirements fix the locat
of outlets and allowable loss of head in the pipes, and hence determine their size.

21. Rainfall, Stream Flow, and Storage

Water falling as rain passes off as surface flow into streams and rivers,
evaporated directly by the sun from the land surface, is taken up by plants
vegetation, or sinks into the ground forming ground storage, some of which fi
its way underground to the streams and furnishes their main supply at all ti
except during and just after storms. Thus, stream flow will vary in both t
amount and distribution throughout the year on two water-sheds having
the same rainfall, but different slopes, amounts of vegetation, depth, and ch
acter of soil cover. Hence, for accurate estimates, data on the flow of the str
in question are necessary. Such data are sometimes obtainable from the publi
tions of U S Geol Surv, state, city, and private reports, but in many cases
entirely lacking. It may then be possible to secure data from a similar wa
shed, bearing in mind what points constitute similarity or, it may be neces
to make a rough estimate from measurements of the slope and cross-sectio
the stream, at its known low-water mark, applying the formulas of Art 11
hydrograph, like that in Fig 53, gives a complete record of stream flow, and
be used for all kinds of estimates in connection with it.

For example, since a dam spillway should be designed to pass safely the flood flow sh
at F, the stream records should be searched for the greatest flood, and an estimate m
from high-water marks of the str
The vertical scale in Fig 53 is cu f
sec per sq mile, which, multiplied
the area of the drainage basin give
total flow. Area of the basin ca
obtained by drawing a divide lin
a topographic map, enclosing the
within which all rain falling
not lost by evaporation and seep
would drain off to the stream at
tion. This area may then be
off with a planimeter.

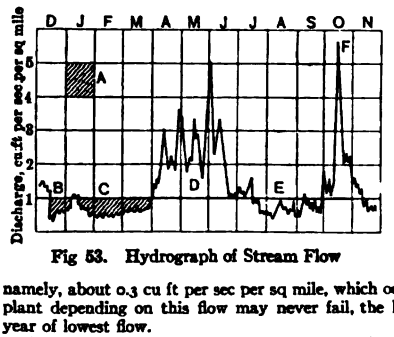


Fig 53. Hydrograph of Stream Flow

If no storage is available, the
flow to be counted on would be
minimum shown in the hydrogr
namely, about 0.3 cu ft per sec per sq mile, which occurs in December. In order that
plant depending on this flow may never fail, the hydrograph used must be that for
year of lowest flow.

If storage is available, the problem may be handled as follows. The area to be
hydrograph represents total flow for the year in cu ft, or the area A of 1 square inch
2 592 000 cu ft, which, divided by 43 560 = 5.95 acre ft, or a volume of 1 cu ft per
flowing for 1 month (30 days). For a flow of 1 cu ft per sec per sq mile, the re
storage will be the shaded areas B and C minus the small bit between them, where
hydrograph line rises above the 1-sec-ft line. This area can be taken off by planim
and changed to cu ft, giving the required storage. This storage is of course filled
the flow corresponding to D in the previous year, and will again be drawn down
although at E a smaller quantity is required than at B and C. If the storage is to
from a contour map of the reservoir site and assumed height of dam, and it is requir

he available flow, the problem is reversed and the line corresponding to the 1-sec- must be found by trial so as to enclose an area equal to the given storage.
: hydrograph shows that the distribution of flow throughout the year is very import- Streams fed by melting snows in mountainous regions show a much more even and r flow than in case of some southwestern rivers, which have little or no steady flow, re subject to violent and erratic floods caused by cloud bursts. In the former case, rsion dam may suffice for an adequate supply, or a small storage may be needed; in tter, a diversion dam must be built, and the flood flows, diverted to a reservoir or having a capacity sufficient to furnish the required supply for some months.
connection with storage reservoirs it should be noted that, as evaporation losses in egions are very large, it may be necessary to allow for 2 or 3 ft or more depth over ntire reservoir for this loss. Seepage losses also are sometimes very high, especially r the first 2 or 3 years following the completion of a reservoir. The total losses often nt to 25% of the influent, and under the best conditions are seldom less than 10%.

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SECTION 39

ENGINEERING THERMODYNAMI(

BY

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Note. — Numbers in parentheses in text (except numbers of equations) refer Bibliography at end of this section.

Introduction. The term thermodynamics has here been extended to include only part of the theory of the subject, but also formulas and numerical data on compression, steam, air and internal-combustion engines, fluid flow, combustion and related reactions. This section, therefore, is one of engineering, rather than of pu thermodynamics. In it reference is frequently made to Prof C. E. Lucke's work Engineering Thermodynamics (1). Methods of computing engine power, steam a fuel consumption, power to compress air and similar problems are given in Art leaving to Sec 40 the subjects of mechanical construction, sizes, costs and actual performance data of engines. Tables of physical and chemical constants frequent used in engineering computations are included in their respective articles.

1. Work and Power

Work in terms of pressure and volume. Work is expressed as the pro uct of force and distance; for the units of work see Sec 36, on Mechani Since pressure is force per unit area, following equations show that work is the product of press and vol:

$$PA = F, \quad \text{hence } W = PAL; \quad \text{but } AL = V, \quad \text{whence } W = PV,$$

ire P = press per unit area, A = area on which press acts, F = force acting, = work, L = distance, ft, and V = volume swept through while press is ng. The result is in ft-lb, if press is in lb per sq ft abs and vol is in cu ft.

iraphical representation of work. In Fig 1, the coordinates are press in er sq ft abs and vol in cu ft. Area denoting work done during any process nat under the curve, as AB, down to line of zero is; thus the work done during the volume change) B is the area $ABCD$.

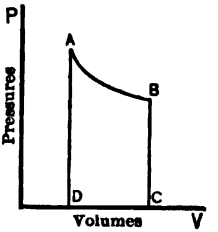


Fig 1. Graphical Representation of Work

Work of expansion or compression alone. When is changes in volume, work is done irrespective change in press; without change in volume, no k is done.

For constant press, $W = P(V_1 - V_2)$ (1)

" variable press, $W = P\,dV$ (2)

general expansion law for all gases is $PV^s = K$. n integrated, equation (2) assumes 2 forms, one n value of s in the general law is unity, the other n s has a value other than unity. Results of integration are given below veral convenient forms, in which P_1 is the greatest press and P_2 the least, eing the least vol and V_2 the greatest.

or special cases s has definite values for every gas. The most important case othermal expansion or compression, in which the temp is constant during expansion or compression, and $s = 1$ for all gases. Another important case is adiabatic compression or expansion, in which no heat is given to or removed from the gas during the process, and s equals specific heat at constant press divided by the specific heat at constant vol. Table 1 gives some adiabatic values of s, lying between 1 and 1.5.

$s = 1$	s not equal to 1
P_1V_1 Nap log $\frac{P_1}{P_2}$	$W = \frac{P_1V_1}{s-1}\left[1 - \left(\frac{P_2}{P_1}\right)^{\frac{s-1}{s}}\right]$ (3)
P_1V_1 Nap log $\frac{V_2}{V_1}$	$= \frac{P_1V_1}{s-1}\left[1 - \left(\frac{V_1}{V_2}\right)^{s-1}\right]$ (4)
P_2V_2 Nap log $\frac{P_1}{P_2}$	$= \frac{P_2V_2}{s-1}\left[\left(\frac{P_1}{P_2}\right)^{\frac{s-1}{s}} - 1\right]$ (5)
P_2V_2 Nap log $\frac{V_2}{V_1}$	$= \frac{P_2V_2}{s-1}\left[\left(\frac{V_2}{V_1}\right)^{s-1} - 1\right]$ (6)

pors may be assumed to behave as gases only when considerably superheated; when their saturation point, s is variable, and work determination by this method is in- . When steam expands so that it is just dry or follows the saturation law, $s =$ 5. For ordinary expansion of steam in a cylinder the average value of s is 1, but rpansion is not isothermal.

Table 1. Adiabatic Values of " s "

Substance	s	Authority	Substance	s	Authority
.....	1.4066	Smithsonian Tables	Methane....	1.316	Mäller
			Nitrogen....	1.41	Cazin
nonia, wet...	1.1	Average practice	Nitrous oxide	1.291	Wullner
nonia, super-ated.....	1.3		Pintsch gas..	1.24	Pintsch Co
			Sulph dioxide	1.26	Cazin
on dioxide...	1.3	Röntgen	Steam, super-heated....	1.3	Smithsonian Tables
on monoxide	1.403	Wullner			
on disulphide	1.2	Beyne	Steam, wet..	1.111	Rankine
rogen.....	1.41	Cazin	Steam, wet..	1+0.014 | ×%	Perry
sulphide	1.276	Mäller	Steam, wet..	1.035+0.01 | moist	Gray

press in lb per sq in, L is length of stroke, ft, a is the net area of piston, sq in and n the number of cycles per min. The term nL is the PISTON SPEED, and $LAn \div 33\,000$, the ENGINE CONSTANT. Horsepower (hp) results from multiplying the constant by (m e p). Values for the constant are given in Table for different size cylinders and piston speeds. Piston speed is best obtained the product of number of working chambers, length of stroke and rev per min divided by number of revolutions to complete one cycle.

Example 4. What is the I H P of a 9 by 12-in single-cylinder, double-acting engine running at 300 rev per min, when the (m e p) is 50 lb per sq in? Diam of cylinder always the first dimension given when expressing size of an engine. In this case there are 2 working chambers, and a cycle is completed in 1 revolution; hence piston speed 600 and constant is 1.16, giving 58 I H P.

Example 5. What is the hp of a 6-cylinder, single-acting, 4-stroke cycle gas engine 6 by 6 in, running at 1 000 rev per min, if the (m e p) is 80 lb per sq in?

Piston speed $= (6 \times 0.5 \times 1\,000) \div 2 = 1\,500$; engine constant $= 1.29$; I H P $= 129 \times$

Brake or shaft hp, so called because it is found by a brake, is that actually available at the shaft or belt pulley, and equals the indicated minus the friction hp. Ratio of brake to indicated is the mechanical efficiency of the engine. In a combined machine, as a direct-connected air compressor or pump, the mech effic is usually taken as the ratio of the I H P of the air or water end to that of the steam end. But in a pump, the output is sometimes considered as the hp equivalent of the lb of water pumped $\times$ head pumped against.

Thermal efficiency is the ratio of work done in the engine cylinder to the work equivalent of the heat required to do it.

Example 6. A pump delivers 1 000 gal of water per min against a net head of 150 lb per sq in. The I H P of water end is 100 and of steam end 120, and it uses 75 lb steam per min, each lb containing 1 200 heat units. What are the mech and thermal effics? Considering the mech effic to be the ratio of work in the cylinders, there results $100 \div 120 = 0.833$ or 83.3%. Considering the output to be the net work of the pump, which $(1\,000 \times 8.3 \times 150 \times 2.31) \div 33\,000 = 87$, there results $87 \div 120 = 0.725$ or 72.5%. Work done per min in the steam cylinder is $33\,000 \times 120$, and the work equivalent of the steam is $778 \times 75 \times 1\,200$; hence thermal effic is 0.056 or 5.6%.

2. Flow of Gases and Vapors

Flow through nozzles and orifices, with large pressure drop. When gases or vapors expand in nozzles the expansion occurs so rapidly that there is no time for escape or addition of heat (although actually a portion of the energy of the jet is turned back into heat by friction against the nozzle walls), hence the process is assumed to be adiabatic, and the preceding equations apply as well in this case, called free expansion, as when the expansion occurs in a cylinder, called balanced expansion. As the fluid must be brought to the nozzle at that which has expanded must be removed, the complete cycle is as in Fig 3, the area $ABFE$ representing the work done by the fluid approaching the nozzle, area $BCDF$ that of expansion, and $CDEG$ that required to remove the expanded fluid at P_b, hence negative. Algebraic sum of these areas is

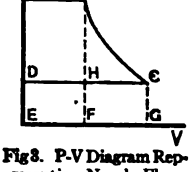


Fig 3. P-V Diagram Representing Nozzle Flow

$$W = \frac{s}{s-1} P_b V_b \left[1 - \left(\frac{P_c}{P_b} \right)^{\frac{s-1}{s}} \right]$$

If V_b represent the volume of 1 lb under the initial conditions, W is the work done by 1 lb, and if this work is used in accelerating the same 1 lb of fluid, then

$$W = (u_2^2 - u_1^2) \div 2g$$

nozzle work, w_1, the original velocity, is generally small as compared to w_2, final velocity, so that w_1^2 may be neglected; in which case, from Eq 9 and 10,

$$w_2 = 8.02 \sqrt{\frac{s}{s-1} P_b V_b \left[1 - \left(\frac{P_c}{P_b}\right)^{\frac{s-1}{s}}\right]} \qquad (11)$$

ight of fluid flowing through the nozzle follows directly, since it equals the duct of velocity, cross-section and wt per cu ft. Let A = area at point of imum velocity, then,

$$w = wA\delta_c = 8.02 A\delta_c \left\{\frac{s}{s-1} P_b V_b \left[1 - \left(\frac{P_c}{P_b}\right)^{\frac{s-1}{s}}\right]\right\}^{0.5}$$

re w = lb per sec, A = area in sq ft, and δ_c = wt of 1 cu ft at the low press. s generally more convenient to express δ_c in terms of original conditions since

$$\delta_c = \frac{1}{V_c} = \frac{1}{V_b}\left(\frac{P_c}{P_b}\right)^{\frac{1}{s}}$$

$$w = \frac{8.02 A}{V_b}\left(\frac{P_c}{P_b}\right)^{\frac{1}{s}} \left\{\frac{s}{s-1} P_b V_b \left[1 - \left(\frac{P_c}{P_b}\right)^{\frac{s-1}{s}}\right]\right\}^{0.5} \qquad (12)$$

ecomes a maximum when first differential = 0, which occurs when $\left(\frac{P_c}{P_b}\right)^{\frac{1-s}{s}}$ $= \frac{1}{2}$, or for the maximum flow,

$$P_c = P_b \left(\frac{2}{s+1}\right)^{\frac{s}{s-1}} \qquad (13)$$

ucing the lower press will not increase wt, but final velocity only. Throat of a nozzle is designed to pass the desired wt for the above press ratio, and mouth area to permit of the complete expansion. A relation between at and mouth area (by Moyer) is

$$\begin{aligned} \frac{\text{Mouth area}}{\text{Throat area}} &= 0.172 \frac{P_b}{P_c} + 0.7, \quad \text{if } \frac{P_b}{P_c} < 25 \\ &= 0.175 \left(\frac{P_b}{P_c}\right)^{0.94} + 0.7, \quad \text{if } \frac{P_b}{P_c} > 25 \end{aligned} \qquad (14)$$

line joining mouth and throat es greatly, but a common form is st a straight line and a 20° angle. he curve in Fig 4 shows the pressure $P_c \div P_b$ for max flow for differ- values of s. For all vapors, unless iderably superheated, s varies in e throughout the expansion, and e results for vapors by above od are not strictly accurate. For tact method see Art 10.

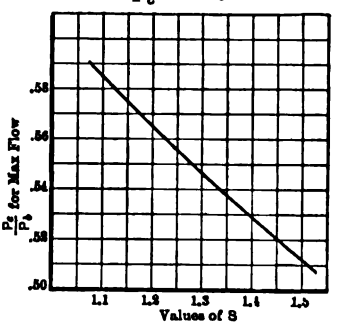


Fig 4. Pressure Ratio for Max Wt Discharged by Nozzle for any Value of "s"

he above equations are for perfect les or orifices, so that the actual lischarged, or the area to pass a n wt, may differ somewhat from calculated, due to stream contrac- For orifices with sharp corners effective area may be as low as 60% of the actual, while for nozzles it be as high as 95%.

For flow occurring when the low press is below the critical value the empir equations in Table 3 apply.

Table 3. Formulas for Flow of Steam and Air from Orifices

Lb dry steam per hr per sq in of orifice	$60\ p^{0.97}$	Grashof
	$51.43\ p$	Napier
	$49.6\ p$	Harter
	$[(10.57 - \log p)\ p] + 1\ 000$	Ratean
Lb superheated steam per hr per sq in of orifice	$60\ p^{0.97}$ [1 + 0.00065 × degrees of superheat]	Moyer
	$49.6\ p$ to $45\ p$ for superheat 0° to 185° at 160 lb	Harter
Lb of air per hr per sq in of orifice	$1\ 900\ p_1 + \sqrt{T_1}$ for flows from 2 atmos or more to 1 atmos	Fliegener
Lb of wet steam per hr per sq in of orifice	$60\ p^{0.97}$ + √dryness fraction	Grashof

Example 7. Calculate loss of air per hr from a tank in which the press is maintain at 90 lb per sq in gage and temp is 60° F, if there be a leak amounting to ⅛ in of effecti opening. Quantity of air escaping can be no greater than if the press without the ta were 53% of that within, so this value will be used for P_c in Eq 12. For conditi given, $V_b = 2$ cu ft; hence, by Eq 12, wt per hr =

$$3\ 600 \times \frac{8.02 \times 0.125}{144 \times 2} \times (0.53)^{0.71} \left\{ 3.5 \times 144 \times 105 \times 2\,[1 - (0.53)^{0.29}] \right\}^{0.5} = 1\ 07$$

By Fliegener's empiric equation the wt per hr = (1 900 × 105) + (520 × 8) = 1 99 Should the press within the tank be only 10 lb gage, the empiric formula would no long apply, since the low press is above the critical value, and the result must be found Eq 12. For above press, $V_b = 8.4$ cu ft, hence the wt per hr will be

$$3\ 600 \times \frac{8.02 \times 0.125}{144 \times 8.4} \times (0.6)^{0.71} \left\{ 3.5 \times 144 \times 25 \times 8.4\,[1 - (0.6)^{0.29}] \right\}^{0.5} = 253 \text{ lb.}$$

Flow through orifices with small press drops occurs in all engine a compressor valves, and in discharge from pipes. Use is made of this drop press for measurement purposes. For water and non-expansive fluids t velocity of flow is merely $\sqrt{2gh}$, but with expansive fluids the work done expanding must also be accounted for. In Fig 3, for a non-expansive fluid t work causing increase in velocity is represented by area *ABHD*, for expan fluids the area *ABCD*. Naturally, for small press differences the net work expansion *BCH* may be neglected. Hence, there are 2 sets of expressions f flow for small drops: (*a*) the exact, for expansive; (*b*) the exact, for non-e pansive and approximate for expansive.

$$\text{Lb per sec per sq ft of effective orifice} = \begin{cases} \sqrt{2g \dfrac{s}{s-1} \dfrac{P_1}{V_1} \left(\dfrac{P_2}{P_1}\right)^{\frac{2}{s}} - \left(\dfrac{P_2}{P_1}\right)^{\frac{s+1}{s}}} & \text{(a)} \\ \sqrt{2g \dfrac{P_1 - P_2}{V}} & \text{(b)} \end{cases} \quad (15)$$

where P_1 and P_2 are initial and final press, in lb per sq ft; V_1, the vol per lb a initial press; and V, the vol per lb at aver press, $(P_1 + P_2) \div 2$. The effect opening is always less than the actual; Table 4 gives values for the ratio effective to actual.

For air Eq (16) is useful, and Table 5 gives values for C found by Durley [6]. The values apply exactly for air flowing into a press of 30 in of mercury, between 40° and F, when the orifice is less than 1/10th the area of the approaching pipe or chamber, h sharp edges, and a thickness of 0.057 in. For a distance, they apply fairly well be

ble 4. Values of Ratio of Effective to Actual Openings in Orifices (Weisbach)

$\div P_2$	1.05	1.09	1.43	1.65	1.89	2.15
for orifice, diam = 0.39 in	0.555	0.589	0.692	0.724	0.754	0.788
$\div P_2$	1.05	1.09	1.36	1.67	2.01	.....
for orifice, diam = 0.84 in	0.558	0.573	0.634	0.678	0.723	.....
$\div P_2$	1.05	1.1	1.3	.....	.....	.....
for short tube, $d = 0.39$, $l = 1.18$	0.730	0.771	0.830	.....	.....	.....
$\div P_2$	1.41	1.69	.....	.....	.....	.....
for short tube, $d = 0.56$, $l = 1.67$	0.813	0.822	.....	.....	.....	.....

above conditions, the character of the orifice edge being the variable having the most ience; values of C as high as 0.94 having been found for orifices with rounded approach.

) air per sec to atmosphere $= 0.6283\ Cd^2\sqrt{h_w + T}$, through a circle d in in diam; (16) re h_w = press drop, in of water, and T = abs temp.

de 5. Discharge Coeffi C for Different Heads and Diam of Orifice (Durley)

Diam of rifice, in	Drop in press, in of water				
	1	2	3	4	5
5/16	0.603	0.606	0.610	0.613	0.616
1/2	0.602	0.605	0.608	0.610	0.613
1	0.601	0.603	0.605	0.606	0.607
1 1/2	0.601	0.601	0.602	0.603	0.603
2	0.600	0.600	0.600	0.600	0.600
2 1/2	0.599	0.599	0.599	0.598	0.598
3	0.599	0.598	0.597	0.596	0.596
3 1/2	0.599	0.597	0.596	0.595	0.594
4	0.598	0.597	0.596	0.594	0.593
4 1/2	0.598	0.596	0.596	0.593	0.592

general, the constants for different s of orifice may be taken between limits shown in Table 6, no rule for ific cases being possible.

f the gas being measured is other a air the wt of flow is found from 17:

$$w_G = w_A\sqrt{V_A \div V_G} \qquad (17)$$

re w_G and w_A are the wt of gas and respectively, and V_G and V_A their per lb.

Table 6. General Limits of Constants (Weisbach)

Type of orifice	Constant
Rounded entrance	0.97–0.99
Sharp edge thin plate	0.55–0.80
Short straight tubes	0.80–0.85
" rounded entrance tube	0.92–0.93
Converging orifices	0.90–0.99

enturi tubes are used commercially for measuring gases and vapors, and e again 2 expressions result:

$$\text{Lb per sec} = A_2\left(\frac{P_2}{P_1}\right)^{\frac{1}{\delta}}\sqrt{2g\frac{s}{s-1}P_1\delta_1\frac{1-\left(\frac{P_2}{P_1}\right)^{\frac{\delta-1}{\delta}}}{1-\left(\frac{A_2}{A_1}\right)^2\left(\frac{P_2}{P_1}\right)^{\frac{2}{\delta}}}} \qquad (a)$$

$$= \frac{A_1}{\sqrt{A_1^2 - A_2^2}}\sqrt{2g(P_1 - P_2)\delta} \qquad (b) \qquad (18)$$

re A = area in sq ft, P = press in lb per sq ft, δ = wt per cu ft of gas, and ratio of specific heats, $C_p \div C_v$, of the gas. Subscript 1 refers to upstream,

and 2 to throat conditions, no subscript referring to the mean value. F steam the exact expression is simplified, since the factor

$$\frac{s}{s-1}\frac{P_1}{\delta_1}\left[1-\left(\frac{P_2}{P_1}\right)^{\frac{s-1}{s}}\right] = 778 \times \text{work of Rankine cycle in B t u}$$

between pressures P_2 and P_1, and may be found by the Mollier diagram, Fig 2 hence,

$$\text{Lb steam per sec} = A_2\left(\frac{P_2}{P_1}\right)^{\frac{1}{s}}\delta_1\sqrt{\frac{2g \times 778\ (\text{work of Rankine cycle})}{1-\left(\frac{A_2}{A_1}\right)^2\left(\frac{P_2}{P_1}\right)^{\frac{2}{s}}}} \quad (1$$

Pitot tube is also used for measuring gases. As it is primarily a velocity meter sets of values are possible for its results, one neglecting the work of expansion, the oth accounting for it. But, to secure accurate readings the velocities must be high, so t the use of the exact expressions is more important than in the orifice or Venturi met As the readings are often in in of water or mercury, as well as in lb, the equations made to apply to these units also.

$$\left.\begin{aligned}\text{Lb per sec} &= A\sqrt{Z}\sqrt{2g\delta_1\left[1-\frac{K}{2s}\,\frac{1+s}{6s^2}K^2\right]-\left[\frac{(1+s)(1+2s)}{24s^3}\right]K^3} \quad (a)\\ &= A\sqrt{Z}\sqrt{2g\delta_1}\end{aligned}\right\}$$

$$\text{where } Z = P_2 - P_1 = 5.2\,(hw_2 - hw_1) = 70.8\,(hm_2 - hm_1) \qquad (b)$$

$K = \left(\frac{P_2}{P_1}-1\right)$, s = ratio of specific heats, $C_p \div C_v$, δ = lb per cu ft of fl

$(P_2 - P_1)$ = diff in press in lb per sq ft, $(hw_2 - hw_1)$ in in of water, and $(hm_2 - hm_1)$ i of mercury, subscript 1 referring to the tangential orifice and 2 to the impact orifice

Flow in pipes. For flow of gases and vapors through pipes there is lit more reliable information than empiric formulas, some of the commoner be given below (Gebhardt). For theory of pipe flow, see (1); for pipe sizes u in practice, see (5).

$$\text{Veloc, ft per sec} = A\sqrt{\frac{(p_1-p_2)d}{\delta L}}, \text{ where } A = \begin{cases}9\,240 \text{ Geipel \& Kilgour}\\ 9\,976 \text{ Hawksley}\\ 10\,350 \text{ Martin}\\ 10\,360 \text{ Hurst}\end{cases}$$

$$\text{Flow, lb per min} = B\sqrt{\frac{(p_1-p_2)\delta d^5}{L}}, \text{ where } B = \begin{cases}50.2 \text{ Geipel \& Kilgour}\\ 59.4 \text{ Hawksley}\\ 56.5 \text{ Martin}\\ 56.5 \text{ Hurst}\end{cases}$$

$$\left.\begin{matrix}\text{Press drop, lb}\\ \text{per sq in}\end{matrix}\right\} = C\frac{w^2L}{\delta d^5}, \qquad \text{where } C = \begin{cases}0.000396 \text{ Geipel \& Kilgour}\\ 0.000337 \text{ Hawksley}\\ 0.000313 \text{ Martin}\\ 0.000313 \text{ Hurst}\end{cases}$$

$$\text{Pipe diam, in} = D\sqrt[5]{\frac{w^2L}{(p_1-p_2)\delta}}, \text{ where } D = \begin{cases}0.2087 \text{ Geipel \& Kilgour}\\ 0.2010 \text{ Hawksley}\\ 0.1990 \text{ Martin}\\ 0.1990 \text{ Hurst}\end{cases}$$

In above equations. d = diam, in; $(p_1 - p_2)$ = press drop, lb per sq in; w = flow per min; L = length of pipe, ft, and δ = lb per cu ft of fluid.

Following values by Hunt show loss in press due to 90° elbows, expressed in equivalent ft of straight pipe.

Pipe, diam	Resistance	Pipe, diam	Resistance	Pipe, diam	Resistance	Pipe, diam	Resistance
1	1.5	5	20	9	44.4	16	90.1
2	4.9	6	25.9	10	50.7	18	104
3	9.4	7	32	12	63.7	20	117
4	14.5	8	38	14	76.7	22	130
....	.....	.....	.....	.....	.....	24	144

Draft due to convection flow, as found in chimneys and in some ventilating systems, depends on temp and height of stack or duct. For STATIC conditions difference in press of draft in in of water (the common unit) is $H\left(\frac{7.64}{T_C}-\frac{7.95}{T_H}\right)$ chimney gases, or $7.64\,H\left(\frac{1}{T_C}-\frac{1}{T_H}\right)$ for air. Due to flow conditions, wall stances, bends, changes of cross-sec in duct work, many of which can not be luated, the actual draft will be less. For chimneys the same is true, besides ch there are fuel-bed, boiler-tube and economizer resistances, rendering any empiric formulas of little practical value. Stirling suggests a corrected ation for actual draft:

$$\text{Actual draft, in water} = 7.64\,H\left(\frac{1}{T_C}-\frac{1}{T_H}\right)-\frac{fw^2CH}{A^3} \qquad (25)$$

re H = height, ft, T_C = abs temp of air, T_H = mean abs temp of hot gases, wt of gases per sec, C = perimeter of stack, A = area of stack, and f = a tant varying from 0.001 to 0.002, 3 values for which are: 0.001 for steel k (abs temp = 800), 0.0015 for steel stack (abs temp = 1 060) and 0.002 brick stack (abs temp = 760).

ndency of heated liquids and gases to rise is often utilized in heating or cooling ms to produce flow, the thermosyphon consisting merely of 2 pipes or flues, con- d at top and bottom; through one pipe the warmer, lighter fluid rises, through the r the cooler and heavier descends.

apacity of chimneys does not increase indefinitely with stack temp, but aximum when the temp is approx twice the abs outside temp. Formulas numerous and are all empiric. Chimney capacities are generally stated in r horsepower (B H P). Common formulas are:

$$\left.\begin{aligned} \text{B H P} &= 3.25\,A\sqrt{H}\ \text{(Christie)}\\ \text{B H P} &= 3.33\,E\sqrt{H}\ \text{(Kent)}\\ D &= 4.92\ (\text{B H P})^{3/8}\ \text{(Stirling)}\end{aligned}\right\} \qquad (26)$$

he last formula can also be stated as B H P = $12.6\,A^{3/4}$.

these formulas, A = area of top of chimney, sq ft; H = ht of chimney, ft; effective area, after deducting 4 in from diam or width, to allow for chimney ion; D = diam of chimney, in. The first 2 formulas include a function of nd therefore require previous determination of an H that will produce the t required to overcome the combined resistance of the full-bed, boiler, and

It is claimed that Stirling formula gives the stack area that costs least to truct, and that this area does not change with H, determined independently.

the coal burned and flue gas produced per B H P are indefinite, the formulas are ated below, giving capacity in lb of flue gas per hr; using in first formula 4 lb coal B H P (Christie); in second, 5 lb coal per B H P (Kent); in both formulas, 24 lb as per lb coal, and in the last, the Stirling value of 120 lb flue gas per B H P.

$$\left.\begin{aligned} \text{Lb flue gas per hr} &= 312\,A\sqrt{H}\ \text{(Christie)}\\ \text{“ “ “ “ “} &= 400\,E\sqrt{H}\ \text{(Kent)}\\ \text{“ “ “ “ “} &= 1\,508\,A^{3/4}\ \text{(Stirling)}\end{aligned}\right\} \qquad (27)$$

3. Work and Capacity of Air Compressors (See also Sec 15)

Isothermal compression requires the least work, but is impossible to a in practice even with the most efficient jacketing.

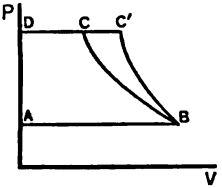


Fig 5. P-V Diagram for Ideal Single-stage Compression

Actual compressors operate with the compres line somewhere between isothermal and adiab Leakage of air from the cylinder during compre causes displacement of compression curve and giv lower value to exponent *s* (Art 1). Some compr diagrams show apparently isothermal compres whereas result is really due to leakage. Fig 5 sh an ideal diagram for the 2 cases in a no-clearan single-stage machine, area *ABCD* representing work for isothermal compression, *ABC'D* that adiabatic, and *CBC'* the saving due to the forma

Stage compression. Isothermal compression is more nearly approache the gas be compressed in one cylinder to some intermediate press, delivered t intercooler, and the compression completed in a second cylinder. This is ter staging, the number of stages being 2, 3, or 4, most commonly 2. Fig 6 shows the case for an ideal, no-clearance, two-stage machine. Area *ABCC'DE* represents the process as given, area *ABCD'E* that for single-stage adiabatic compression for same amount of air, and area *ABC'D''E* for single-stage isothermal compression. Saving due to staging is area *CC'DD'*.

The air is generally cooled to about the original temp in an intercooler, and in the following equations this assumption is made. The press to which the air is compressed, termed the RECEIVER PRESS, affects the saving due to staging. Maximum saving occurs in 2 stages when

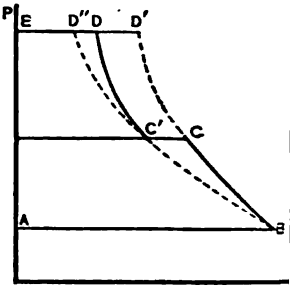


Fig 6. P-V Diagram for Ideal Tw stage Compression

$$\text{receiver press} = (\text{low press} \times \text{high press})^{.5}$$

and in 3 stages when

$$\text{first receiver press} = (\text{low press})^{.67} (\text{high press})^{.33}$$

$$\text{second receiver press} = (\text{low press})^{.33} (\text{high press})^{.67}$$

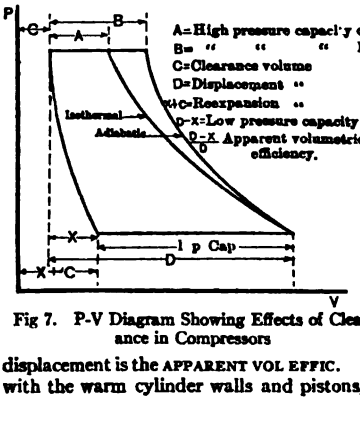


Fig 7. P-V Diagram Showing Effects of Clearance in Compressors

As real compressors have CLE ANCE vol, not all the air in th cylinder can be expelled on eac stroke, and the vol remaining m reëxpand to the low press bef any fresh air can be drawn i Hence, the quantity of air c pressed is always less than th piston displacement (Fig 7). Th ratio of the low-press capacity t displacement is the APPARENT VOL EFFIC. Since the entering air comes in contact with the warm cylinder walls and pistons, it expands, and hence the quantity

r drawn in, when measured under external conditions of press and temp, en less than would appear from the value of apparent vol effic. The ratio e actual vol drawn in to the cylinder displacement is the TRUE VOL EFFIC. o by Lucke is an attempt to formulate this quantity.

$$\text{rue vol effic} = \frac{\text{apparent vol effic}}{1 + (0.3 \text{ to } 0.5)\left(\frac{\text{abs temp air delivered}}{\text{abs temp air supplied}} - 1\right)} \qquad (30)$$

lelivered in other than isothermal compression is at a higher temp than supplied, and when cooled to original temp decreases in vol. Vol actually ered is termed the high-press capacity hot; when cooled, the high-press ity cold. These volumes for isothermal compression are equal.

pressions for work and capacity of single and 2-stage compressors are in Tables 7 and 8, in which following symbols are used:

low press, lb per sq in absolute
high press " " " "
receiver press, lb per sq in absolute
clearance, as a fraction of displacement
capacity, cu ft
diam of cylinder, in
stroke, in
cycles per min
displacement, cu ft per min

s = val of exponent of volume in equation for the compression curve $PV^s = K$, = 1.406 for adiabatic compression
T_1 = temp of entering air, abs
T_2 = temp of delivery air, abs
δ_1 = wt per cu ft of entering air
R_p = ratio, high press over low press
E_v = apparent vol effic
C_v = specific heat at constant volume
w = wt of air

scripts 1 and 2 refer to low and high-press cyl respectively.

ork required to compress a given quantity of air is endent of the clearance, but the size of the cyl- required varies with the clearance. A comparison e expressions for work for the entire single-stage ressor cycle and for compression alone (Eq 5) s that the work for the entire cycle is s times that ompression alone, that is, in Fig 8, area $ABCD$ imes area $BCEF$. Hence, Eq 7, if multiplied by es the complete work for adiabatic compression.

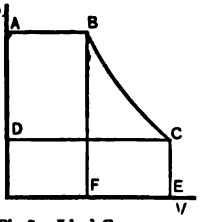


Fig 8. Ideal Compressor Card

ork of complete cycle for adiabatic compression
$$= JsC_v w\,(T_2 - T_1) \qquad (31)$$

se quantities of Tables 7 and 8 most frequently required may be found graphically Fig 9 to 11. Fig 9 is for finding the apparent vol effic for any clearance, press ratio, . Project from upper horiz scale to the proper curve for s, then horiz to the proper nce curve, and then down to lower horiz scale, where answer is read. By project-

om the s curve to left-hand scale the value of $R_p^{\frac{1}{2}}$ is found, which is useful in finde high-press cap hot, when the low-press cap or displacement is known, since

press cap hot = low-press cap $\div R_p^{\frac{1}{2}}$. Fig 10 is to find the work required to compress t, or the hp required to compress 1 000 cu ft of free air per min in one stage, between ressures and for any value of s. The upper of the 2 lower scales, when multiplied by ent vol effic, gives the m e p in lb per sq ft. To use, project downward from the value to the proper s curve, then horiz to the low-press curve, and finally downward to read nswer. Fig 11 is a diagram for 2-stage compression with best receiver press and ct intercooling. Results obtained by the expressions of Tables 7 and 8, or from the s, are for perfect compressors. The amount of saving by compressing in 2 stages single-stage adiabatic is seen in Fig 12. Here the horiz scale is the ratio of high to low, the vertical scale the ratio of the work required to compress a given amount in two stages to that required to compress an equal amount to the same pressure in

one stage. As the value of s effects the result, curves for various values of this are g on the figure. Knowing the press ratio, project from the value to the proper s cu and then horizontally the number so found when multiplied by the work required s stage will give that required for 2-stage.

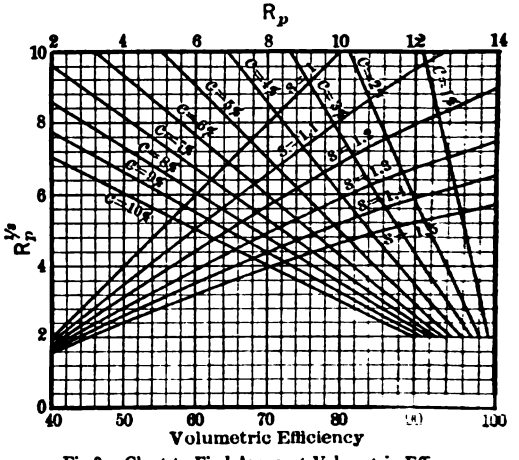


Fig 9. Chart to Find Apparent Volumetric Effic

Blowers are used for compressing large volumes of air or other gas to press below 1 lb. As the work to compress a given quantity of gas from one press to an is independent of the type of machine, all the above work formulas apply equa

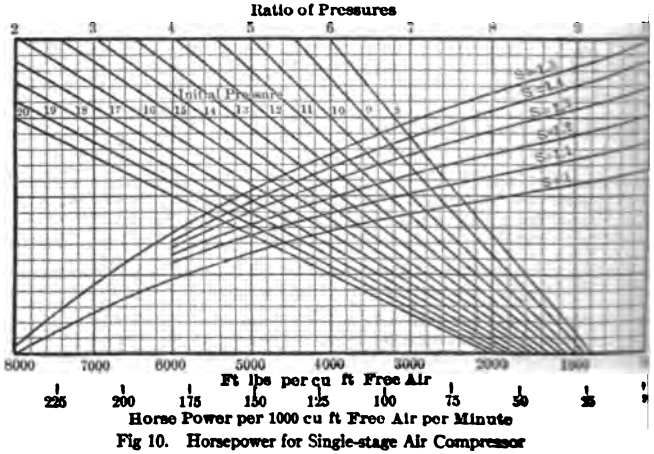


Fig 10. Horsepower for Single-stage Air Compressor

blowers. In case of fans the press to which the air is compressed is so little above of the atmosphere that the work of compression alone becomes negligible, so that work may be represented by the volume multiplied by the increase of press

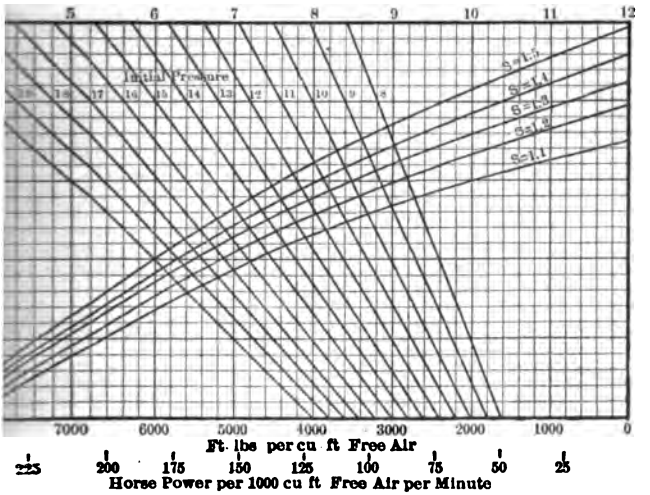
	Isothermal compression	Non-isothermal compression
Work per min	$144\,(l\ p)\,(l\text{-}p\ cap)\,\text{Nap log}\,R_p$	$144\,\frac{s}{s-1}\,(l\ p)\,(l\text{-}p\ cap)\left[(R_p)^{\frac{\theta-1}{\theta}}-1\right]=sc_vw\,(T_2-T_1)^*$
" " cu ft low-press gas	$144\,(l\ p)\,\text{Nap log}\,R_p$	$144\,\frac{s}{s-1}\,(l\ p)\left[(R_p)^{\frac{\theta-1}{\theta}}-1\right]=sc_v\delta_1\,(T_2-T_1)^*$
" " " high " " hot		$144\,\frac{s}{s-1}\,(l\ p)\left[(R_p)^{\frac{\theta-1}{\theta}}-1\right]R_p^{\frac{1}{\theta}}=sc_v\delta_1\,(T_2-T_1)\,R_p^{\frac{1}{\theta}*}$
" " " " " " cold	$144\,(h\ p)\,\text{Nap log}\,R_p$	$144\,\frac{s}{s-1}\,(h\ p)\left[(R_p)^{\frac{\theta-1}{\theta}}-1\right]=sc_v\delta_1\,(T_2-T_1)\,R_p^*$
M e p, lb per sq in	$(l\ p)\,E_v\,\text{Nap log}\,R_p=(l\ p)\,[1+c-cR_p]\,\text{Nap log}\,R_p$	$\frac{s}{s-1}\,(l\ p)E_v\left[(R_p)^{\frac{\theta-1}{\theta}}-1\right]=\frac{s}{s-1}\,(l\ p)\left[1+c-cR_p^{\frac{1}{\theta}}\right]\left[(R_p)^{\frac{\theta-1}{\theta}}-1\right]$
Displacement per min, cu ft	$\frac{d^2ln}{2\,200}$	$\frac{d^2ln}{2\,200}$
Low-press capacity per min	$DE_v=D\,[1+c-cR_p]$	$DE_v=D\left[1+c-cR_p^{\frac{1}{\theta}}\right]$
High " " hot per min		$\frac{DE_v}{R_p^{\frac{1}{\theta}}}=\frac{D\left[1+c-cR_p^{\frac{1}{\theta}}\right]}{R_p^{\frac{1}{\theta}}}=\frac{l\text{-}p\ cap}{R_p^{\frac{1}{\theta}}}$
" " " cold " "	$DE_v+R_p=D\,[1+c-cR_p]+R_p=(l\text{-}p\ cap)+R_p$	$\frac{DE_v}{R_p}=\frac{D\left[1+c-cR_p^{\frac{1}{\theta}}\right]}{R_p}=\frac{l\text{-}p\ cap}{R_p}$
Apparent vol effic	$1+c-cR_p=\frac{l\text{-}p\ cap}{D}$	$1+c-cR_p^{\frac{1}{\theta}}=\frac{l\text{-}p\ cap}{D}$

Table 8. Expression for Work and Capacity of Two-stage Compressors with Perfect Intercooling

	Any receiver pressure	Best receiver pressure
Work per min.....	$144\frac{s}{s-1}(l\,p)(l\text{-}p\ cap)\left[\left(\frac{r\,p}{l\,p}\right)^{\frac{s-1}{s}}+\left(\frac{h\,p}{r\,p}\right)^{\frac{s-1}{s}}-2\right]$	$288\frac{s}{s-1}(l\,p)(l\text{-}p\ cap)\left[(R_p)^{\frac{s-1}{2s}}-1\right]$
" " cu ft low-press gas.....	$144\frac{s}{s-1}(l\,p)\left[\left(\frac{r\,p}{l\,p}\right)^{\frac{s-1}{s}}+\left(\frac{h\,p}{r\,p}\right)^{\frac{s-1}{s}}-2\right]$	$288\frac{s}{s-1}(l\,p)\left[(R_p)^{\frac{s-1}{2s}}-1\right]$
" " " high " " hot....	$144\frac{s}{s-1}(l\,p)\left(\frac{h\,p}{r\,p}\right)^{\frac{1}{s}}\left(\frac{r\,p}{l\,p}\right)\left[\left(\frac{r\,p}{l\,p}\right)^{\frac{s-1}{s}}+\left(\frac{h\,p}{r\,p}\right)^{\frac{s-1}{s}}-2\right]$	$288\frac{s}{s-1}(l\,p)(R_p)^{\frac{s+1}{2s}}\left[(R_p)^{\frac{s-1}{2s}}-1\right]$
" " " " " " cold....	$144\frac{s}{s-1}(h\,p)\left[\left(\frac{r\,p}{l\,p}\right)^{\frac{s-1}{s}}+\left(\frac{h\,p}{r\,p}\right)^{\frac{s-1}{s}}-2\right]$	$288\frac{s}{s-1}(h\,p)\left[(R_p)^{\frac{s-1}{2s}}-1\right]$
Displacement per min.....	$\frac{d_1^2 l_1 n}{2\,200}$ for l-p cylinder	$\frac{d_1^2 l_1 n}{2\,200}$ for l-p cylinder
Low-press capacity per min	$D_1\left[1+c_1-c_1\left(\frac{r\,p}{l\,p}\right)^{\frac{1}{s}}\right]$ for l-p cylinder	$D_1\left[1+c_1-c_1R_p^{\frac{1}{2s}}\right]$ for l-p cylinder
High " " hot per min...	$D_1\left[1+c_1-c_1\left(\frac{r\,p}{l\,p}\right)^{\frac{1}{s}}\right]\left(\frac{l\,p}{r\,p}\right)\left(\frac{r\,p}{h\,p}\right)^{\frac{1}{s}}=(l\text{-}p\ cap)\left(\frac{l\,p}{r\,p}\right)\left(\frac{r\,p}{h\,p}\right)^{\frac{1}{s}}$	$D_1\left[1+c_1-c_1R_p^{\frac{1}{2s}}\right]+R_p^{\frac{s+1}{2s}}=(l\text{-}p\ cap)+(R_p)^{\frac{s+1}{2s}}$
" " " cold " . " ...	$D_1\left[1+c_1-c_1\left(\frac{r\,p}{l\,p}\right)^{\frac{1}{s}}\right]+R_p=(l\text{-}p\ cap)+R_p$	$D_1\left[1+c_1-c_1R_p^{\frac{1}{2s}}\right]+R_p=(l\text{-}p\ cap)+R_p$
M e p, referred to 1st stage	$\frac{s}{s-1}(l\,p)\,E_{v1}\left[\left(\frac{D_1E_{v1}}{D_2E_{v2}}\right)^{\frac{s-1}{s}}+R_p\left(\frac{D_2E_{v2}}{D_1E_{v1}}\right)^{\frac{s-1}{s}}-2\right]$	$\frac{2s}{s-1}(l\,p)\,E_{v1}\left[(R_p)^{\frac{s-1}{2s}}-1\right]$
Apparent vol effic, 1st stage	$1+c_1-c_1\left(\frac{r\,p}{l\,p}\right)^{\frac{1}{s}}=\frac{l\text{-}p\ cap_1}{D_1}$	$1+c_1-c_1R_p^{\frac{1}{2s}}=\frac{l\text{-}p\ cap_1}{D_1}$

of press must include not only that actually observed, but also that required to ce the velocity with which the air leaves the fan or blower, or,

$$W = 144\,(p_s + p_v)\,Q \tag{32}$$



1. Horsepower for Two-stage Compression of Air, with Best Receiver Press and Perfect Intercooling

: Q = cu ft of air per min, p_s = increase of press, and p_v = velocity head, both in r sq in. Since

$$V = \frac{Q}{60\,A} \qquad p_v = \frac{\delta}{144 \times 2g}\left(\frac{Q}{60\,A}\right)^2, \quad \text{or,}$$

$$W = 144\,Q\left[p_s + \frac{\delta}{144 \times 2g}\left(\frac{Q}{60\,A}\right)^2\right] \tag{33}$$

ich δ = wt per cu ft of gas, and A = area of discharge opening, sq ft.

ıe above equations and es are for the ideal or per- nachine. Actual machines ire greater hp, to allow for ional and other losses. Re- n between the ideal and hp, termed the DIAGRAM OR, varies from 0.5 to 0.85; is, the power actually re- d in average practice will rom 2 to 1.2 that calcu- l for isothermal compres-

For further data on the pression of air, see Sec 15.

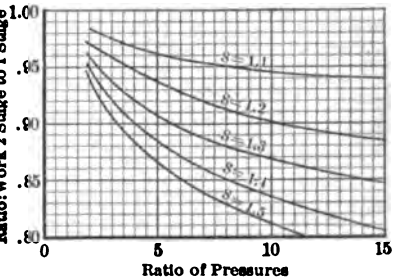


Fig 12. Diagram Showing Saving in Two-stage Air Compression

xample 8. What cylinder dis- ment per min and hp are required to compress adiabatically 1 200 cu ft of air from s press to 60 lb gage, in single-stage compressors, cylinder having 3% clearance?

From Table 7 the apparent vol effic is $1 + 0.03 - (0.03 \times 5) = 94\%$. Assuming inlet air to be at 60° F, the real effic (Eq 30) is

$$\frac{0.94}{1 + 0.4[1.58 - 1]} = 76\%.$$

The displacement per min will be 1 580 cu ft, and the work

$$144 \times 3.5 \times 1\,200[5^{0.29} - 1] = 110 \text{ hp}.$$

Similar results are obtainable from Fig 9, 10.

Example 9. What are the high and low-press capacities of a 2-stage compressor, and 17 by 12 in cyl, speed 120 rev per min, with 3% clearance in low-press cyl, operating at best receiver press and perfect intercooling? Compression is to 9 atmos abs. What power is required and how does it compare with that for a single-stage machine?

From Eq 28, receiver press = $(15 \times 135)^{\frac{1}{2}} = 45$ lb; hence, low-press apparent effic is $1 + 0.03 - (0.03 \times 3) = 96.5\%$. Assuming an initial temp of 60° F, the real effic (Eq 30) is $0.965 \div [1 + 0.4(1.36 - 1)] = 85\%$. Low-press capacity will be 85% of displacement, or 323 cu ft. High-press capacity hot (Table 8) is $323 \times (\frac{1}{3})^{0.78}$, or 43 cu ft; the high-press capacity cold being $323 \div 9$, or 36 cu ft. Power required for an ideal compressor is (Table 8 or Fig 11) 53.6, and from Fig 12 is seen to be about 84% of that for compression in one stage.

4. Power, Fuel Consumption and Thermal Efficiency of Piston Steam and Air Engines

Simple engines are those in which the fluid does work on the piston, and is then exhausted to the atmos or to a condenser. The fluid may be admitted for the entire stroke, or may be cut off at some point in the stroke and then expanded. Similarly, the exhaust may be for an entire stroke, or for only a part, followed by compression. Fig 13 shows the series of operations forming a cycle, for which 2 strokes are required. When the fluid is admitted to one end of cylinder only, the engine is single-acting; if to both ends, double-acting. Referring to Fig 13: A = admission, inlet valve opens; B = cutoff, inlet valve closes; C = release, exhaust valve opens; D = compression, exhaust valve closes; A–A' = admission at constant vol; A'–B admission at constant press; B–C = expansion; C–C' = exhaust at constant vol; C'–D = exhaust at constant press.

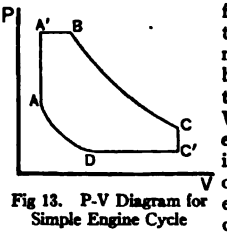


Fig 13. P-V Diagram for Simple Engine Cycle

Expansion curve BC may be of any form, but in steam practice it is a hyperbolic curve, so that $PV = K$; for air, it is usually assumed to be adiabatic, so that $PV^{1.41} = K$. Two sets of expressions are given in Table 9, one for

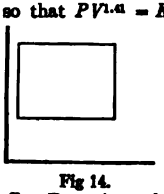


Fig 14. Zero Expansion and Compression

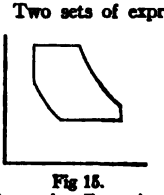


Fig 15. Incomplete Expansion and Compression

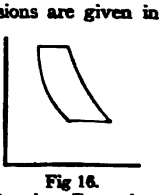


Fig 16. Complete Expansion and Compression

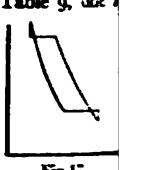


Fig 17. Over Expansion and Compression

$PV = K$, the other a general one for $PV^s = K$, where s may have any value other than 1. Four degrees of expansion and compression may exist (Figs 14, 15, 16, 17). In complete expansion the fluid expands in the cyl to the exact press of the exhaust, in incomplete expansion it fails to reach exhaust pres-

in over expansion it falls below it. Similarly, for complete compression the n trapped in the cyl at the end of exhaust is compressed to the initial press.

dicated horsepower (i h p) of an engine is given by the expression + 33 000, where p is the mean effective press (m e p)$_i$ lb per sq in, L the h of stroke, ft, a the net area of piston of piston minus area of rod, if there be sq in, n number of cycles per min. The tity LAn + 33 000 is the ENGINE CONSTANT Table 2 and its explanation).

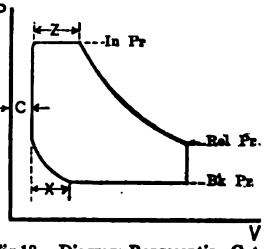


Fig 18. Diagram Representing Cut-off, Compression, and Pressures in Steam Engine Cycle

nsumption of engines is expressed in , or in lb of high-press fluid per hp hr, the r value being termed the WATER RATE. he equations, the letters have following nations: Z = cutoff = fraction of stroke leted when inlet valve closes; X = com-ion = fraction of stroke remaining when ust valve closes; c = clearance expressed fraction of the stroke; δ_1 = wt of a cu ft uid at the high press, as found from steam tables; (in pr), (bk pr), and (rel initial, exhaust, and release press, in lb per sq in abs (Fig 18).

ble 9. Expressions for m e p and Consumption of Steam or Air Engines

	For $PV=K$	For $PV^s=K$
p	$(\text{in pr})\left[Z+(Z+c)\,\text{Nap log}\frac{1+c}{Z+c}\right]$ $-(\text{bk pr})\left[\begin{matrix}(1-X)+(X+c)\\ \times\ \text{Nap log}\frac{X+c}{c}\end{matrix}\right]$	$(\text{in pr})\left\{Z+\frac{Z+c}{s-1}\left[1-\left(\frac{Z+c}{1+c}\right)^{s-1}\right]\right\}$ $-(\text{bk pr})\times$ $\left\{(1-X)+\frac{X+c}{s-1}\left[\left(\frac{X+c}{c}\right)^{s-1}-1\right]\right\}$
ft of fluid hr, per p	$\frac{13750}{\text{m e p}}\left[(Z+c)-(X+c)\frac{\text{bk pr}}{\text{in pr}}\right]$	$\frac{13750}{\text{m e p}}\left[(Z+c)-(X+c)\left(\frac{\text{bk pr}}{\text{in pr}}\right)^{\frac{1}{s}}\right]$
of fluid per per i h p	$\frac{13750}{\text{m e p}}\left[(Z+c)-(X+c)\frac{\text{bk pr}}{\text{in pr}}\right]\delta_1$	$\frac{13750}{\text{m e p}}\left[(Z+c)-(X+c)\left(\frac{\text{bk pr}}{\text{in pr}}\right)^{\frac{1}{s}}\right]\delta_1$
l press	$\left(\frac{Z+c}{1+c}\right)(\text{in pr})$	$\left(\frac{Z+c}{1+c}\right)^s(\text{in pr})$

r complete expansion and compression Z and X have special values as follows:

For $PV=K$	For $PV^s=K$	
$Z=(1+c)\frac{\text{bk pr}}{\text{in pr}}-c$	$Z=(1+c)\left(\frac{\text{bk pr}}{\text{in pr}}\right)^{\frac{1}{s}}-c$	(34)
$X=c\left(\frac{\text{in pr}}{\text{bk pr}}\right)-c$	$X=c\left(\frac{\text{in pr}}{\text{bk pr}}\right)^{\frac{1}{s}}-c$	(35)

iagram factor of an engine is the ratio of the m e p actually realized in the to that calculated as above. Actual m e p is always less, due to valve in-erence with steam flow and other causes. Values of diagram factors for erent types of engines are given under description of engines, averaging about to 0.7. SHAFT OR BRAKE HP of an engine is always less than the i h p, due friction losses. Ratio of brake hp to i h p is the MECHANICAL EFFICIENCY.

Actual consumption of engines is greater than the above values, du leakage past valves and from one side of piston to other, and to condensa of steam in the cyl. During exhaust stroke the cyl tends to cool down, s the temp of exhaust steam is relatively low. On admission of high-press ste some is condensed on striking the cooler cyl walls, allowing more to enter.

Missing water is the difference between actual and computed consumpt It is difficult to predict, as the condition of the engine (frequently unkno affects it greatly. Following empiric formulas (Perry) are in common use.

$$\text{For non-condensing engines, } R = m\,\frac{1+\frac{1}{Z}}{d\sqrt{N}}$$

$$\text{For condensing engines, } R = 120\,\frac{1+\frac{1}{Z}}{d\sqrt{ns}}$$

in which R is missing water divided by computed consumption, Z the ratio d the diam of cyl, in, N the rev per min, n the cycles per min, p the initial pre and m a constant varying from 5, in engines in very good condition and w jackets, to 30 or more in engines without jackets.

Multiple-expansion engines are those in which the steam, after do work in one cyl, is exhausted to one or more cyls and does more work by furth expansion. Amount of work which 1 cu ft of steam can do in expanding b tween initial and back press is independent of the number of cyls in wh expansion takes place, but a greater range of expansion can be had in a mul ple than in a single-cyl engine. This, together with a decrease in condensi and leakage losses, is the reason for employing compound and triple-exp sion engines. Should expansion occur in 2 stages, the engine is termed a co pound, a triple if in 3 stages, and quadruple if in 4. But, 2 cyls may be us instead of 1 large one for the lower press cyl, so that a compound is not nec sarily a 2-cyl, or a triple a 3-cyl engine. Exhaust from 1 cyl may be led direc to the next, or to an intermediate receiver. Should the fluid be heated betwe cyls, the receiver is a reheating receiver. It is considered good practice reheat air, but not steam. Since the size of receiver, arrangement of cran type of expansion curve, clearance, cutoff, and compression, in each cyl all ent into the expressions for m e p for multiple-expansion engines, these expressio become too complicated for presentation here (1). (See textbooks.)

Expressions for cu ft or lb of fluid consumed, as given for simple engines, may be us for multiple-expansion engines as follows: Z, X, c, and the pressures may be taken i either cyl and the m e p taken as referred to that cyl for which the values of Z, etc. a taken. The m e p referred to 1 cycle is a hypothetical m e p, which, if existing in a sim cyl of the same size as that to which it is referred, gives the same hp as the whole engi

$$\text{For compound engines: } \begin{cases} (\text{m e p}) \text{ ref to high press} = (\text{m e p})_H + (\text{m e p})_L \dfrac{D_L}{D_H} \\ (\text{m e p}) \text{ ref to low press} = (\text{m e p})_H \dfrac{D_H}{D_L} + (\text{m e p})_L \end{cases}$$

where D_H and D_L are the displacements of the high and low-press cyls.

Thermal effic of steam engines is the ratio of output to input. Redu ing both to 1 hp, this becomes thermal effic = 2 545 ÷ (water rate × H_1), in w H_1 is the heat content of 1 lb of steam above the feed-water temp, for value which see Art 12.

Thermal effic of air engines is generally considered as the work obtain from the engine for a given quantity of air, divided by the work to compr

ame quantity. Should there be preheating and reheating, the work-alent of the heat so supplied must be added to the work of compression. he work of engines, see above; for work of compressors, see Sec 15. Work-alent of the heat of preheat or reheat will be $778 w C_p (T_2 - T_1)$ where w of air, C_p = specific heat at constant press = 0.24 for air, and $(T_2 - T_1)$ umber of degrees through which the air is heated.

mple 10. Find the hp which a 15 by 18-in double-acting engine, with 5% clear-running at 200 rev per min, could probably furnish at the belt pulley for following ions. Initial press, 125 lb gage; back press, atmosphere; cutoff, ½; compression, What is the probable steam consumption? Taking s as 1 (as is general custom am work), from Table 9 the ideal m e p would be

$$140\left[0.05 + 0.55 \text{ Nap log } \frac{1.05}{0.55}\right] - 15\left[0.8 + 0.25 \text{ Nap log } \frac{0.25}{0.05}\right] = 102.$$

e a diagram factor of 80%, and a mech effic of 85%, as fair values. Then prob-i e p = 0.8 × 102 = 82.

piston speed = 2 × 200 × 1.5 = 600, hence from Table 2 hp constant is 3.21. h p will then be 3.21 × 82 = 263 and the b h p, 0.85 × 263 = 223. From Table indicated steam per i h p per hr is $\frac{13\,750}{82}\left[0.55 - 0.25 \times \frac{15}{140}\right] \times 0.31 = 27.5$ lb.

Eq 37 $R = \frac{30 \times (1+2)}{15\sqrt{200}} = 0.42$. The total steam per hr will then be 27.5 × 1.42 = 10 270 lb.

mple 11. If the engine in above problem be run for best economy, complete expan-nd compression must be had; hence (Eq 34, 35) $Z = 1.05 \times \frac{15}{140} - 0.05 = 0.062$ and $0.05 \times \frac{140}{15} - 0.05 = 0.42$. From Table 9, the ideal m e p is 21. Probable s then 46. The indicated consumption (Table 9) is $\frac{13\,750}{17}\left[0.112 - 0.47 \times \frac{15}{140}\right]$ 1 = 15.5, and R (Eq 37) is 2.4. Total steam per hr is 15.5 × 3.4 × 54 = 2 846 lb.

Power, Fuel Consumption and Thermal Efficiency of Internal Combustion Engines

these engines the heat is developed directly in the working cyl by burning el therein. Such engines may be divided into 2 classes; (*a*) those burn-e fuel at constant vol; (*b*) those at constant press. Fig 19 and 20 show the s on which they operate. Fig 19 s the Otto cycle, where

3 = adiabatic compression
) = " expansion
: = heating at constant volume
l = cooling " " "

0 shows the Diesel cycle, where

3 = adiabatic compression
) = " expansion
: = heating at constant pressure
l = cooling " " volume

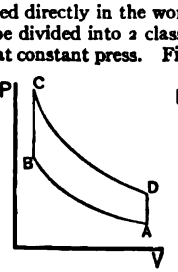


Fig 19. P-V Diagram of Otto Cycle

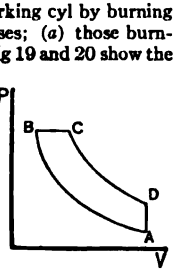


Fig 20. P-V Diagram of Diesel Cycle

ctually operated, both types draw in a new charge in each cycle, and expel roducts. These transfer phases may each take 1 stroke, or occur at the dead-r periods. In first case, the engine is a 4-cycle, in second, a 2-cycle, the 4 indicating the number of strokes per cycle. Expressions for m e p, thermal and fuel consumption, for Otto and Diesel cycles are given in Table 10.

Table 10. Expressions for m e p, Efficiency, and Fuel Consumption for (and Diesel Cycles

	Otto cycle	Diesel cycle
m e p.....	$\frac{JQ_1\left[1-\left(\frac{P_a}{P_b}\right)^{\frac{s-1}{s}}\right]}{144\,V_a\left[1-\left(\frac{P_a}{P_b}\right)^{\frac{1}{s}}\right]}$ or $5.4\frac{H_1}{s+1}\left[1-\left(\frac{P_a}{P_b}\right)^{\frac{s-1}{s}}\right]$	$\frac{J\left\{Q_1-C_vT_a\left[\left(1+\frac{Q_1}{C_pT_b}\right)^{s}-1\right.\right.}{144\,V_a\left[1-\left(\frac{P_a}{P_b}\right)^{\frac{1}{s}}\right]}$
Thermal efficiency..	$1-\left(\frac{P_a}{P_b}\right)^{\frac{s-1}{s}}$ or $1-\left(\frac{V_b}{V_a}\right)^{s-1}$ or $1-\frac{T_a}{T_b}$	$1-\frac{c_vT_a\left[\left(1+\frac{Q_1}{C_pT_b}\right)^{s}-1\right.}{Q_1}$
Fuel consumption per hr per i h p	$\frac{2\,545}{EH}$	$\frac{2\,545}{EH}$

J = Joule's equiv = 778; Q_1 = heat added per lb of working gases in cyl; P_a = pr at beginning of compression; P_b = press at end; V_a = vol per lb of air-gas mixture a ginning and V_b at end of compression; C_p and C_v = specific heats of gases at consta press and constant vol; H_1 = b t u per cu ft of gas or vapor; H = b t u per cu ft gas or per lb of oil; E = thermal effic, a = the air required to burn 1 cu ft gas or vap T_a = abs temp at beginning of compression; T_b = abs temp at end of compression.

Efficiency of Otto cycle depends only upon the degree of compression and is in pendent of amount of heat added. It may be expressed in terms of temp, vol or pre before and after compression, the latter being commonest. Table 11 gives values f efficiency for different compression ratios for the usual value of $s = 1.4$. Efficiency

Table 11. Values of Otto Cycle Efficiency

$\frac{P_b}{P_a}$	$1-\left(\frac{P_a}{P_b}\right)^{\frac{1}{s}}$	$1-\left(\frac{P_a}{P_b}\right)^{\frac{s-1}{s}}=E$	$\frac{P_b}{P_a}$	$1-\left(\frac{P_a}{P_b}\right)^{\frac{1}{s}}$	$1-\left(\frac{P_a}{P_b}\right)^{\frac{s-1}{s}}=$
3	0.544	0.269	5.8	0.715	0.395
3.2	0.564	0.283	6	0.722	0 401
3.4	0.583	0.295	7	0.751	0.427
3.6	0.599	0.307	8	0.774	0 448
3.8	0.615	0.317	9	0,792	0.466
4	0.628	0.327	10	0.807	0 482
4.2	0.641	0.336	15	0.855	0.535
4.4	0.653	0.345	20	0.882	0 573
4.6	0.664	0.353	25	0.900	0 601
4.8	0.673	0.361	30	0.912	0 622
5	0.683	0.369	35	0.921	0 638
5.2	0.672	0.376	40	0.928	0 651
5.4	0.700	0.382	45	0.934	0 663
5.6	0.708	0.389	.....	.....	.....

Diesel cycle varies with the degree of compression, and with amount of heat ad In practice, efficiency is further decreased at heavy loads due to incomplete combus when large amounts of fuel are added. The m e p in each case is work divide change in volume. In above expressions, work is the product JQ_1E, and vol cha is expressed as a function of original vol. Heat liberated per cycle Q_1 divide vol of mixture admitted, or vol change, is the heat content per cu ft of mixtu

e. the m e p expression for Otto cycle may have the simpler form given in e 10. Table 12 gives the quantity $\frac{H_1}{a+1}$ or B t u per cu ft of air-fuel mixture for ommon fuels, at 32° F and 14.7 lb. For other conditions of press and temp, the s vary directly as the abs press and inversely as abs temp. Hence, solutions of prob- on the Otto cycle are chiefly a matter of finding tabular quantities. For solution e p by the other expressions, the heat liberated per lb of stuff in cyl must be known. idering Va to be the vol of 1 lb of mixture external to the cyl,

$$V_a\left[1-\left(\frac{P_a}{P_b}\right)^{\frac{1}{s}}\right]\frac{H_1}{a+1}=Q_1.$$

Table 12. Values of B t u per Cu Ft of Best Air and Gas Mixture

Fuel	$\frac{H_1}{a+1}$ at 32° F and 14.7 lb	Fuel	$\frac{H_1}{a+1}$ at 32° F and 14.7 lb
rbon monoxide...	100	Average coal gas.....	90
drogen.....	88	" oil gas.....	97
thane.....	91	" coke oven gas.....	90
nzene.....	103	" blast-furnace gas.....	60
ohol.....	101	" water gas.....	88
solene.....	107	" anthracite producer gas	67
ade oil.....	110	" lignite producer gas....	69
rosene.....	107	.....	.....

te. — Values at any other press and temp = values at 32° and 14.7 lb × $33.45\frac{p}{T}$, e p is press in lb per sq in and T is abs temp in deg F.

ble 11 gives values for $1-(P_a+P_b)^{\frac{1}{s}}$ and as the air-fuel mixture is largely air the er lb may be considered the same as air, values for which are given in Table 20. e fuel mixture external to engine is generally at atmos press and temp, an aver value is not greatly in error.

r liquid fuels the value for $\frac{H_1}{a+1}$ averages about 105, for city and natural 90, and producer gases 70. Values given for m e p and effic are all for perfect es. The diagram factor varies far more than for steam engines, but a rough e is 45 to 55%. Values for different types of engines are given in Sec 40.

ample 12. Find full-load hp, fuel consumption, and thermal effic of a single- g Otto 4-cycle 3-cyl 10 by 12-in engine, running at 500 rev per min, with a compres- press of 90 lb gage; fuel, alcohol. Compression ratio = (90 + 15) ÷ 15 = 7. Effic le 10) = $1-\left(\frac{1}{7}\right)^{\frac{s-1}{s}}$ = 0.427; or, from Table 11 = 0.427. Assuming diagram factor ; real effic = 0.21. Fuel per hp hr (Table 10), and value of 12 100 B t u per lb ol (Table 26) = 2 545 ÷ (0.21 × 12 100) = 1. This value is the probable consumption, the probable effic of 0.21 was used. Same result could be found by using the hypo- al value 0.427, and dividing by diagram factor 0.5. B t u per cu ft mixture, at o° F and atmos press (Table 12) = 95.5; hence m e p = 5.4 × 95.5 × 0.427 = 220 Applying same diagram factor, real m e p = 110. Piston speed = (3 × 500) ÷ 2 0. Hp constant from Table 2 = 1.785; hence, hp = 196.

ample 13. Find the m e p, fuel consumption, and thermal effic, of a single-cyl, le Diesel engine, operating on crude oil at full load. Compression ratio, 40. At etical full load in a Diesel there would be heat liberated due to best mixture, but lly there is always excess air. Assuming best mixture, Q_1 = 13 × 104 × 0.93 = , for temp of 60° F external to cyl. From the expressions following Eq 42, the temp equals the press ratio raised to $(s-1)\div s$ power, hence $T_b = 2.87 \times T_a$ = 1 500 abs;

$$\text{hence } E = 1-\frac{0.17\times 520\left(1+\frac{1\,250}{0.24\times 1\,500}\right)^{1.4}-1}{1\,250}=0.515$$

Assuming diagram factor of 0.5, real thermal effic becomes 26%. Fuel consumption assuming 19 000 B t u per lb oil, is 2 545 ÷ (0.26 × 19 000) = 0.51 lb.

M e p = (778 × 1 250 × 0.515) ÷ (144 × 0.3 × 0.93) = 290 lb.

Applying same diagram factor, real m e p = 145 lb.

6. Heat and Temperature Units

Thermometer scales in common use are Cent and Fahr, usually the la in American engineering practice. The fixed points on each are the freez and boiling points of water, at standard press of 760 mm of mercury. Fre ing point is denoted by 0° and 32°, boiling point, by 100° and 212°, on the C and Fahr scales respectively. 1° F = 5/9° C, or 1° C = 9/5° F. Temp F = temp C + 32, or temp C = 5/9 (temp F − 32). (See also Sec 37.)

Absolute temperature. Scales given above are purely arbitrary, and not be used in thermodynamic calculations for which abs temps are requir The more permanent gases, as air, hydrogen, oxygen, etc, or those much ab their condensation temp, when cooled at constant press, lose a definite amo in vol per degree of cooling. If cooled at constant vol, there is a definite crease in press of 1/273 per deg C, which is the same as the decrease in vol constant-press cooling. Considering a unit press or vol at 0° C, cooled thr 273°, there would be zero press if the cooling were at constant vol, or zero if cooling were at constant press. Temp of −273° C is therefore called abs zero Cent, and 0° on the same scale becomes 273° C abs. Similarly, 0 Fahr scale is 460° abs. The above values are in the nearest round numbe More exact values are:

0° C scale = 272.85° abs (Regnault)
0° F " = 459.64° " (Marks & Davis)

Units of heat. There are 2 units of heat, one corresponding to each of t above thermometer scales. That corresponding to the Cent scale, or met unit, is the calorie, or heat required to raise the temp of 1 kg of water fr 15° C to 16° C. This unit is sometimes called the large calorie, to distinguish it from the small calorie, which is the heat required to raise the temp of 1 g water an equal amount. The unit used in English-speaking countries is t British Thermal Unit (B t u), and is the heat required to raise the temp 1 lb water from 62° F to 63° F (Peabody) or 1/180 of the heat required to ra 1 lb of water from 32° F to 212° F (2). These values would be identical if the specific heat of water constant. The variation is slight, and for practi purposes the error arising from considering it constant and equal to unity ma be neglected. For detailed discussion of different values of specific heat water, see (2). 1 B t u = 0.252 calorie; 1 calorie = 3.968 B t u.

Mechanical equivalent of heat varies in different latitudes; a commo value in engineering work is 1 B t u = 778 ft lb (Table 13).

Table 13. Power Conversion Factors

Ft lb	B t u	Calorie	Hp sec	Hp min	Hp hr
1	1.26×10^{-3}	0.324×10^{-3}	1.818×10^{-6}	0.303×10^{-4}	5.05×10^{-7}
778	1	0.252	1.414	2.356×10^{-2}	3.93×10^{-4}
3 086	3.968	1	5.61	9.35×10^{-2}	1.56×10^{-3}
550	0.707	0.178	1	1.67×10^{-2}	2.78×10^{-4}
3.3×10^{4}	43.44	10.70	60	1	1.67×10^{-2}
1.98×10^{6}	2.545	641	3 600	60	1

7. Specific Heats

hen substances are so heated that no change of state occurs, as from solid iquid or liquid to vapor, and no heat is conducted away, the temp rises. ntity of heat required to cause temp of equal weights of different substances ise the same number of degrees varies. The standard is the heat required aise 1 lb water 1° F. Ratio of the heat required to raise 1 lb of a substance to that required to raise 1 lb water 1° F is its SPECIFIC HEAT.

eat may be applied to a substance at constant press or at constant volume. irst case the vol changes, and hence work is done; in second case no work lts. As work is done at the expense of heat, more is required to cause a n temp rise in the case of constant-press heating. Hence, there are 2 ific-heat values, C_p for constant press, and C_v for constant vol conditions. , in case of solids and liquids, the change in volume is so small as to be neg- le in engineering work, hence for these substances only one value is reported. ific heats of substances are not constant over the entire range of temps, have the form of equation, $C = a + bt + ct^2$, where a, b, and c are constants. he values of b and c are not definitely determined, and expressions involving ific heats become very complicated when the variable values are used, it is omary to use the constant values and apply correction factors where neces- at end of the calculation. See Tables 14, 15, 16.

Table 14. Specific Heats of Solids

(Compiled from Smithsonian and Landolt-Bornstein Physical Tables)

Substance	Sp heat	At deg F	Substance	Sp heat	At deg F
uminum	0.2089	32	Iron, wrt	0.115	60–212
bestos	0.195	70–210	" "	0.199	1 800–2 200
ass	0.092	70–210	Lava, Ætna	0.26	90–1 400
ickwork	0.21	.....	Lead	0.031	70–212
onze	0.104	70–210	Limestone	0 216	60–212
kspar ($CaCO_3$)	0.2	32–212	Magnetite (Fe_3O_4)	0.156	70–115
rbon, diamond	0.113	52	Manganese	0.124	70–212
" graphite	0.160	52	Marble	0.21	32–212
ment	0 271	82	Mica	0.208	70–210
ay	0.197	.....	Nickel	0.13	932
al	0.2–0.25	.....	Pyrites (FeS)	0.13	60–210
pper	0.094	32–212	Quartz, (SiO_2)	0 174	32
lena (PbS)	0.0466	32–212	"	0.305	750–2 200
ass	0.16–0.18	.....	Rock salt	0.219	55–110
ld	0.032	32–212	Sandstone	0.22	.....
anite	0.192	50–212	Silver	0.056	22–212
matite (Fe_2O_3)	0.1645	60–212	Steel	0.118	70–212
rnblende	0.1952	70–210	Sulphur	0.18	32–120
	0.5	.....	Tin	0.055	70–212
n, cast	0.119	70–212	Zinc	0.0935	32–212

le 15. **Specific Heat of Liquids** (Compiled from Smithsonian Physical Tables)

Substance	Sp heat	At deg F	Substance	Sp heat	At deg F
ohol, ethyl	0.648	100	Mercury	0.033	212
" methyl	0.6	70	Petroleum	0.511	70–135
mmonia	1.07	32	Sea water	0.9 to 1.0	62
" solution	1	.....			
ad (melted)	0.0356	620–590			

Table 16. Specific Heat of Gases

Substance	Specific heat C_p	Specific heat C_v	At deg F	Authority for C_p
Air	0.2389	0.1703	70–212	1
Ammonia	0.5202	0.4011	70–212	1
Benzole	0.2990	0.2131	70–240	1
Carbon dioxide	0.2025	0.1558	60–212	2
" monoxide	0.2425	0.1734	75–210	1
Ethylene	0.4040	0.3404	50–400	2
Hydrogen	3.4100	2.4219	70–212	1
Methane	0.5929	0.1734	65–405	2
Nitrogen	0.2419	0.1715	70–825	3
Oxygen	0.2240	0.1603	70–825	3

1. Wiedeman. 2. Regnault. 3. Holborn-Austin. All values for C_v from (1)

For a mixture of substances the specific heat may be found by Eq 39:

$$\text{sp ht of mixture} = \frac{c_1w_1 + c_2w_2 + c_3w_3}{w_1 + w_2 + w_3}$$

where c_1, c_2, etc, are the specific heats and w_1, w_2, etc, the weights of the components. Equation 40 (Dulong and Petit) is useful where no experimental values are available:

$$\text{Specific heat of solids} = 6.4 \times \text{atomic weight}$$

Quantity of heat absorbed by or removed from a body on heating or cooling is given by $H = Cw\,(t_2 - t_1)$, where H = heat, w = wt, C = specific heat, and t_2 and t_1 temps

8. Coefficients of Expansion

Substances may be heated at constant vol or at constant press, but the change in vol of liquids and solids when heated at constant press is very small and is usually expressed in the form of a coefficient or change per deg. Solids may undergo both linear and volumetric change, the latter being substantially 3 times the former. Liquids are subject to volumetric expansion only. All gases have substantially the same coef over a wide range, which is $\frac{1}{460}$ of the vol at 32° F and atmos press. A gas following this law exactly is termed a perfect gas. Most of the common gases follow the law closely enough for practical purposes, but vapors, being gases only when at temperatures considerably above their condensation points, follow it less closely, so that for them results are only approx. Tables 17 and 18 give the coef of expansion for common substances

Table 17. Coefficient of Linear Expansion of Solids
(Compiled from Smithsonian Tables)

Substance	Coef × 10^6 per deg F	At deg F	Substance	Coef × 10^6 per deg F	At deg F
Aluminum	0.129–0.175	100–1100	Marble	2.565	[illegible]
Brasses and bronzes	0.095–0.117	32–1700	Masonry	0.025–0.05	
Concrete	c.0795	.....	Nickel	0.071	100
Copper	0.0927	32–212	Porcelain	0.023	[illegible]
Glass	0.032–0.05	32–212	Silver	0.1065	100
Ice	0.283	4–30	Steel	0.0735	100
Iron, cast	0.059	100	Tin	0.1241	100
Iron, wrt	0.0635	0–212	Wood, with grain	0.014–0.053	35–[illegible]
Lead	0.1625	100	Wood, cross grain	0.18–0.3	35–[illegible]

Table 18. Coefficient of Cubical Expansion of Liquids and Solids
(Compiled from Smithsonian Tables)

Substance	Coef×10⁴ per deg F	At deg F	Substance	Coef×10⁴ per deg F	At deg F
hol	7.96	−35–160	Mercury	0.1	75–575
nine	6.49	20–140	Petroleum	0.058	75–250
rspar	0.35	50–115	Quartz	0.196	122–140
nd spar	0.08	122–140	Rock salt	0.67	122–140
netite	0.16	32–212	Zincite	0.015	100

9. Pressure, Volume, and Temperature Relations for Gases

For a perfect gas, $PV = wRT$ (41)

ere P = abs press, lb per sq ft; V = vol, cu ft; w = wt, lb; R = the gas nt = 778 $(C_p - C_v)$; T = abs temp, deg F.

When a gas expands, $P_1V_1^s = P_2V_2^s$ (42)

comparison with Eq 41 shows the following relations:

$$= \left(\frac{V_2}{V_1}\right)^s = \left(\frac{T_1}{T_2}\right)^{\frac{s}{s-1}}$$

$$= \left(\frac{P_2}{P_1}\right)^{\frac{1}{s}} = \left(\frac{T_2}{T_1}\right)^{\frac{1}{s-1}}$$

$$= \left(\frac{P_1}{P_2}\right)^{\frac{s-1}{s}} = \left(\frac{V_2}{V_1}\right)^{s-1}$$

elations are shown graph- in Fig 21, the vertical epresenting press ratios, ale at bottom vol ratios, ri: scale at top, temp ra- The curves denote values eft-hand group of curves ations between press and ght-hand, between press np. Thus, for air expand- n 7 atmos to 1 atmos, the tio = 7, final vol will be 4 riginal, and final abs temp 4 times original abs temp.

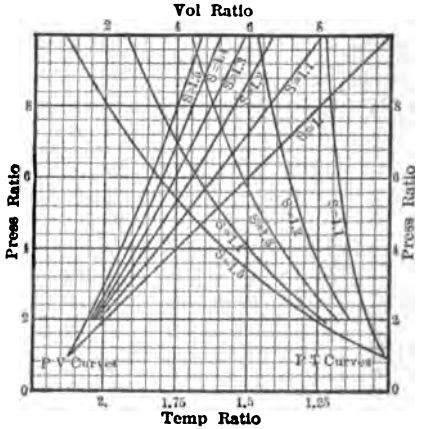


Fig 21. P-V-T Change Relations for Gases

19. Specific Volumes and Densities at 32° F and 14.7 Lb per Sq In and Gas Constants of Common Gases

Gas	Lb per cu ft	Cu ft per lb	Gas constant	Authority
	0.0807	12.39	53.33	Rayleigh
onia	0.0476	21.02	90.47	Leduc
on dioxide	0.1227	8.15	35.08	Rayleigh
monoxide	0.0781	12.81	55.14	Leduc
lene	0.0795	12.58	54.15	Saussure
ogen	0.0056	177.91	765.89	Rayleigh
ane	0.0447	22.35	96.20	Thomson
gen	0.0783	12.77	55.98	Rayleigh
en	0.0892	11.21	48.24	"

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Volume per lb
temp; Table 19
molecular wt of t
press and temp co
δ = density of th
0.00559 lb per cu
as the most reliab
0.00279 × molecu
tions give the valu
into that for any

Vol at a

where T is abs te
under the desired
the upper is the r
certain gas be des
known, simply mu
100 on lower scale
to the top scale, w

Table 20. Volu

Abs press, lb per sq in	20	
8	21.75	22
9	19.11	19
10	17.40	17
11	15.81	16
12	14.50	14
13	13.38	13
14	12.42	12
15	11.60	11
16	10.87	11
17	10.23	10
18	9.66	9
19	9.16	9
20	8.70	8
25	6.96	7
30	5.80	5
35	4.98	5
40	4.34	4
45	3.87	3
50	3.48	3
55	3.17	3
60	2.90	2
65	2.68	2
70	2.49	2
75	2.32	2
80	2.17	2
85	2.05	2
90	1.93	1
95	1.83	1
100	1.74	1
105	1.66	1
110	1.58	1
115	1.51	1
120	1.45	1
125	1.39	1

ions. Density, or lb per cu ft under the actual conditions, is of course the recip- of the vol per lb. The vol per lb dry air at different press and temp being often ed is given in Table 20. Table 21, he accompanying simple equations, the place of tables giving the wt of r and of water vapor contained per f moist air, and lb of moisture per lb for all conditions of saturation, temp arom press, which would be too for insertion.

= temp, deg F
= barom press, in of Hg
= partial press of vapor at $t°$, lb per sq in abs
= partial press of vapor at $t°$, in of Hg
= partial press of saturated air, lb per sq in abs
= partial press of saturated air, in of Hg
= cu ft per lb dry air
= abs temp $= 460 + t°$
= density of vapor = wt per cu ft
$= P_B - P_V$

$$= 0.37 \frac{T}{P_A} = 0.7534 \frac{T}{P'_A}$$

apor per lb air $= Z\delta$.
l wt per cu ft of mixture $= (1+Z) \div \delta$

Table 21. Properties of Saturated Air

$t°$ F	Vapor press		Wt per cu ft $= \delta$
	Lb per sq in	In of Hg	
20	0.0492	0.1026	0.000176
25	0.0622	0.1298	0.000222
30	0.0807	0.1640	0.000276
35	0.1000	0.2034	0.000340
40	0.1217	0.2477	0.000410
45	0.1475	0.3002	0.000492
50	0.1780	0.3625	0.000587
55	0.2140	0.4357	0.000700
60	0.2562	0.522	0.000828
65	0.3054	0.622	0.000977
70	0.3626	0.739	0.001148
75	0.4288	0.873	0.001346
80	0.505	1.029	0.001570
85	0.594	1.209	0.001832
90	0.696	1.417	0.002131
95	0.813	1.655	0.002467
100	0.946	1.926	0.002851
105	1.098	2.236	0.003282
110	1.271	2.589	0.003766

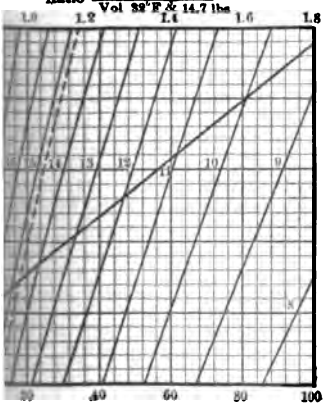


Diagram to Determine Change in Specific Vol, with Press and Temp

For non-saturated conditions, results sufficiently accurate for most purposes can be obtained by finding dew point from tables or curves (Sec 23, Fig 1), computed for standard atmos conditions.

Example 14. Find wt of 1 cu ft of air, saturated with moisture at 100° F; wt of the air alone for the same conditions; and wt of water per lb of air, for barom reading of 27 in of Hg. For 100° the vapor press is 0.946 lb per sq in, the total press at 27 in of Hg is 13.25 lb per sq in; hence, the partial press of air is $13.25 - 0.95 = 12.30$. Wt of vapor per cu ft (Table 21) is 0.0028 lb. Wt of air per cu ft is $1 + Z$, or $1 + \left(0.37 \times \frac{510}{12.3}\right) = 0.0653$, and total wt $= 0.0653 + 0.0028 = 0.0681$ lb. Water per lb air $= 0.0028 \times 15.3 = 0.0429$ lb.

Example 15. Consider the above problem for air 54% saturated. The dew point (Sec 23, Art 1) is 80° F. Vapor press is then 0.505 lb per sq in; hence, partial press of the air is $13.25 - 12.75$ lb; wt of vapor $= 0.0016$, wt of air $= 0.0676$, total wt $= 0.0016 + 0.0676 =$ lb. Water per lb of air $= 0.0016 \times 14.8 = 0.0215$ lb, by same method as in Ex 14.

10. Vapors

apor is a gas near its condensation temp, or the gaseous form of a sub- normally a liquid or solid. Any liquid will evaporate or give off vapor temp, but only while the press exerted by the vapor is less than a certain

value, different for each temp and each substance. On reaching this p evaporation ceases, but if the temp be increased evaporation again takes p until a new pressure value is attained. Thus, for a liquid and vapor toget fixing the temp fixes the vapor press, and the curve showing this relatio $P - T$ coordinates is called the vapor press-temp curve, or the vapor-tem curve, of the substance. Relation between press and temp may be expre by an equation.

The press referred to in this paragraph is the press of the vapor alone, and is inde dent of the presence of other gases. Thus, if water be placed in a vessel in which the an absolute vacuum, and the temp maintained at 170° F, the press will rise to 6 lb sq in. If, instead of having a vacuum in the vessel, it had been filled with dry a 170° F and 15 lb, and the water then added, the press would have risen to 21 lb = 15 the air pressure not influencing the water. Here, the values 6 and 15 are termed the tial pressures of the water vapor and air. Vapor in presence of its liquid can not ha temp other than that fixed by the press; under these conditions it is SATURATED. B removed from the liquid it may be heated to a temp greater than the saturated Vapor under these conditions is SUPERHEATED, and the amount of superheat is expre in deg above saturation temp.

Dalton's Law. When a space is filled with 2 or more gases, the press erted equals the sum of all the partial pressures, and the mixture behaves single gas. Ratio of any partial press or vol to that of the whole may be pressed as a function of the weights and gas constants as follows:

$$\frac{P_1}{P_m} = \frac{w_1 R_1}{w_m R_m}, \quad \text{and} \quad \frac{V_1}{V_m} = \frac{w_1 R_1}{w_m R_m}$$

where P = press, V = vol, w = wt, and R = gas constant, the subscript 1 referring single constituent and m to the mixture. If one constituent is a vapor, its partial p is limited by the temp, and, when the press is that corresponding to the temp, the s said to be saturated, since the maximum amount of vapor is present. Actually it is space that is saturated, as the presence of the gas does not affect the vapor, but customary though quite incorrect to say that the gas is saturated. For this case following relation holds true,

$$W_v + W_g = P_v M_v + P_g M_g$$

where W = wt, P = press, M = molecular wt, subscript v = vapor and g = gas. If gas contains less wt of vapor than corresponds to the saturated state, the ratio of actual to saturation is termed the relative humidity (Sec 23, Art 1). If a gas-vapor mixture which relative humidity is less than 100%, is cooled at constant press, the temp at wh moisture begins to condense (that is, when saturation occurs), is called the dew point

Two humidity problems are important, dealing with moisture in the atm and with preparation of liquid fuels for internal-combustion engines. T humidity of the air is found by the psychrometer (Sec 23, Art 1) (1). T problem of humidifying air with liquid fuels can not be treated briefly enou to be inserted here (15).

Sublimation is the process by which solids pass directly to the vapor state with the intervening state of liquid.

11. Fusion and Evaporation

In Art 7 it was stated that if substances were heated without change of s the heat was manifested by a change in temp. But, under continued hea solids and liquids finally reach temp at which fusion, or change from so liquid, and evaporation, or change from liquid to vapor, will occur. Dur either process, the temp is constant if the press be fixed, and the heat ad directly proportioned to the wt of substance changed. Quantity of hea required to fuse or evaporate 1 lb of a substance is the latent heat of fus vaporization, and is exactly equal to the quantity which must be remo cause the condensation or solidification of 1 lb of the substance. LATENT

APORIZATION varies considerably with change of press, but the heat of fusion actically constant. Any change in the latter is of little importance in en-

e 22. **Temperature and Latent Heat of Fusion** (From Smithsonian Tables)

ibstance	Melting point, deg F.	Latent heat of fusion, B t u	Substance	Melting point, deg F.	Latent heat of fusion, B t u	Substance	Melting point, deg F.	Latent heat of fusion, B t u
ninum....	1 217	139	Iron.....	2 770	40–90	Potas sulphate	372	....
mony....	1 166	.....	Lea	620	10	Silicon.....	2 588	....
um.....	1 563	.....	chloride.....	940	.....	Silver.....	1 760	38
rate.....	1 100	.....	Magnesium....	1 205	.....	chloride....	850	....
uth.....	518	24	chloride.....	1 306	.....	nitrate.....	417	....
x.....	1 040	.....	sulphate.....	129	.....	sulphate....	1 230	....
: acid.....	365	.....	Manganese....	2 237	.....	Sodium.....	207	57
nium.....	610	66	chloride.....	189	.....	chloride....	1 463	....
ium.....	1 481	....	sulphate.....	129	.....	hydrate....	140	....
oride.....	1 404	73	Mercury.....	−38	5	nitrate.....	600	117
on.....	7630	.....	Mercuric chl'de	554	.....	chlorate....	527	....
uide.....	−70	.....	Nickel.....	2 642	8.3	carbonate..	1 565	....
ulphide...	−170	.....	nitrate.....	135	.....	sulphate....	1585	....
mium....	2 740	.....	sulphate.....	210	.....	Sulphur.....	235–245	17
me alum.	192	.....	Phosphorus...	111	9	Stannic chl'de	−27	....
ılt.....	2 714	.....	Platinum.....	3 190	49	stannous "	482	....
er.....	1 980	77	Potassium.....	144	29	Tin.....	450	25
ic chloride	930	.....	carbonate...	1 544	.....	Zinc.....	788	58
ous.....	815	.....	chlorate.....	680	.....	chloride....	504	....
.....	1 945	.....	chloride.....	1 436	.....	nitrate.....	97	....
le.....	237	21	nitrate.....	644	88	sulphate....	122	....

le 23. **Temperature and Latent Heats of Evaporation at Atmos Press**
(Compiled from Smithsonian and Landolt-Bornstein Physical Tables)

stance	Temp of evap, deg F.	Latent heat of evap, B t u per lb	Substance	Temp of evap, deg F.	Latent heat of evap, B t u per lb	Substance	Temp of evap, deg F.	Latent heat of evap, B t u per lb
ic acid.	245	153	Hydrobr'c acid	90	.....	Phosphorus	558	.....
ylene..	−121	.....	Hydrochl'c "	118	.....	trichloride	167	.....
.....	−314	80	Iodine.....	>200	43	trioxide....	343	.....
ethyl.	172	372	Iron.....	4 442	.....	trisulphide	914	.....
thyl ..	151	482	nitrate.....	257	.....	Potassium...	1 372	.....
inum.	3 272	.....	Lead.....	2 777	.....	Silicon chl'de	136	.....
mony	2 622	.....	Magnesium....	2 047	.....	Silver.....	3 552	.....
ole....	212	238	nitrate.....	289	.....	Sodium.....	1 382	.....
uth...	2 607	.....	Manganese....	3 452	.....	Sulphur.....	837	.....
ine...	142	82	Mang'se chl'de	223	.....	dioxide....	12.6	.....
nium..	1 440	.....	nitrate.....	265	.....	trioxide...	114	.....
on.....	6 510	.....	Mercury.....	674	.....	Sulph'c acid.	640	.....
ulphide	115	151	chloride (ic).	579	.....	Tin.....	4 118	.....
rine...	−29	.....	nitrate.....	278	.....	chloride(ic)	237	.....
mium.	4 000	.....	Nitric acid....	187	.....	" (ous)	1 143	.....
rate...	258	.....	oxide.....	−243	.....	Zinc.....	1 686	.....
er.....	4 192	.....	Nitrogen.....	−412	90	chloride...	1 347	.....
rate...	338	.....	Nitrous oxide .	192	.....	nitrate....	268	.....
'chl'de	1 820	.....	Oxygen.....	−297	110			

gineering, as fusion or solidification in most cases takes place at atmos pr
For equations expressing the heats of fusion and vaporization in terms of
quantities and their derivation, see Bib (8). The temp and latent heat
fusion, and temp and latent heats of evaporation, at atmos press, for
substances are given in Tables 22 and 23.

12. Properties of Steam

Properties of dry saturated and superheated steam are given in Tables
25, abridged from (2). For discussion of sources of data, see the original ta

Table 24. Properties of Saturated Steam

(Condensed from Marks and Davis's Steam Tables and Diagrams, 1909, by permis
of the publishers, Longmans, Green & Co)

Vacuum, in of mercury	Abs press, lb per sq in	Temp, °F	Total heat above 32° F: In the water h heat-units	Total heat above 32° F: In the steam H heat-units	Latent heat, $L = H - h$ heat-units	Volume, cu ft in 1 lb of steam	Wt of 1 cu ft steam, lb	Entropy of the water	En
29.74	0.0886	32	0.00	1073.4	1073.4	3294	0.000304	0.0000	
29.67	0.1217	40	8.05	1076.9	1068.9	2438	0.000410	0.0162	
29.56	0.1780	50	18.08	1081.4	1063.3	1702	0.000587	0.0361	
29.40	0.2562	60	28.08	1085.9	1057.8	1208	0.000828	0.0555	
29.18	0.3626	70	38.06	1090.3	1052.3	871	0.001148	0.0745	
29.89	0.505	80	48.03	1094.8	1046.7	636.8	0.001570	0.0932	
28.50	0.696	90	58.00	1099.2	1041.2	469.3	0.002131	0.1114	
28.00	0.946	100	67.97	1103.6	1035.6	350.8	0.002851	0.1295	
27.88	1	101.83	69.8	1104.4	1034.6	333.0	0.00300	0.1327	
25.85	2	126.15	94.0	1115.0	1021.0	173.5	0.00576	0.1749	
23.81	3	141.52	109.4	1121.6	1012.3	118.5	0.00845	0.2008	
21.78	4	153.01	120.9	1126.5	1005.7	90.5	0.01107	0.2198	
19.74	5	162.28	130.1	1130.5	1000.3	73.33	0.01364	0.2348	
17.70	6	170.06	137.9	1133.7	995.8	61.89	0.01616	0.2471	
15.67	7	176.85	144.7	1136.5	991.8	53.56	0.01867	0.2579	
13.63	8	182.86	150.8	1139.0	988.2	47.27	0.02115	0.2673	
11.60	9	188.27	156.2	1141.1	985.0	42.36	0.02361	0.2756	
9.56	10	193.22	161.1	1143.1	982.0	38.38	0.02606	0.2832	
7.52	11	197.75	165.7	1144.9	979.2	35.10	0.02849	0.2902	
5.49	12	201.96	169.9	1146.5	976.6	32.36	0.03090	0.2967	
3.45	13	205.87	173.8	1148.0	974.2	30.03	0.03330	0.3025	
1.42	14	209.55	177.5	1149.4	971.9	28.02	0.03569	0.3081	
lb gage	14.70	212	180.0	1150.4	970.4	26.79	0.03732	0.3118	
1.3	16	216.3	184.4	1152.0	967.6	24.79	0.04042	0.3183	
3.3	18	222.4	190.5	1154.2	963.7	22.16	0.04512	0.3273	
5.3	20	228.0	196.1	1156.2	960.0	20.08	0.04980	0.3355	
7.3	22	233.1	201.3	1158.0	956.7	18.37	0.05445	0.3430	
9.3	24	237.8	206.1	1159.6	953.5	16.93	0.05907	0.3499	
11.3	26	242.2	210.6	1161.2	950.6	15.72	0.0636	0.3564	
13.3	28	246.4	214.8	1162.6	947.8	14.67	0.0682	0.3623	
15.3	30	250.3	218.8	1163.9	945.1	13.74	0.0728	0.3680	
17.3	32	254.1	222.6	1165.1	942.5	12.93	0.0773	0.3733	
19.3	34	257.6	226.2	1166.3	940.1	12.22	0.0818	0.3784	
21.3	36	261.0	229.6	1167.3	937.7	11.58	0.0863	0.3832	
23.3	38	264.2	232.9	1168.4	935.5	11.01	0.0908	0.3877	
25.3	40	267.3	236.1	1169.4	933.3	10.49	0.0953	0.3920	
27.3	42	270.2	239.1	1170.3	931.2	10.02	0.0998	0.3962	

Table 24 (*Continued*). — Properties of Saturated Steam

ge in. er in	Abs press, lb per sq in	Temp, °F	Total heat above 32° F: In the water h heat-units	Total heat above 32° F: In the steam H heat-units	Latent heat, $L = H - h$ heat-units	Volume, cu ft in 1 lb of steam	Wt of 1 cu ft steam, lb	Entropy of the water	Entropy of evaporation
.3	44	273.1	242.0	1171.2	929.2	9.59	0.1043	0.4002	1.2681
3	46	275.8	244.8	1172.0	927.2	9.20	0.1087	0.4040	1.2607
3	48	278.5	247.5	1172.8	925.3	8.84	0.1131	0.4077	1.2536
3	50	281.0	250.1	1173.6	923.5	8.51	0.1175	0.4113	1.2468
3	52	283.5	252.6	1174.3	921.7	8.20	0.1219	0.4147	1.2402
3	54	285.9	255.1	1175.0	919.9	7.91	0.1263	0.4180	1.2339
.3	56	288.2	257.5	1175.7	918.2	7.65	0.1307	0.4212	1.2278
.3	58	290.5	259.8	1176.4	916.5	7.40	0.1350	0.4242	1.2218
3	60	292.7	262.1	1177.0	914.9	7.17	0.1394	0.4272	1.2160
.3	62	294.9	264.3	1177.6	913.3	6.95	0.1438	0.4302	1.2104
.3	64	297.0	266.4	1178.2	911.8	6.75	0.1482	0.4330	1.2050
.3	66	299.0	268.5	1178.8	910.2	6.56	0.1525	0.4358	1.1998
.3	68	301.0	270.6	1179.3	908.7	6.38	0.1569	0.4385	1.1946
3	70	302.9	272.6	1179.8	907.2	6.20	0.1612	0.4411	1.1896
3	72	304.8	274.5	1180.4	905.8	6.04	0.1656	0.4437	1.1848
.3	74	306.7	276.5	1180.9	904.4	5.89	0.1699	0.4462	1.1801
.3	76	308.5	278.3	1181.4	903.0	5.74	0.1743	0.4487	1.1755
3	78	310.3	280.2	1181.8	901.7	5.60	0.1786	0.4511	1.1712
.3	80	312.0	282.0	1182.3	900.3	5.47	0.1829	0.4535	1.1665
.3	82	313.8	283.8	1182.8	899.0	5.34	0.1873	0.4557	1.1623
.3	84	315.4	285.5	1183.2	897.7	5.22	0.1915	0.4579	1.1581
3	86	317.1	287.2	1183.6	896.4	5.10	0.1959	0.4601	1.1540
.3	88	318.7	288.9	1184.0	895.2	5.00	0.2001	0.4623	1.1500
.3	90	320.3	290.5	1184.4	893.9	4.89	0.2044	0.4644	1.1461
3	92	321.8	292.1	1184.8	892.7	4.79	0.2087	0.4664	1.1423
3	94	323.4	293.7	1185.2	891.3	4.69	0.2130	0.4684	1.1385
3	96	324.9	295.3	1185.6	890.3	4.60	0.2172	0.4704	1.1348
3	98	326.4	296.8	1186.0	889.2	4.51	0.2215	0.4724	1.1312
3	100	327.8	298.3	1186.3	888.0	4.429	0.2258	0.4743	1.1277
3	102	329.3	299.8	1186.7	886.9	4.347	0.2300	0.4762	1.1242
3	104	330.7	301.3	1187.0	885.8	4.268	0.2343	0.4780	1.1208
3	106	332.0	302.7	1187.4	884.7	4.192	0.2336	0.4798	1.1174
3	108	333.4	304.1	1187.7	883.6	4.118	0.2429	0.4816	1.1141
3	110	334.8	305.5	1188.0	882.5	4.047	0.2472	0.4834	1.1108
3	112	336.1	306.9	1188.4	881.4	3.978	0.2514	0.4852	1.1076
3	114	337.4	308.3	1188.7	880.4	3.912	0.2556	0.4869	1.1045
3	116	338.7	309.6	1189.0	879.3	3.848	0.2599	0.4886	1.1014
3	118	340.0	311.0	1189.3	878.3	3.786	0.2641	0.4903	1.0984
3	120	341.3	312.3	1189.6	877.2	3.726	0.2683	0.4919	1.0954
3	122	342.5	313.6	1189.8	876.2	3.668	0.2726	0.4935	1.0924
3	124	343.8	314.9	1190.1	875.2	3.611	0.2769	0.4951	1.0895
3	126	345.0	316.2	1190.4	874.2	3.556	0.2812	0.4967	1.0865
3	128	346.2	317.4	1190.7	873.3	3.504	0.2854	0.4982	1.0837
3	130	347.4	318.6	1191.0	872.3	3.452	0.2897	0.4998	1.0809
3	132	348.5	319.9	1191.2	871.3	3.402	0.2939	0.5013	1 0782
3	134	349.7	321.1	1191.5	870.4	3.354	0.2981	0.5028	1.0755
3	136	350.8	322.3	1191.7	869.4	3.308	0.3023	0 5043	1.0728
3	138	352.0	323.4	1192.0	868.5	3 263	0.3065	0 5057	1.0702
3	140	353.1	324.6	1192.2	867.6	3.219	0 3107	0.5072	1.0675
3	142	354.2	325.8	1192.5	866.7	3.175	0.3150	0.5086	1.0649
3	144	355.3	326.9	1192.7	865.8	3.133	0.3192	0 5100	1.0624
3	146	356.3	328.0	1192.9	864.9	3.092	0 3234	0.5114	1.0599

Table 24 (*Concluded*). — Properties of Saturated Steam

Gage press, lb per sq in	Abs press, lb per sq in	Temp, °F	Total heat above 32° F: In the water h heat-units	Total heat above 32° F: In the steam H heat-units	Latent heat, $L = H - h$ heat-units	Volume, cu ft in 1 lb of steam	Wt of 1 cu ft steam, lb	Entropy of the water	Entropy of evaporation
133.3	148	357.4	329.1	1193.2	864.0	3.052	0.3276	0.5128	1.[illegible]
135.3	150	358.5	330.2	1193.4	863.2	3.012	0.3320	0.5142	1.[illegible]
140.3	155	361.0	332.9	1194.0	861.0	2.920	0.3425	0.5175	1.[illegible]
145.3	160	363.6	335.6	1194.5	858.8	2,834	0.3529	0.5208	1.[illegible]
150.3	165	366.0	338.2	1195.0	856.8	2.753	0.3633	0.5239	1.[illegible]
155.3	170	368.5	340.7	1195.4	854.7	2.675	0.3738	0.5269	1.[illegible]
160.3	175	370.8	343.2	1195.9	852.7	2.602	0.3843	0.5299	1.[illegible]
165.3	180	373.1	345.6	1196.4	850 8	2.533	0.3948	0.5328	1.[illegible]
170.3	185	375.4	348.0	1196.8	848.8	2.468	0.4052	0.5356	1.[illegible]
175.3	190	377.6	350.4	1197.3	846.9	2.406	0.4157	0.5384	1.[illegible]
180.3	195	379.8	352.7	1197.7	845.0	2.346	0.4262	0.5410	1.[illegible]
185.3	200	381.9	354.9	1198.1	843.2	2,290	0.437	0.5437	1.[illegible]
195.3	210	386.0	359.2	1198.8	839.6	2.187	0.457	0.5488	0.[illegible]
205.3	220	389.9	363.4	1199.6	836.2	2.091	0.478	0.5536	0.[illegible]
215.3	230	393.8	367.5	1200.2	832.8	2.004	0.499	0.5586	0.[illegible]
225.3	240	397.4	371.4	1200.9	829.5	1.924	0.520	0.5633	0.[illegible]
235.3	250	401.1	375.2	1201.5	826.3	1.850	0.541	0.5676	0.[illegible]
245.3	260	404.5	378.9	1202.1	823.1	1.782	0.561	0.5719	0.[illegible]
255.3	270	407.9	382.5	1202.6	820.1	1.718	0.582	0.5760	0.[illegible]
265.3	280	411.2	386.0	1203.1	817.1	1.658	0.603	0.5800	0.[illegible]
275.3	290	414.4	389.4	1203.6	814.2	1.602	0.624	0.5840	0.[illegible]
285.3	300	417.5	392.7	1204.1	811.3	1.551	0.645	0.5878	0.[illegible]
335.3	350	431.9	408.2	1206.1	797.8	1.334	0.750	0.6053	0.[illegible]
385.3	400	444.8	422	1208	786	1.17	0.86	0.621	0.[illegible]
435.3	450	456.5	435	1209	774	1.04	0.96	0.635	0.[illegible]
485.3	500	467.3	448	1210	762	0.93	1.08	0.648	0.[illegible]
535.3	550	477.3	459	1210	751	0.83	1.20	0.659	0.[illegible]
585.3	600	486.6	469	1210	741	0.76	1.32	0.670	0.[illegible]

Dry saturated steam is steam having the temp due to its press, but containing no moisture; it is rarely found in practice, as commercially "dry" steam usually contains from 1 to 4% of water, or is superheated. Quality of steam is designated by the amount of dry steam present in 1 lb of actual steam mixture, or by the degrees of superheat; for example, 95% dry steam means steam and stuff containing 95% of steam and 5% of water, and 100° superheat means steam having a temp 100° above that due to the press. As the values in the saturated table are for 100% dry steam, made from water at 32° F, corrections must generally be applied first for quality and second for the original temp. Two methods are available, first to follow the procedure actually occurring in the boiler, and second, to correct the table values for the heat not added to the steam.

Example 16. Find the heat required to make 1 lb of steam at 100 lb abs press. dry, from feed water at 100° F. By first method there must first be added enough to raise water from 100° F to the boiling point at 100 lb. From the tables the heat of the liquid at 100° F is 68, and at 100 lb is 298, or a difference of 230. If the specific heat of water were unity throughout the entire range, this should be the same as the difference between the temp at 100 lb and 100° F, or 228. But, the resulting error is negligible for practical work, and hence the heat to raise the water to the boiling point from the water temp may be considered as equal to the temp difference. When water has been raised to boiling point, 95% is evaporated; therefore, to the heat necessary to heat water is added 0.95 of latent heat of evaporation, or in this case 0.95×888, hence total heat $230 + 0.95 \times 888 = 1074$. By the second method the total heat from tables is, for

Table 25. Properties of Superheated Steam

pec vol in cu ft per lb, h = total heat, from water at 32° F, in B t u per lb, s = entropy, m water at 32°. (Condensed from Marks and Davis's Steam Tables and Diagrams)

lb. per sq. in.	Temp sat steam	Degrees of superheat									
		0	20	50	100	150	200	250	300	400	500
0	228.0	v 20.08	20.73	21.69	23.25	24.80	26.33	27.85	29.37	32.39	35.40
		h 1156.2	1165.7	1179.9	1203.5	1227.1	1250.6	1274.1	1297.6	1344.8	1392.2
		s 1.7320	1.7456	1.7652	1.7961	1.8251	1.8524	1.8781	1.9026	1.9479	1.9893
0	267.3	v 10.49	10.83	11.33	12.13	12.93	13.70	14.48	15.25	16.78	18.30
		h 1169.4	1179.3	1194.0	1218.4	1242.4	1266.4	1290.3	1314.1	1361.6	1409.3
		s 1.6761	1.6895	1.7089	1.7392	1.7674	1.7940	1.8189	1.8427	1.8867	1.9271
0	292.7	v 7.17	7.40	7.75	8.30	8.84	9.36	9.89	10.41	11.43	12.45
		h 1177.0	1187.3	1202.6	1227.6	1252.1	1276.4	1300.4	1324.3	1372.2	1420.0
		s 1.6432	1.6568	1.6761	1.7062	1.7342	1.7603	1.7849	1.8081	1.8511	1.8908
0	312.0	v 5.47	5.65	5.92	6.34	6.75	7.17	7.56	7.95	8.72	9.49
		h 1182.3	1193.0	1208.8	1234.3	1259.0	1283.6	1307 8	1331.9	1379.8	1427.9
		s 1.6200	1.6338	1.6532	1.6833	1.7110	1.7368	1.7612	1.7840	1.8265	1.8658
0	327.8	v 4.43	4.58	4.79	5.14	5.47	5.80	6.12	6.44	7.07	7.69
		h 1186.3	1197.5	1213.8	1239.7	1264.7	1289.4	1313.6	1337.8	1385.9	1434.1
		s 1.6020	1.6160	1.6358	1.6658	1.6933	1.7188	1.7428	1.7656	1.8079	1.8468
0	341.3	v 3.73	3.85	4.04	4.33	4.62	4.89	5.17	5.44	5.96	6.48
		h 1189.6	1201.1	1217.9	1244.1	1269.3	1294.1	1318.4	1342.7	1391.0	1439.4
		s 1.5873	1.6016	1.6216	1.6517	1.6789	1.7041	1.7280	1.7505	1.7924	1.8311
0	353.1	v 3.22	3.32	3.49	3.75	4.00	4.24	4.48	4.71	5.16	5.61
		h 1192.2	1204.3	1221.4	1248.0	1273.3	1298.2	1322.6	1346.9	1395.4	1443.8
		s 1.5747	1.5894	1.6096	1.6395	1.6666	1.6916	1.7152	1.7376	1.7792	1.8177
0	363 6	v 2.83	2.93	3.07	3.30	3.53	3.74	3 95	4.15	4.56	4.95
		h 1194.5	1207.0	1224.5	1251.3	1276 8	1301.7	1326.2	1350.6	1399.3	1447.9
		s 1.5639	1.5789	1.5993	1.6292	1.6561	1.6810	1.7043	1.7266	1.7680	1.8063
0	373.1	v 2.53	2.62	2.75	2.96	3.16	3.35	3.54	3.72	4.09	4.44
		h 1196.4	1209.4	1227.2	1254.3	1279.9	1304.8	1329.5	1353.9	1402.7	1451.4
		s 1.5543	1.5697	1.5904	1.6201	1.6468	1.6716	1.6948	1.7169	1.7581	1.7962
0	381.9	v 2.29	2.37	2.49	2.68	2.86	3.04	3.21	3.38	3.71	4.03
		h 1198.1	1211.6	1229.8	1257.1	1282.6	1307.7	1332.4	1357.0	1405.9	1454.7
		s 1.5456	1.5614	1.5823	1.6120	1.6385	1.6632	1.6862	1.7082	1.7493	1.7872
0	389.9	v 2.09	2.16	2.28	2 45	2.62	2.78	2.94	3.10	3.40	3.69
		h 1199.6	1213.6	1232.2	1259.6	1285.2	1310.3	1335.1	1359.8	1408.8	1457.7
		s 1.5379	1.5541	1.5753	1.6049	1.6312	1.6558	1.6787	1.7005	1.7415	1.7792
0	397.4	v 1.92	1.99	2.09	2.26	2.42	2.57	2.71	2.85	3.13	3.40
		h 1200.9	1215.4	1234.3	1261.9	1287.6	1312.8	1337 6	1362.3	1411.5	1460.5
		s 1.5309	1.5476	1.5690	1.5985	1.6246	1.6492	1.6720	1.6937	1.7344	1.7721
0	404.5	v 1.78	1.84	1.94	2.10	2.24	2.39	2 52	2 65	2.91	3.16
		h 1202.1	1217.1	1236.4	1264.1	1289.9	1315.1	1340.0	1364.7	1414.0	1463.2
		s 1.5244	1.5416	1.5631	1.5926	1.6186	1.6430	1.6658	1.6874	1.7280	1.7655
0	411.2	v 1.66	1.72	1.81	1.95	2.09	2.22	2.35	2.48	2.72	2.95
		h 1203.1	1218.7	1238.4	1266.2	1291.9	1317.2	1342.2	1367.0	1416.4	1465.7
		s 1.5185	1.5362	1.5580	1.5873	1.6133	1.6375	1.6603	1.6818	1.7223	1.7597
0	417.5	v 1.55	1.60	1.69	1.83	1.96	2.09	2.21	2.33	2.55	2.77
		h 1204.1	1220 2	1240.3	1268.2	1294.0	1319.3	1344.3	1369.2	1418.6	1468.0
		s 1.5129	1.5310	1.5530	1.5824	1.6082	1.6323	1.6550	1.6765	1.7168	1.7541
0	431.9	v 1.33	1.38	1.46	1.58	1.70	1.81	1.92	2.02	2.22	2.41
		h 1206.1	1223.9	1244.6	1272.7	1298.7	1324.1	1349.3	1374.3	1424.0	1473.7
		s 1.5002	1.5199	1.5423	1.5715	1 5971	1.6210	1.6436	1.6650	1.7052	1.7422
0	444.8	v 1.17	1.21	1.28	1.40	1.50	1.60	1 70	1.79	1.97	2.14
		h 1207.7	1227.2	1248 6	1276 9	1303.0	1328.6	1353 9	1379.1	1429.0	1478.9
		s 1.4894	1.5107	1.5336	1.5625	1.5880	1.6117	1 6342	1.6554	1.6955	1.7323
0	456.5	v 1.04	1.08	1.14	1 25	1.35	1.44	1 53	1.61	1.77	1.93
		h 1209	1231	1252	1281	1307	1333	1358	1383	1434	1484
		s 1.479	1.502	1.526	1.554	1.580	1.603	1.626	1.647	1.687	1.723
0	467.3	v 0.93	0.97	1.03	1.13	1 22	1.31	1.39	1.47	1.62	1.76
		h 1210	1233	1256	1285	1311	1337	1362	1388	1438	1489
		s 1.470	1.496	1.519	1 548	1.573	1.597	1.619	1.640	1.679	1.715

a, 1 186. But, there was already in feed water the heat necessary to raise it to 100° F 32° F, or 68 units; also, 5% of latent heat was not utilized, because only 95% of water vaporated; hence, actual total heat = 1 186 − 68 −(0.05 × 888) = 1 074, as be' the superheated condition, the correction for heat in the feed water only is req'

13. Combustion and Combustion H

The process of combustion is exothermic or heat liber
the rapid oxidation of practically only 2 chemical elem
free or in combination with each other. Few natural
elements in the free state; they are generally combine
ner. The formation of the compound took place with th
tion of heat, so that when the fuel is burned there is liber
of heat which would result from the burning of an equiv
C and H, but a greater or less quantity depending upon w
of the compound develops or absorbs heat. The HEAT OF C
heat of formation of the products less the heat of formation
engineering calculations, it is generally sufficiently exact to
bustion as equal to the heat of formation of products, inasm
work the heat of combustion of a fuel should be determi

When H burns the product is H_2O in the gaseous state, equal
originally present. Usually the products pass off at a temp gre
lower temp the vapor will condense to liquid, giving up 970 B t u
per lb of H. Hence, fuels containing H may have 2 heating v
when the products cool to less than 212°, and the LOW VALUE,
definite common exists as to which should be used, nor is it more
clear understanding between the parties interested. Generally,
for boiler and other open fires, and the low values for explo
values are usually obtained by the calorimeter.

Table 26 gives heating values per lb and per cu ft at 32° F and
substances, and for common liquid and gaseous fuels. The lat
but give a general idea of what may be expected from such fuels

Table 26. Heats of Combustion

(Selected from Lucke's Engineering Thermodyn

Symbol	Substance	High value B	
		per lb.	per
C_2H_2	Acetylene	21 500	1 5
C_2H_5OH	Alcohol, ethyl	12 780	1 7
CH_3OH	" methyl	10 250	9
	Anthracite, Pennsylvania, gen. av.	14 500	8
	Benzene, av.	17 950	15
	Bituminous, Pocahontas, gen. av.	14 600	8
	Bituminous, gen. av.	14 500	8
C	Carbon to CO_2	14 500	
C	" to CO	4 450	
CO	Carbon monoxide	4 450	
	Coal gas, av.	4 350	5
	Coke, Connellsville	13 100	8
	Coke, gen. av.	12 800	8
CH_4	Ethane	22 000	12
C_2H_6	Ethylene	21 600	12
	Gasoline	21 500	17
CH_4	Methane	23 500	9
H_2	Hydrogen	61 500	4
	Kerosene	20 000	15
	Lignite, Washington	10 000	7
CH_4	Marsh gas	23 500	10
	Natural gas, av. Pennsyl.	22 000	10
	Oil, gas	22 000	9
C	Graphite	14 600	
	Petroleum, av.	19 900	
	" crude, Pennsyl.	20 000	

leating value of a fuel. To compute it, the chemical analysis must be wn. The fractional quantity of each constituent is then multiplied by its ting value, and the sum of these products is the heating value of the fuel. lyses of solid and liquid fuels are reported in percentages by wt, gaseous s, by volume. Coals are reported in 2 ways: proximate and ultimate anal-, the former ascertaining moisture, volatile matter, fixed carbon and ash, latter the chemical constituents, C, H, S, O, and ash. As the character of volatile matter can not be determined, attempts to compute the calorific ıe of coals from proximate analysis are not very successful, the ultimate ses being generally used. Following formula, by Lucke, applies with fair ıracy over a rather wide range of coals:

$$Q = 14\,544\,C + 27\,000\,V\left[1 - \frac{1}{\frac{C}{V} + 0.5}\right] \qquad (46)$$

hich Q = B t u per lb, C = fractional wt of fixed carbon, V = fractional wt olatile matter. When ultimate analysis shows both O and H, it is assumed ; the O, and as much of the H as can chemically combine with it, exist as er, and hence only the remaining H is available as a heat producer. Fol-ing formula, proposed by Am Soc Mech Engs, is based on this assumption:

$$Q = 14\,600\,C + 62\,000\left(H - \frac{O}{8}\right) + 4\,000\,S \qquad (47)$$

hich Q = B t u per lb, and C, H, O, and S are the fractional wt of carbon, rogen, oxygen, and sulphur per lb of coal.

ample 17: The calorific value of a coal was 13 900 by calorimeter. Proximate ysis: moisture, 1.25; vol matter, 39.03; fixed C, 53.72; ash, 6.00. Ultimate analysis: 1.47; C, 87.57; N_2, 1.09; O_2, 2.87; S, 0; ash, 6.00. Calculated values would be:

) by Eq 46, $14\,544 \times 0.537 + 27\,000 \times 0.39\left[1 - \frac{1}{1.38 + 0.5}\right]$ = 12 750 B t u

) by Eq 47, $14\,600 \times 0.876 + 62\,000\left(0.0247 - \frac{0.0287}{8}\right)$ = 14 085 "

) by calculation from ultimate analysis,

$$14\,544 \times 0.876 + 60\,626\left(0.0247 - \frac{0.0287}{8}\right) = 14\,003 \text{ "}$$

erman and Kropff found that the heating value of oils is a function of density, and ced following expression, which is within 2 to 3% of the calorimetric values:

$$Q = 18\,650 + 40\,(Be' - 10) \qquad (48)$$

= deg on Beaumé hydrometric scale, the usual mode of reporting oils. Stanton Strong revised the constant in Eq 48 for gasolene, and Allen and Strong for kerosene:

$$\left.\begin{aligned} Q &= 18\,320 + 40\,(Be' - 10), \text{ for gasolene} \\ Q &= 18\,440 + 40\,(Be' - 10), \text{ for kerosene} \end{aligned}\right\} \qquad (49)$$

an error of less than 1% in all tests.

ample 18. Find heating value of the following gas high and low values:

CO	H_2	CH_4	C_2H_4	N_2	O_2	CO_2
7	38	40	3	8	2	4

ultiply quantity of each combustible con-ent by its heating value (see table). sults found are for 32° and 14.7 lb, since eating values for the constituents are for press and temp. For any other press and the value will be the above $\times$ 33.5 $\frac{p}{T}$, e p = press, lb per sq in abs, and T = abs F. At 70° F, the values are 576 and 527.

High value	Low value
7 × 338 = 24	7 × 338 = 24
38 × 341 = 122	38 × 292 = 111
40 × 1 060 = 424	40 × 960 = 384
3 × 1 700 = 51	3 × 1 595 = 48
621	567

Air needed for combustion is most readily computed from the equations representing the reactions. The quantity required in actual equations is that which is chemically necessary. But, in burning [illegible] the possible exception in the case of internal-combustion engines, [illegible] more air must be supplied completely to burn the fuel than is theoretically required. The difference between these amounts is termed the excess [illegible]

Chemical symbols indicate the amounts as well as the substance entering into reaction. Thus the equation, $CH_4 + 2\,O_2 = CO_2 + 2\,H_2O$, may be stated in various ways: (a) CH_4 (methane) combining with O forms CO_2 and water vapor; (b) 16 lb CH_4 combined with 64 lb O forms 44 lb CO_2 and 36 lb water vapor; (c) 1 cu ft CH_4 with 2 cu ft O forms 1 cu ft CO_2 and 2 cu ft water vapor. Thus, not only does a symbol name the substance, but also gives the wt and vol reacting, as each symbol represents 1 unit vol, or the molecular wt of the substance in any units. In equations the wt is taken in lb and the vol in cu ft. The molecular wt of any substance is the sum of the atomic wts of the elements forming it, taken as many times as the symbol indicates. CH_4, for example, being composed of 1 C (atomic wt, 12) and 4 H atoms (atomic wt, 1), has a molecular wt of $12 + (4 \times 1) = 16$, and the symbol CH_4 indicates 16 lb of methane. It should be noted that the molecule of simple gas is composed of 2 atoms, hence we write H_2, O_2, or N_2 to represent hydrogen, oxygen, or nitrogen, and not by H, O, or N. For the atomic weights of the elements, see Sec 37.

As air is approx 21% O by vol, or 23% by wt, the air required for complete combustion of a substance is found by dividing the vol of O required by 0.21, or the wt of O required by 0.23. Table 27 gives the air required per lb or cu ft of different substances.

Table 27. Air Chemically Required for Complete Combustion

Symbol	Substance	1 lb requires	
		cu ft	lb
C_2H_2	Acetylene	165.2	13.3
$C_2H_5(OH)$	Alcohol, ethyl	132.8	9.0
$CH_3(OH)$	" methyl	80.5	6.5
	Anthracite producer gas, air	22.9	1.86
C_6H_6	Benzene	369.7	29.1
	Bituminous producer gas, air	72.6	1.86
	Blast-furnace gas, air	71.6	1.73
C	Carbon to CO	71.4	5.71
	" " CO_2	142.8	11.43
	" anthracite	30.6	2.5
CO	Coal gas, air	137.5	11.1
	Coke-oven gas, air	138.2	10.3
C_2H_4	Ethene	220.2	18.0
C_2H_6	Ethylene	384.2	31.1
	Gasoline	288.3	23.2
C_6H_6	Benzene	288.3	23.2
H_2	Hydrogen	238.1	34.8
	Kerosene	280.3	22.1
	Lignite producer gas, air	76.8	1.8
CH_4	Methane	202.3	17.2
	Natural gas (Ohio)	187.2	15.1
	Oil gas, air	183.7	14.8
S	Sulphur	132.2	4.3
	Water gas, air	68.2	2.8
	" " carbureted, air	100.3	8.0

To find the air required by a mixture of substances, if its analysis is given by volume, add the quantities of air needed for each constituent. To find the wt of air required for solids and liquids, add the wts required for each constituent. To find the vol required for gases and vapors [illegible]

mple 19. Find the air required to combine chemically with 1 lb coal, the ultimate sis of which is given under Example 17:

By chemical equations:

$$H_2 + O = H_2O \qquad C + O_2 = CO_2$$
$$2 + 16 = 18 \qquad 12 + 32 = 44$$

1 lb H_2 requires 8 lb O_2, and 1 lb C requires 32 ÷ 12 lb O_2; therefore, the O required al per lb is

$$\left(0.0247 - \frac{0.0287}{8}\right) \times 8 + 0.8757 \times (32 \div 12) = 2.51 \text{ lb}$$

Se air required is 2.51 ÷ 0.23 = 10.9 lb. Should the vol be required, the wt may ultiplied by the cu ft per lb at the press and temp desired.

By Table 27,

$$\left(0.0247 - \frac{0.0287}{8}\right) \times 34.6 + (0.8757 \times 11.6) = 10.9 \text{ lb}$$

mple 20. Find the air required for chemical combination with a gas, the analysis ich is given under Example 18:

By chemical equations:

$$2H_2 + O = 2H_2O \quad CH_4 + 2O_2 = CO_2 + 2H_2O \quad C_2H_4 + 3O_2 = 2CO_2 + 2H_2O$$
$$0.5 = 1 \quad 1 + 0.5 = 1 \quad 1 + 2 = 1 + 2 \quad 1 + 3 = 2 + 2$$

1 cu ft H_2 or CO requires 0.5 cu ft O_2, 1 cu ft CH_4 requires 2 cu ft O_2, and 1 cu ft requires 3 cu ft O_2; therefore, the gas requires

$$(7 \times 0.5) + (0.38 \times 0.5) + (0.4 \times 2) + (0.03 \times 3) - (0.02 \times 1) = 1.095 \text{ cu ft } O_2,$$

he air needed = 1.095 ÷ 0.21 = 5.22 cu ft. To convert to lb, multiply by the y at the existing press and temp.

By Table 27,

$$(0.07 \times 2.39) + (0.38 \times 2.39) + (0.4 \times 9.59) + (0.03 \times 14.34) - (0.02 \times 4.8) = 5.22 \text{ cu ft.}$$

$$\text{Wt air per lb fuel} = 11.5\,C + 34.6\,H \text{ (approx)} \qquad (50)$$

e above expressions and methods are useful in finding the air chemically red. The actual quantity of air for combustion is found by analysis of lue gas. Then, the air required per lb of C is determined by Eq 51, for containing but little H; when there is much H, the relations are too com- to be of much use.

$$\text{Lb of air per lb } C = (3\,N_2 + 0.165\,O_2) + (CO + CO_2), \qquad (51)$$

N_2, O_2, CO, and CO_2 are percentages by vol by flue-gas analysis.

oducts formed by combustion are chiefly CO_2, water vapor, and the N the air. When excess air is used this is also found in the products. When ficient air is supplied, it is generally impossible to predict the analysis of roducts, because the quantity of the available air received by each com- ble can not be foretold.

mple 21. Compute the products of combustion for the fuels of Example 19 for ase of the necessary and of twice the necessary air. From the equations, 1 lb C s 44 ÷ 12 lb CO_2, and 1 lb H_2 makes 9 lb H_2O; hence, 9 × 0.0247 = 0.222 lb H_2O + 12) × 0.8757 = 3.21 lb CO_2 + 0.0109 lb N_2 from fuel, and 0.77 × 10.9 = 8.4 lb om air. With 100% excess air, the water vapor, CO_2, and N from the coal will be me, and the N from the air twice as great, and there will be 10.9 × 0.23 lb of O the excess air.

ss due to incomplete combustion of C to CO instead of CO_2 is found flue gas analysis by Eq 52:

$$\text{Loss due to unburned CO per lb C} = 10\,193\left(\frac{CO}{CO + CO_2}\right) \qquad (52)$$

Producer reactions. Gas producers produce a combustible gas from fuel, as coke, anthracite coal, and to a less extent from bituminous coa nite, etc. The nearer pure carbon the fuel, the better the result. If the ducer blast be air alone, the reactions are: $C + O = CO$, $C + O_2 = CO_2$, $CO_2 + C = 2\,CO$; if with steam alone, not a continuous process, the rea is $C + H_2O = CO + H_2$; or steam and air may be used together, in which the composition of the gas formed is too complex to discuss here (1).

Final temp and press due to combustion. Combustion may take at either constant press or constant vol, explosively or nonexplosively, in press being characteristic of explosive combustion, but not necessarily fining it. For explosive combustion a mixture of fuel and O, with or wit neutral gases, as N, must be present and intimately mixed, so that a once started proceeds rapidly throughout the mass.

A rise in temp always occurs, but which can only be estimated, because of uncertai of specific heats and other phenomena. Mode of procedure is to apply correction fac found by experience to the hypothetical results computed. Temp rise due to combusti

$$t_2 - t_1 = C\,\frac{Q}{17\,w}, \text{ for constant vol} \qquad (a)$$

$$t_2 - t_1 = C\,\frac{Q}{0.24\,w}, \text{ for constant press} \qquad (b)$$

where t_1 and t_2 are the initial and final temp, Q the heat liberated, w the wt of prod formed, and C a constant. Value of C lies between 0.45 and 0.55 for both furna bustion and gas-engine work, when there is about the chemically correct amount of a but it has higher values for excess air. The vol and press changes due to combustion

$$V_2 - V_1 = \frac{V_1}{T_1}\left(\frac{CQ}{0.24\,w}\right); \quad P_2 - P_1 = \frac{P_1}{T_1}\left(\frac{CQ}{0.17\,w}\right).$$

where V_1, P_1, T_1 are the original vol in cu ft, press in lb per sq ft, and temp in de V_2, P_2, T_2 the final values, Q the heat liberated, C the constant as above, and w t of products. The press rise is more conveniently expressed by

$$\text{Press rise, lb per sq in} = 72.3\,C\left(\frac{\text{initial press, lb per sq in abs}}{\text{initial temp, }^\circ\text{ F abs}}\right) \times \left(\begin{matrix}\text{B t u per cu ft of mixture}\\ \text{at } 32^\circ\text{ F \& } 14.7\text{ lb abs}\end{matrix}\right)$$

Temp of ignition is the temp to which a substance must be heated b combustion takes place. Experiments by Falk and others indicate that temp is not constant.

Falk proved that for gaseous mixtures, it varies with the proportion of gas to air. gives following values, as true over a 100% range of proportions.

Ignition temp of gasolene mixtures with air..... 986° F
" " " alcohol " " " ..... 1 292° F

Beyond certain limits of the air-gas ratio, mixtures will not ignite. Table 28 gives this limit for several gases, expressed in terms of the percentage of detonating mixture which must be present for ignition to occur. By DETONATING MIXTURE is meant the combustible portion of fuel plus the O required for combustion.

Table 28. Percentage of Detonating Mixture in Total Air-gas Mixture at Explosive Limits of Proportion (Bunte)

Gas	For excess air	For excess gas	Gas	For excess air	For ex gas
Carbon monoxide.	24.8	15 8	Ethylene.....	16.4	2
Hydrogen.....	14.2	21.2	Alcohol.....	15.0	21 2
Water gas.....	18.2	21.0	Methane.....	18.3	27 5
Acetylene.....	11.7	14.0	Benzene.....	19 9	22 3
Coal gas.....	17.4	31.2			

e **of combustion** of coal is controlled by the draft. Following rela-) holds for small-size anthracite and bituminous coal.

$$\text{Lb coal per sq ft grate surface per hr} = C\sqrt{\text{Draft, in of } H_2O} \qquad (56)$$

ere C has the following values: Anthracite rice, 17.3; anthracite No 1 heat, 24.5; anthracite pea, 31.5; semi-bituminous, 50.2; bituminous, run- e, 81.6; bituminous, slack, 59.3.

ciency of combustion in furnaces is the ratio of heat developed to that ned in the coal; sometimes termed furnace effic, and ranges from 80 to In producers, the effic is a matter of gasification, and is the ratio of the 1 the gas from 1 lb of coal to the heat originally in the coal.

14. Heat Transfer

ept in the burning of explosive mixtures, heat is practically never devel- n the substance heated. In all other cases, heat must be conducted or nitted to the desired point, generally through separating walls. Flow of ıay be grouped in 3 classes, internal conduction, radiation, and convection.

le 29. B t u Transmitted per Hr per Sq Ft per Deg F. Difference in Temp per In Thickness

ıbstance	K	Substance	K
.....	0.1405	Limestone,	
iinum.....	966 (1+0.0003 (t−32))	marble,calcite,	
nony.....	128 (1−0.00058 (t−32))	dolomite, etc.	13.6–16.2
stos paper.	1.245	Magnesia.....	0.45–1.3
uth.....	51 (1−0.00041(t−32))	Magnesium.....	1 090
ing paper..	0.435	Mercury.....	0.0148–0.0189
; yellow...	592 (1−0.00136 (t−32))	Micaceous flag-	
l.....	713 (1−0.00089 (t−32))	stone	
nium.....	638 (1−0.00039 (t−32))	along cleavage	18.3
on.....	1.32	across "	12.8
noxide.....	0.145	Nickel.....	412
nide.....	0.0891	Paraffine.....	0.66 at 32° to 4.88 at 212°
orundum..	1.45	Pasteboard.....	1.3
or oil.....	1.23	Petroleum.....	1.03
k.....	0.58	Plaster of Paris.	2.03
ntantin.....	1 565–1 855	Platinum.....	483
er.....	2 080 (1−0.00003 (t−32))	Portland cement	2.06
on wool....	0.125	Quartz.....	1.04
ssed.....	0.0957	Sand.....	2.7
t.....	2.08	Sandstone.....	15.8–16.4
tomic earth.	0.377	Sawdust.....	0.348
nite.....	1.1	Silver.....	3 180
.....	0.252	Slate	
brick.....	0.81	along cleavage	16–19
man silver..	203 (1−0.00148(t−32))	across "	9.2–10.4
ss.....	3.2–6.7	Snow, packed..	1.48
nite.....	15.6	Soil.....	0.96 dry, 4.6 wet
rcloth.....	0.122	Steel.....	180–320
.....	6.47–16.48	Strawboard.....	0.871
t.....	507 (1−0.00083 (t−32))	Tin.....	443 (1−0.00038 (t−32))
rt.....	600	Water.....	3.5–6.5
d.....	242 (1−0.00048 (t−32))	Wood	
ther (cow-		cross grain....	0.261
ide).....	1.22	with grain....	0.871
ne.....	0.84	Zinc.....	443

Internal conduction is the passage of heat along a bar of any subs from a hot to a cooler zone. In this case, the characteristics of different m rials are fairly well known.

Values for common substances are given in Table 29, in which K is the B t u mitted per hr through a plate 1 ft square and 1 in thick per deg difference in temp not constant, but varies with the temp in a linear relation; that is, $K_t = K_{32}[1 + a$ 32)]. Where value of a is known, it is given in Table 29; when not given, the s value found must be used regardless of temp. If the temp range be great, the value of the coeff must be used, the mean value of K between 2 points being

$$K_{32}\left(1 + \frac{a}{2}[(t_2 - 32) + (t_1 - 32)]\right);$$

hence the heat flow through a body by internal conduction is

$$Q = \frac{A}{l} K_{32}\left(1 + \frac{a}{2}[(t_2 - 32) + (t_1 - 32)]\right)(t_2 - t_1),$$

where Q = B t u per hr, A = cross-sec of path in sq ft, l = length of path, in, K_{32} = c at 32° F, t_2 = high temp, t_1 = low temp. Temp t_2 and t_1 must be taken in the b itself; not in the fluid surrounding it.

Radiation of heat by a body is the transfer of heat from a hotter to a co body, without warming the intermediate space. Most common exampl the sun's heat, which warms the earth, but not the intervening medium. permeability of different materials to heat waves varies as with light waves. a substance easily passed by one is not necessarily passed by the other. most accepted law for heat radiation is that of Stefan and **Bolzman**; the h radiated from an incandescent body, black when cold, varies as the differe of the fourth powers of the temp, thus:

$$\text{B t u radiated per hr per sq ft} = 16 \times 10^{-13}(T_2^4 - T_1^4),$$

where T_2 = abs temp F of the radiating, and T_1 = that of the receiving, bod

Practically the only black body (that is, one which is 100% efficient in absorbing h rays and reflecting or transmitting none) is carbon, the crystalline form excepted other bodies reflect some heat and fail to radiate to the same extent. As radiatin creases with roughness and darkness, surfaces from which radiation is not desire frequently polished. Table 30 gives relative radiating and reflecting power of differ substances, compared to carbon as unity for radiation and zero for reflecting power course heat rays striking at an angle are partly reflected, and for small angles nearly the heat may be reflected.

Table 30. Radiation and Reflection Coefficients (Kent)

Substance	Radiating or absorbing power	Reflecting power	Substance	Radiating or absorbing power	Reflecting power
Brass cast	0.11	0.89	Ivory, Jet, Marble	0.93–0.98	
Brass polished	0.07	0.93	Mercury	0.23	
Carbon, porous	1.00	0.00	Platinum, polished	0.24	
Carbonate of lead	1.00	0.00	" sheet	0.17	
Cast iron	0.25	0.75	Silver polished	0.03	
Copper, hammered	0.07	0.93	Steel "	0.17	
" varnished	0.14	0.86	Tin	0.15	
Glass	0.90	0.10	Wrought iron	0.23	
Gold, plated	0.05	0.95	Zinc	0.19	
Ice	0.85	0.15			

Convection is the actual conveyance of heat by the movement of partic of a substance. The most important example is the heating of water in vessel

e the heat applied to the bottom is carried up through the liquid by the ties of water, which, on becoming warmer, grow less dense and tend to or rise through the mass being heated. Convection can occur only in ds or gases, substances in which the particles are free to move, and as tion and conduction generally take place at the same time it is difficult late the convection effect.

eat transfer between separated fluids. When heat flow passes from ody to another, a high resistance to flow is encountered, even a carefully- joint being equivalent to several inches of the bar itself. Hence, boiler s are not allowed in contact with the fire, because the high resistance of oint would prevent the carrying away of heat fast enough to prevent the ide from overheating. Between a plate and a fluid a film adheres to the , causing high resistance; by reducing this, agitation of the fluid or high ity increases the rate of heat transfer.

erally, in heat transfer apparatus, there is high resistance to flow on one side or ther of the separating wall, and but little improvement is realized by agitation or r effort on the low-resistance side. In boilers, for example, the high resistance lies en the gases and tubes, and effort spent on the water side avails little. The total ance to heat transfer is therefore made up of the resistance of the materials through the heat must pass, plus the joint resistances, plus the film resistances, of which the s of very few can be determined. Considering a boiler tube, the path is made up of r of soot and ash of unknown thickness and conductivity, a film resistance between ases and this, a joint resistance between the soot or ash and tube, the tube itself film resistance between tube and water. The practical solution of such a problem ys a coeff of heat transmission experimentally determined for various frequently- ring conditions, and including all the separate resistances and radiation and con- n effects. Such coeff, generally designated by U, expresses the B t u transmitted r per sq ft of surface per deg difference in temp, measured in the separated fluids, f which is receiving heat from the other. Values of U in aver practice are in Table The heat transmitted per hr is

$$Q = AUt_m, \tag{59}$$

: Q = B t u per hr, A = area in sq ft, U = coeff, t_m = mean temp difference between uids.

able 31. Coefficients of Heat Transfer U. Averages Used in Practice

Giving heat	Receiving heat		
	Liquid warming	Liquid boiling	Gas warming
uid cooling	50–75	{25–50 / 100*}	2–6
s cooling	2–5	2–5	2–5
or condensing	{150–350 / 1 000*}	400–600	2–4

* High liquid velocity or agitation.

ean temp difference is merely the difference between the hot and cold s, if throughout each the temp be constant. But, in most cases the temp of or both is changing in accordance with which one of the 4 possible cases n below is being considered:

bstance losing heat may be	Substance receiving heat may be·
) at constant temp	at constant temp
) " " "	rising in temp
) falling in temp	" " "
) " " "	at constant temp

Case (*c*) may occur for either parallel or counter-current flow; in the former both fluids enter at same end of the system, the final temps approaching the same value; in the latter, the fluids enter at opposite ends, final temp of one approaching initial of other. For cases where either or both fluids are changing in temp

$$\text{Mean temp difference} = \frac{\text{Initial temp difference} - \text{final temp difference}}{\text{Nap log}\left(\dfrac{\text{Initial temp difference}}{\text{Final temp difference}}\right)}$$

Example 22. A solution the specific heat of which is 1.2 is to be heated to 170° in a heater by 60° by exhaust steam at atmos press and 90% dry. Find the surface required to heat 50 000 lb per hr, also the steam needed, neglecting any heating by condensed steam.

$$\text{By Eq 60, } t_m = \frac{(212 - 60) - (212 - 170)}{\text{Nap log}\left(\dfrac{212 - 60}{212 - 170}\right)} = 86$$

$$Q = 50\,000 \times 1.2 \times 110 = 6\,600\,000 \text{ B t u}$$

$$\text{Heat per lb steam} = 970 \times 0.9 = 873, \text{ and lb steam} = \frac{6\,600\,000}{873} = 7\,550$$

$$\text{From Table 31, } U = 150 \text{ to } 350, \text{ say } 300, \text{ and from Eq 59, } A = \frac{6\,600\,000}{300 \times 86} = 256$$

Example 23. The same quantity of the same liquid as in Example 22 is to be used as it cools to heat boiler-feed water, by passing the liquids in a counter-current direction through concentric pipes. 10 000 lb of water per hr are to be heated from 50° to 160°. Find required heating surface; also the required surface should it be desired to raise the final temperature of the feed water to 160°. The heat required to be transmitted will be $10\,000 \times 100 = 1\,000\,000$ B t u, hence fall in temp of solution will be $1\,000\,000 \div (50\,000 \times 1.2) = 16.7°$, and final temp will be 153.3°

$$\text{By Eq 60, } t_m = \frac{(153.3 - 50) - (170 - 150)}{\text{Nap log}\left(\dfrac{153.3 - 50}{170 - 150}\right)} = 50.5$$

From Table 31, assume $U = 60$, whence from Eq 59, $A = 1\,000\,000 \div (60 \times 50.5) = 333$ sq ft. For second case, $Q = 1\,100\,000$; final temp of sol = 151.6; $t_m = 52.1$; $A = 464$; showing the large increase in area needed for increased final temp of the cold liquid as it approaches the initial temp of the hot.

Capacity of boilers and other heat-transfer apparatus is usually expressed in BOILER HORSE POWER (b h p). A b h p was originally assumed to be such that a boiler of a certain hp would supply enough steam to run a steam engine of the same hp. The term became too indefinite, since engines differ widely in consumption of steam for 1 hp, and the following definitions were adopted by the Am Soc of Mech Engs.

1 b h p = 34.5 lb steam made at standard atmos press, or temp of 212° F per hr from feed water at 212° F, or 33 305 B t u absorbed per hr.

Originally these terms were equivalent, but later investigations show that if the first definition is true the second should be 33 479 B t u. The first is generally used for ordinary feed-water temp and steam press, 34.5 lb from and at 212°, as it is ter- not differ greatly from 30 lb under common conditions, and this being an even is generally used as the h p unit for rating boilers, feed-water heaters, condensers. That is, a feed-water heater of 100 b h p capacity will heat $30 \times 100 = 3\,000$ per hr. Boilers require from 10 to 12 sq ft of heating surface per hp, depending on and are frequently rated by dividing the sq ft of heating surface by these numbers.

Efficiency of a boiler is the ratio of the heat absorbed by the water steam to that in the coal and may be separated into 2 parts:

$$\left.\begin{array}{ll} (a) \text{ Furnace effic} & = \text{heat liberated} \div \text{heat in coal} \\ (b) \text{ Surface effic} & = \text{heat absorbed} \div \text{heat liberated} \\ (c) \text{ Total boiler effic} & = a \times b = \text{heat absorbed} \div \text{heat in coal} \end{array}\right\} \quad (65)$$

ler effic defined above is sometimes designated as boiler and furnace effic. Among ses tending to reduce furnace effic are coal lost to ash pit, carbon burning to CO rd of CO_2, H, and hydrocarbons passing out of stack unburned; that tending to : surface effic is the sensible heat of the stack gases above the room temp. As gases can not be cooled below steam temp, the heat required to warm them from temp to steam temp is often called the necessary loss. For description of a com- heat balance, defining each loss and a manner of determination, see (18).

15. Entropy and Entropy Diagrams

finitions. Just as work may be expressed as the product of press and hange, when these quantities are expressed in proper units, so may heat pressed as the product of temp and the change in a quantity termed en- . One of the best definitions of entropy is that it is "heat per degree of rature." In engineering calculations it is the CHANGE IN ENTROPY dur- process, rather than the ABSOLUTE ENTROPY at any one time, which is tant; hence any arbitrary zero of entropy may be chosen.

steam work the zero generally taken is that of 1 lb of at 32° F, and in steam tables (Art 12) the values given ove that value as a base. For ordinary work on gases int in the process having the least entropy is selected zero. Entropy is usually designated by the Greek ϕ, and a diagram drawn to temperature-entropy coor- s is the $T\phi$ or heat diagram, precisely as one drawn to ordinates is the P-V or work diagram. The area rep- ing the amount of heat change during a process is the der the curve down to line of zero temp, this quantity expressed to the absolute scale. Thus, in Fig 23, the hange from A to B is represented by area $ABCD$.

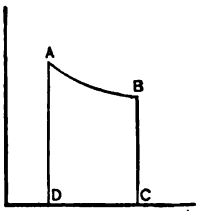
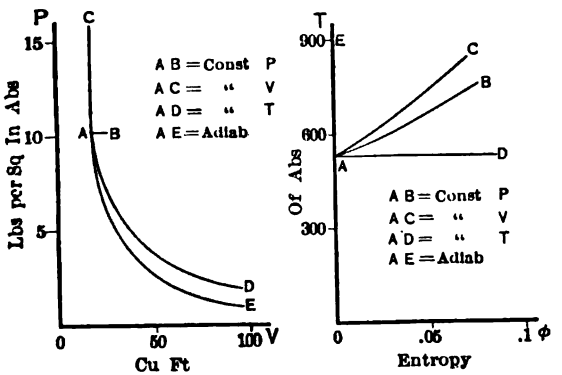


Fig 23

ange in entropy, being the change in heat divided by the temp at which urs, is simply expressed if the change takes place at constant temp. If mp is also changing, a differential form is required, for which a relation



g 24. Constant Press, Vol, Temp, and Adiabatic Lines, for Air to P-V and T-ϕ Coordinates

:en heat and temp is necessary. This is found in the quantity known as fic heat, and the change in entropy is found by integrating. The results iven in Eq 62 for constant specific heats, the expressions resulting from ng specific heats being too complex for general use.

$$\left.\begin{aligned}
&\phi_2 - \phi_1 = w\,Q \div T, \text{ for constant temp}\\
&\phi_2 - \phi_1 = w \text{ Nap log } (T_2 \div T_1), \text{ for water (sp ht = 1)}\\
&\phi_2 - \phi_1 = wC_v \text{ Nap log } (T_2 \div T_1), \text{ for gases at constant vol}\\
&\phi_2 - \phi_1 = wCp \text{ Nap log } (T_2 \div T_1) \quad " \quad " \quad " \quad " \quad \text{press}
\end{aligned}\right\} \quad (1)$$

For superheated vapors the values for gases may be used with fair accura especially for high degrees of superheat.

The four commonest processes, isothermal and adiabatic expansion, cons press-vol change, and constant vol-press change for gases, are shown in F to P-V and T-ϕ coordinates.

Temperature-entropy diagram. For vapors 2 entropy diagrams used, the temp-entropy and the B t u-entropy or Mollier diagram. Fig shows on a small scale the temp-entr diagram for steam.

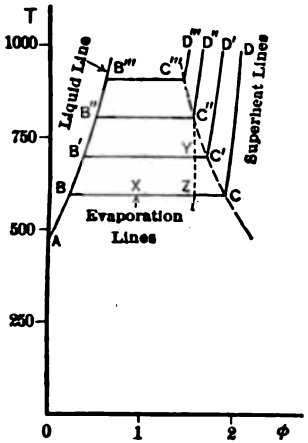


Fig 25. T-ϕ Diagram for Steam

The liquid line ABB''' represents the he of water from 32° F, the area below it deno the heat required, or the heat of the liq any point above 32° F. Lines BC, $B'C'$ represent steam-making at constant tem area under any horiz line denoting the la heat at that temp. The third group CD etc, represents superheating at constant Lines of constant vol, constant quality or stant heat may also be added (omitted fr diagram for clearness). Adiabatic expans compression is represented by a vertical l of the chief uses of the diagram is to deter the quality after an adiabatic change. length of any of the horizontals BC is eq the latent heat divided by the temp. Sho steam be wet all of the latent heat will be used; hence the line will be shorter and move to the left to some indefinite posit Then as BX is proportional to the quanti latent heat used, and BC to all the latent for the same press, $BX \div BC$ is the percent of steam present, or the dryness fractio $XC \div BC$ the percentage of moisture present. The quality after any adiabatic can be quickly determined by projecting from the original point up or down on the gram. For example, if adiabatic expansion occurs from point C (representing dry at some press), when the press drops to Y the quality will be $B'Y \div B'C'$, and w pressure Z is reached, the quality will be $BZ \div BC$. The diagram shows that ad expansion of dry steam causes wetness, expansion of superheated steam cause superheat, and expansion of very wet steam causes drying. Compression under t conditions causes the reverse action. For more complete description see (1).

Mollier diagram for steam, drawn to as large a scale as possible, is shown Fig 26. This, named from its originator, Prof Mollier, is of much w than the temp-entropy diagram. On it, horizontals are lines of tota verticals, lines of constant entropy; lines nearly horiz but sloping toward lower right-hand corner represent constant quality, and lines making an an of more than 90° with them are lines of constant press. By projecting to side from the intersection of a quality and press line the total heat may be directly, or the entropy by projecting downward. Adiabatic expans compression is represented by a vertical line, constant total heat by a hori

The greatest use of the diagram is in solving Rankine-cycle problems; Ex for effic and for velocity of steam jets. To find the work done in a Rank cycle it is merely necessary to read the total heat corresponding to any g press and quality, project from the original point downward to the line rep

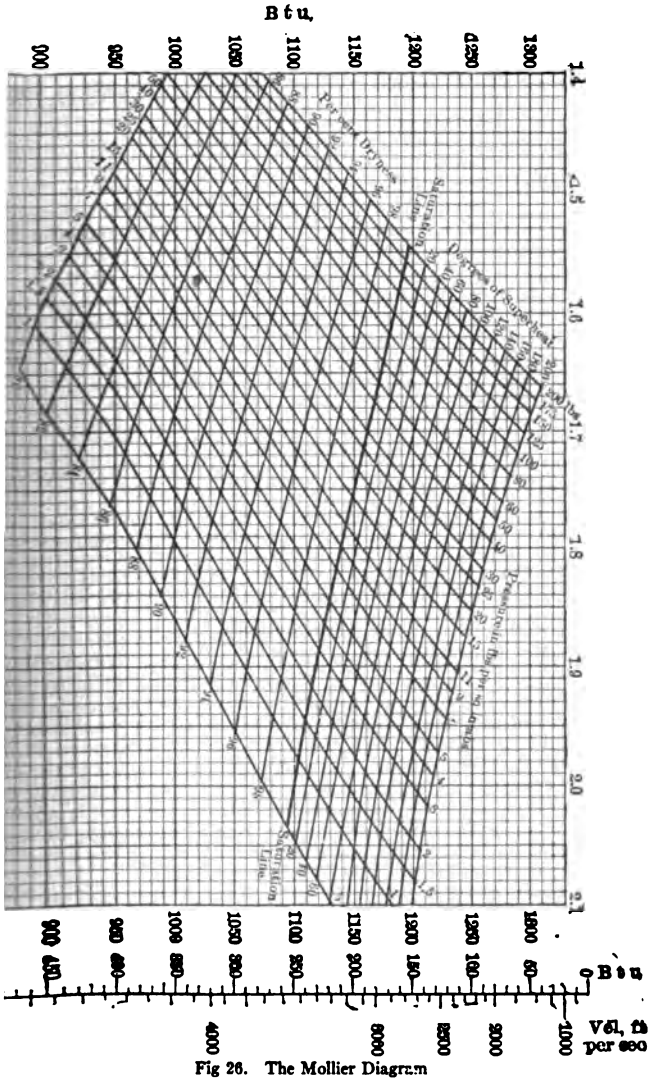


Fig 26. The Mollier Diagram

g final press, read the total heat at this point and subtract the two. The quality needed for determining m e p is found from the nearest quality ntersecting the final press line.

As explained in Ar
and as the work is
an extra scale givin

Example 24. Fin
diagram.

Remaining quantiti
method of solution w

The Mollier diag
throttling calorim
press (generally at
expanded, and sub
total-heat expansio
press, the amount
no superheat at th
applicable. When
may be found on
heat) to the high
the horiz and high-

Example 25. In
the press was 110 lb
situation, sea level.
and saturation temp
Mollier diagram hori

Definitions. W
ess some heat may
expense of the hea
in this heat conte
or contained in a
above any arbitra
in intrinsic energy
one time, which is
difference between
energy, and this
namics. It applie
For a complete
as at the beginnin
from the heat ad
abstracted as such
added heat must b
similarly for the w

Work done in

and as the effic of

Effic of any

Steam-engine
cribed on the

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sis of heat-cycle analysis, and expressions for effic and m e p derived in this
For the phases of constant-press, admission and exhaust of the P-V diagram
bstituted the equivalent constant press-vol change processes of steam mak-
rith possibly superheating, and
nsation on the heat diagram.
o standard cycles represent the
steam engine process, the Carnot
e Rankine. Both are shown to
nd T-ϕ coordinates; the former
steam only in Fig 27, the latter
t and dry steam, and steam with
nt degrees of superheat, in Fig
follows:

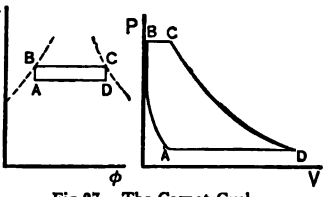


Fig 27. The Carnot Cycle

$ABCD$,	initially wet, finally more wet			
$ABC'D'$,	"	dry,	"	"
$ABC''D''$,	"	superheated, finally wet		
$ABC'''D'''$,	"	"	"	dry
$ADC^{IV}D^{IV}$,	"	"	"	superheated

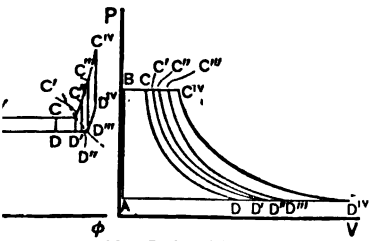
not cycle, which assumes complete adiabatic expansion and compression, rmerly universally accepted as standard, chiefly because it was the most it cycle between any 2 temperatures, and hence for steam pressure limits. ic depends only on the temp, or,

$$\text{Carnot-cycle effic} = (T_2 - T_1) \div T_2 \qquad (65)$$

T_2 and T_1 are the high and low abs temp, respectively.

ıkine cycle assumes complete adiabatic expansion, but no compression, now widely used as a standard of comparison, chiefly because it repre-an effic more nearly attainable, is equally applicable to superheated and steam, and serves as well for a basis for turbine work as for engine work, es not possessed by the Carnot. A common procedure is to speak of s as attaining a certain fraction of the Rankine cycle possibility.

ig 28 the constant-vol, press-rise line of the P-V diagram is represented on the liagram by a liquid heating line, which is substantially a constant-vol process. Heat is added from A to B and then from B to C, C', etc, as the case may be, adiabatic expansion follows to D, D', etc, and finally heat is removed from D, D', etc, to A. Considering one case only, that of initially dry steam, for example (as the rest are similar), the heat added is $H_{C'} - H_A$, that removed $H_{D'} - H_A$, and the work done is $(H_{C'} - H_A) - (H_{D'} - H_A)$ or $H_{C'} - H_{D'}$. Thus, the work of the Rankine cycle is merely the difference in the total heats at beginning and end of expansion. That at beginning is readily found



3. **The Rankine Cycle, with Different Initial Qualities**

e **steam tables** (Art 12), when the initial press and quality are known, or from llier **Diagram** (Fig 26). The final quality is not known, but can be computed or adily obtained directly from Fig 28. For method of computation, see Example 26. l be $(H_{C'} - H_{D'}) \div (H_{C'} - H_A)$, H_A being the heat in the liquid at beginning of the r at the temp corresponding to the back press, and may be found from the steam The m e p is the work done divided by the vol change, and since originally there is uid present, the vol of which may be neglected as it is very small compared to the

final vol, the vol change may be taken as equal to the final vol. tabular value (from steam table) times the quality, if wet, and a divided by the saturation temp, both absolute, if superheated. equally to turbines, and all of the above except the expression for to both piston engines and turbines. But, there is one more quanti bines only, viz, the velocity of the jet. Considering 1 lb of steam and ity, the final velocity will be $\sqrt{2\,gW}$ or $8.02\sqrt{H_{2'} - H_{2}}$. These ex in Table 32.

Table 32. Rankine Cycle

Work	$H_1 - H_2$
Efficiency	$(H_1 - H_2$
M e p (lb per sq in)	$778\,(H_1 -$
Heat consumption per hr per h p	$2\,545\,(H_1 -$
Steam consumption per hr per h p	$2\,545$
Jet velocity	$\sqrt{(H_1 - H_2}$

H_1 is the heat in the liquid at the low press; H_2, the heat in the st sion; H_3 the heat in the steam after expansion.

The thermal effects of change of initial press, back press, superheating and reheating, in a steam engine are best seen from examination of Rankine cycle. In Fig 29 (a), $ABCD$ represents a cycle between t

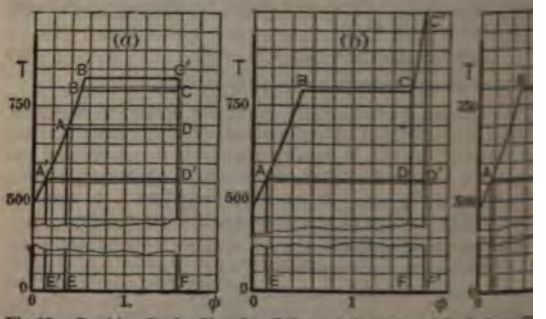


Fig 29. Rankine Cycles Showing Effect of Initial and Back-Press C Reheat, and Jacketing

lb and 20 lb, the area being proportional to the work of 1 lb of initially d shows the case for an initial press of 160 lb, all other conditions being in work being $BB'C'C$. $A'BAD'$ is the case of the original initial pr press lowered to 1 lb, the gain in work being $A'ADD'$. Thus, the gai is considerably greater than for the former, although the change in ciencies in the 3 cases are: $ABCD \div EABCF$, $AB'C'D \div EA'B'C$ $E'A'BCF$. Increase of initial or decrease of exhaust pressure both t efficiency. Complete expansion is assumed in all cases of the Rankine possible to realize in real engines with high ratios of expansion, that high initial or very low back press, so that the gains in cyclic perfe fully attained in practice or may possibly be losses.

In Fig 29 (b), $ABCD$ represents the work of a cycle for saturated st press limits, and $ABCC'D'$ for steam with 300° of superheat and th the area $DCC'D'$ denoting the gain in work. Efficiencies for the s $EABCF$ and $ABCC'D' \div EABCC'F'$. It is obvious that superhe thermally with but little increase in efficiency. In pra ages and heat transmission losses generally produce a be expected from the cyclic effect.

cle $ABCD$ is for saturated steam with adiabatic expansion, and $ABCD''$ ith jackets imparting heat enough to the steam to cause it to remain xpansion. Other cases, in which insufficient or more than sufficient g expansion to produce continued saturation, are possible, but ideally hermal efficiency and in practice either a loss or such a small gain as the use of jackets inadvisable. Cycle $ABCxyzD'$ represents the case rtial expansion, a process, if used at all, occurring between the cylinders sion engine, or between an engine and a low-press turbine. Point X, is allowed to proceed, is arbitrary, as is also the reheating temp. A to make the receiver pressure X such that equal work is done in eheating can not well be carried above the original temp. As with e to reheating is either negative or too small to warrant its use on s for the 3 cycles in Fig 29(c) are $ABCD \div EABCDF$ for the original, F'' for jackets, and $ABCXYZD' \div EABCXYZF'$ for reheating.

nd the cyclic effic, water, and steam consumption of an engine oper- of 140 lb abs, with 100° superheat and back press of 10 lb abs. Also for same press drop, and the m e p.

ables (Art 12) the heat at beginning of expansion is 1 248 B t u, and e liquid at 10 lb abs (= 193° F) is 161 B t u. The quality after ex- follows. The entropy originally (Table 25) is 1.6395, and since ex- the final entropy must have same value. The total entropy of steam ts if wet, or 3 if superheated, precisely as is the total heat. If for y is less than the dry saturated value for that press, the steam is wet opy must be part of that of evaporation; the percentage of moisture ssing entropy divided by the entropy of evaporation for the given dry steam at 10 lb is 1.7874, hence the amount missing is 1.7874 − As the entropy of evaporation for this press is 1.5042, the percentage r dryness is approx 90; hence, the total heat is 1 045 B t u. The final ead directly from a temp-entropy or a Mollier diagram (Art 15).

e effic will be (1 418 − 1 045) ÷ (1 240 − 161) = 18.7%. Steam con- 2 545 ÷ (1 248 − 1 045) = 12.5 lb, and jet velocity

$$= \sqrt{(1\,248 - 1\,045) \times 778 \times 64.4} = 3\,190.$$

g 90%, the final vol will be 90% of the tabular, or 0.9 × 38.4 = 34.6 p = 778 (1 248 − 1 045) ÷ 144 × 34.6 = 37 lb.

uld be obtained with practically no calculation by the Mollier diagram, s worked by that method in Art 15.

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SECTION 40

STEAM ENGINES, BOILERS, PUMP WATER TURBINES, GAS ENGINES

BY

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DEPARTMENT OF MECHANICAL ENGINEERING, COLUMBIA UNIVERSITY

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STEAM ENGINES

1. Classifications and Characteristics

sifications. I. SIMPLE ENGINES: (*a*) high-speed, single valve; (*b*) high-speed, alve; (*c*) medium and low-speed, multivalve. II. COMPOUND ENGINES, with or t receiver: (*a*) tandem; (*b*) cross-compound; (*c*) duplex. Each may be high, n, or low speed, and they are generally multivalve. SPEED CLASSIFICATION is based itive speed, not piston speed; 175 to 500 r p m being classed as high speed, 110 to) m, medium speed, and less than 110 r p m, low speed. PISTON SPEED is the r of linear ft piston travel per min. In different types of engine, it ranges from 750 ft per min, short-stroke engines having the lower speed, long-stroke, the higher. am engines are double acting, excepting a few special types, and all are built in the horiz or vert form (1, 3, 4, 5).

gh-speed, single-valve, simple engines are made in sizes up to 500 hp. are low in first cost, but have rather high steam consumption. Simple en- are rated at some definite steam press and approx 0.25 cutoff (about the economical point for non-condensing). For simple, condensing engines, the conomy is at ⅛ to 0.2 cutoff, at which they carry a 25% overload without y increasing the WATER RATE (lb dry steam per ihp-hr). Water rate in- s rapidly as the load falls below 50% of rating.

t engines of this type have automatic cutoff governor; sometimes a throttle gov- A throttle governor reduces initial cost, but cannot give close speed regulation, wasteful of steam when the engine runs at loads much below its max power.

gh-speed, multivalve, simple engines, because of the greater number of have higher first cost than single-valve engines and require more care, but regulate more closely and their steam consumption is lower. They are in sizes up to 1 000 hp.

edium and low-speed, multivalve, simple engines are made in sizes up to hp. Corliss or trip valve-gear is practically confined to low-speed engines, he 4-valve gear may be used for any speed. Unit steam consumption of ium and low-speed simple engines is lower than that of high-speed, single- , simple engines of same power and operating under same conditions, but is little difference in this respect between them and the high-speed multi- engine.

mpound engine, theoretically, has no better efficiency than a simple engine. ractical advantages are: lower leakage and condensation losses, higher ratio pansion, and more uniform crank effort. Initial and repair costs are higher, he space occupied per hp is greater. They are made in sizes up to 6 000 hp etimes larger). Diams of high and low-press cyls are proportioned to lize the work approx between the two, when the engine is run at its rated

To do this the ratio of piston areas depends upon the ratio of expansion een initial steam press and back press, ranging from about 1 : 3 to 1 : 6.

large compound engines the low-press cyl is sometimes divided into 2 cyls, to avoid ifficulty of making very large cyls and the still greater difficulty of properly balancing reciprocating masses. In order that the low-press cyl may not be unduly large respect to the rest of the engine, and hence increase cost of manufacture, steam is r expanded below 6 lb per sq in, when the engine is operating at its rated load. pting some locomotive engines and a few smaller types of stationary engines, com- d engines are designed to operate with a receiver between high and low-press cyl.

iple and quadruple-expansion engines are seldom built except for marine use. e some of the best thermal effics on record have been made by them (generally in ng large pumps), their very high first cost has caused them to be supplanted by n turbines.

2. Sizes, Powers, and Speeds

Water-rate, in its generally accepted sense, means lb of st per ihp-hr. Table 1 gives the averages to be expected fro

Table 1. Average Water Rates of Steam Engi

Type of engine	Water r lb. per h
High-speed, simple, single-valve, non-condensing	3
" " " " condensing	2
" " " multivalve, non-condensing	2
Low-speed, " " "	2
" " " condensing	1
High-speed, compound, non-condensing	2
" " " condensing	2
Low-speed, " non-condensing	2
" " " condensing	1

operated at rated load under normal steam press with saturat superheated steam reduces the water rate by 5 to 20% for each heat. Reduction in coal consumption will be less, as some of th must be used for superheating the steam, unless waste heat is u liarities in load conditions also cause water rates to vary.

Cost of steam engines varies widely with size, make, speed, gearing. The following aver costs (*Engineering*, Nov 15, 190 for comparison:

	Cost (dollars)		
Simple high-speed engines.	"	"	30 +
Compound high-speed engines.	"	"	1 20 +
Simple low-speed engines.	"	"	1 20 +
Compound low-speed engines.	"	"	2 00 +
Setting, high-speed engines.	"	"	60 +
Setting, low-speed engines.	"	"	300 +

Following tables are abridged from builders' catalogues. Each en operated satisfactorily at steam pressures and speeds other than those sponding variation in hp. Builders' catalogues should be consulted for types other than those listed.

Table 2. Horsepower of Acme Engines (Stirling Mach

Cylinder		Steam press. 80 lb.; mean effec press. 50 lb. for r p m stated				Steam press. 100 lb.; mean effec press. 66 lb. for r p m stated				Steam press. 125 lb.; mean effec press. 86 lb. for r p m stated
Diam. in	Stroke, in	150	225	300	400	150	225	300	400	150
2½	3½	0.47	0.712	0.95	1.26	0.62	0.94	1.25	1.67	0.80
3	3½	0.93	1.4	1.87	2.5	1.22	1.86	2.48	3.29	1.62
3½	3½	1.24	1.71	2.29	3.4	1.51	2.26	3.2	4.02	1.98
3½	5	1.63	2.45	3.26	4.35	2.15	3.24	4.3	5.75	2.83
4	5	2.38	3.57	4.76	6.35	3.14	4.71	6.28	8.37	4.13
4½	5	3.1	4.65	6.17	8.3	4.08	6.14	8.17	10.9	5.36
5	5	3.71	5.58	7.42	9.88	4.91	7.3	9.84	13.1	6.44
5	7	5.2	7.8	10.4	13.9	6.8	10.3	13.7	18.3	9.0
6	7	7.5	11.2	15.05	20.0	9.9	14.8	19.75	26.4	12.95
7	7	10.2	15.3	20.4	27.2	13.5	20.2	27.0	36.0	17.7

Point of cutoff, 60% stroke. Any of above sizes may be operated u

ble 3. Horsepower of Single-cylinder, High-speed Engines (McIntosh, Seymour and Co) (Non-condensing, cutting off steam at 0.25 of stroke)

of cyl, in	Rev per min	Initial steam press, lb		Size of cyl, in	Rev per min	Initial steam press, lb	
		80	100			80	100
×9	300	21.5	27.8	14½×15	210	108.8	139.6
	360	25.8	33.4		255	132.0	169.3
×9	300	28.0	36.2	16×15	210	131.6	169.9
	360	33.6	43.5		255	159.8	206.4
×9	300	35.6	46.0	17½×15	210	156.8	202.7
	360	42.7	55.2		255	190.3	246.0
×10½	270	41.4	53.5	18½×17	190	180.9	233.5
	330	50.6	65.4		235	223.9	289.0
×10½	270	50.8	65.6	18½×18	190	191.5	247.2
	330	62.1	80.2		235	237.0	306.0
×12	240	56.7	73.3	23×17	180	256.8	331.7
	300	70.8	91.6		210	322.4	416.6
×12	240	73.3	94.7	26×17	185	345.6	446.8
	300	91.4	118.2		205	382.9	495.0
×15	220	90.9	117.4	29×19	165	431.3	557.0
	265	109.5	141.4		185	484.2	625.9

Table 4. Horsepower of High-speed Compound Engines (McIntosh, Seymour & Co)

Type A. 5/16 cutoff in high-press cyl			Type B. 5/16 cutoff in high-press cyl			Type C. 5/16 cutoff in high-press cyl		
Size of engine	Rev per min	Init'l steam press	Size of engine	Rev per min	Init'l steam press	Size of engine	Rev per min	Init'l steam press
		100			80			120
8& 14½×10½	275	69.9	9½ & 14½×10½	275	74.7	9& 16×12	260	113.8
9& 16×12	260	94.8	10½ & 16×12	260	98.0	11& 19×15	230	181.6
0& 17½×12	260	113.6	12& 17½×12	260	117.4			150
11& 19×15	230	152.2	13& 19×15	230	157.1	13& 23×17	200	320.9
13& 23×17	210	224.7	15& 23×17	210	232.2	15& 26×17	195	408.5
		120			100	16½& 29×19	175	508.2
15& 26×17	200	346.2	16½ & 26×17	200	341.6	Types B & C are preferable for varying load, or in cases where, from unreliability of cond'g water supply or other causes, ability to maintain a vacuum is uncertain.		
6½& 29×17	195	406.2	18& 29×17	195	389.5			
18& 32×19	175	486.0	20& 32×19	175	478.5			

Type D. 7/16 cutoff in high-press cyl			Type E. 3/8 cutoff in high-press cyl			Type F. 7/16 cutoff in high-press cyl		
		100			130			120
½& 14½×10½	275	71.1	8& 14½×10½	275	73.5	10½& 16×12	260	115.4
0½& 16×12	260	93.2	9& 16×12	260	99.8	13& 19×15	230	187.8
2& 17½×12	260	111.7	10& 17½×12	260	119.3			140
13& 19×15	230	148.2	11& 19×15	230	164.5	15& 23×17	200	322.6
15& 23×17	210	220.0	13& 23×17	210	243.4	16½& 26×17	195	384.1
			15& 26×17	200	309.7			150
		120	16½ & 29×17	195	367.3	18& 29×19	175	491.5
16½& 26×17	200	328.3			140	Types F & D are preferable for varying load. Engines of Type D, under load less than 5/8; or Type F, less than 60%; or Type F, less than 40% rated load, will not effect proport'n'l saving of fuel, as compared with larger load.		
18& 29×17	195	395.1						
20& 32×19	175	486.0	18& 32×19	175	486.0			

Table 5. Net Horsepowers of Compound Engines (B. F. Sturtevant Co)

Size			Wt, lb	Rev per min		Steam press, lb per sq in							
Diam h-p cyl	Diam l-p cyl	Stroke				100	110	120	125	130	140	150	160
6¼	10½	6¼	3 500	450	Non-con	22.4	24.9	27.4	29.0				
					Cond'g	28.8	30.7	32.5	33.9	Maximum press.			
7	12	7	4 600	400	Non-con	27.7	31.7	35.7	37.7	125 lb per sq in			
					Cond'g	37.5	39.9	42.4	44.2				
8	14	8	10 500	400	Non-con	42.9	49.1	55.4	58.6	61.8	68.3	74.7	81.0
					Cond'g	58.5	62.1	65.7	68.5	71.3	76.7	82.2	87.7
10	18	10	14 500	350	Non-con	77.9	89.2	100.2	106.1	112.2	124.2	135.4	146.5
					Cond'g	105.4	112.2	119.0	124.1	129.2	139.5	149.1	158.7

Note. — Horsepowers are based on cutoff at ⅓ stroke, and, when condensing, on vac of 25 in. Engines should not be operated at pressures exceeding those for which pow are given. The 8 & 14 × 8 engine as listed is built for press up to 180 lb; a lighter de is built for a limit of 125 lb.

3. Valves and Valve Gearing

Single slide valves, driven directly by eccentric, are used in low-power, high-sp engines, and in the cheaper forms of medium-power, high-speed engines. The s plicity of this valve gear and its non-liability to derangement, make it especially u ful for small mine hoists. DIAGRAM FACTOR (Sec 39, Art 4) of simple slide-valve engi ranges from 50% in smallest sizes to 85% in the largest. Distortion of the diagram particularly noticeable at very short cutoffs. Corliss valves are limited to a speed of greater than 125 rev per min. Their diagram factor ranges from 85 to 90%. They advantageous for intermittent-running engines, such as hoists, in that they may be thro by hand, for admitting steam to the cyl even if the piston has passed the normal point cutoff. For the better class of high-speed engines of medium and high powers, valv are used (as in the Corliss gear), but are fixed rigidly to their gearing and not trip like the Corliss. Diagram factors for ordinary 4-valve engines (those having separ admission and exhaust valves at each end of the cylinder) average slightly lower th for the Corliss. Double slide valves, driven by separate eccentrics, are used on some typ of engines, with results similar to those of the 4-valve gear. SETTING ENGINE VAL For details see Bib (1, 3, 4), also Kent's Mech Engs Pocketbook, Ed 1913, pp 103 1047.

In Europe, poppet valves have long been used on medium and high-power engin particularly with superheated steam. They are actuated by a cam shaft, geared to th main shaft. Their diagram factors range as high if not higher than those of Corliss valv A few poppet-valve engines are now being made in this country, and it seems probab that their use will be extended.

4. Special Types

Rotary engines form the largest class of steam engines other than those already de scribed. To 1915 none has been produced which can compare in steam economy wi reciprocating engines of same power and operating under same steam conditions. Mo of them are wasteful of steam even when new and become rapidly worse with wear. Th only valid claims made for them are high speed and lack of vibration. These feature are also true of the steam turbine (Art 5–7), which is now made in small sizes with steam consumption approaching that of the best reciprocating engines, and in large sizes ha displaced the reciprocating engine for all except special purposes.

"Unaflow," or straight-flow, engine derives its name from the peculiarity of th steam cycle. The inlet valve, generally of the poppet type, admits steam in the usual manner at beginning of stroke, but the exhaust port, which has no valve, is uncovered by the piston at end of stroke, and all of the steam remaining in the cyl when the exhaust port is covered by the returning piston is compressed into the clearance. Hence the steam flows through the cyl in one direction. These engines cost more than simple

es of same power, but their steam consumption is as good as that of high-grade ound engines. Also, their steam consumption per hp-hr varies but slightly over a range of load.

STEAM TURBINES

5. Principles and Operation (9, 10, 11, 12)

finition. A steam turbine utilizes the expansive energy of steam in its kinetic as distinguished from reciprocating or rotary engines, in which the steam acts by essure directly upon a piston.

al steam turbine receives steam at line press, utilizes all the expansive energy elerating flow of the steam, and discharges it at the exhaust press at zero velocity, ut loss by friction, impact, or radiation. This result is attainable only by trans- g all the momentum of the steam to moving parts. The chief difficulty in design- machine to approach the ideal is the economical utilization of the high velocities o expanding steam; thus, expansion of steam between the ordinary limits of line xhaust press will give a steam velocity of 3 000 to 4 000 ft per sec. The number igns on the market indicate the numerous attempts made to overcome this difficulty.

thod of operation is shown by Fig 1. *BMC* is the cross-section of a turbine vane. ner surface of which turns through 180°. *AB* represents in length the veloc of a jet. If the vane is at rest and there is no on loss, the jet will be turned in the opposite tion and leave the vane with veloc *CD*, equal 3. In this case no work is done, since there is ange of momentum, as would also be the case vane were moving at same veloc as the jet. if the vane moves at veloc *III*, the relative of the steam to the vane will be $AE = AB$ *I*. Relative veloc of the steam leaving the is $CF = AE$. But, since the vane has an absolute veloc *IH*, the absolute residual of the steam equals *FG* and is in the direction of the motion of the vane. Hence, vane be moving at half the veloc of the jet, the steam will be brought to rest on leav- he vane. But, as there must be a series of vanes in a practical turbine, it is impos- sible to turn the steam through 180° in the same plane without mutual interference.

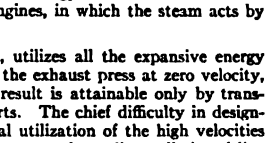
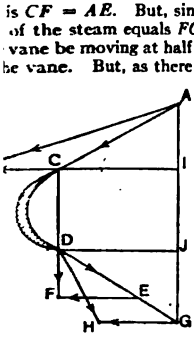


Fig 1. Steam Turbine, Velocity Relations



2. Operation of Steam Turbine

Fig 2 shows the principle of a practical turbine. Line *AB* represents veloc and direction of the steam jet, *CB* the veloc and direction of the vane. Then *AC* is the relative veloc and direction of jet to vane. If there be no friction loss, *DG* represents the relative exit veloc of the jet, and this, solved with *GH* (= *CB*) the veloc of the vane, shows the residual absolute veloc of the jet to be *DH*. But, due to fric-

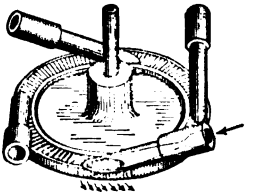


Fig 3. DeLaval Turbine

loss, the exit relative veloc will be represented by some length of line, as *DE*, which n with the veloc of the vane ($CB = GH$), shows the residual absolute veloc of the be *DE*.

avoid impact losses the jet should enter the vane tangent to its face; in this case ntrance angle is equal to the angle *ACI*. Thus, for best conditions, the angle of , veloc of vane, and veloc of jet are interdependent. In Fig 2 the exit angle of vane ual to entrance angle; a common construction, though by no means necessary. 3 shows a diagram of the DeLaval turbine, with casing removed; it is the simplest of impulse turbine.

Impulse and reaction turbines. In the impulse type all the veloc, and hence kinetic energy, is generated outside of the moving parts; in the reaction type, with the moving parts. No commercial turbines operate solely on the reaction principle but those using a combination of both principles are commonly known as reaction turbines for example, the true Parsons design.

Commercial types. The vane veloc must be very high if the entire energy of expanding steam is to be utilized in a single wheel. The great variety of commercial designs results from efforts to produce an economical turbine with a shaft speed low enough for the direct drive of ordinary machinery. The chief modes of meeting this problem are: (*a*) A single impulse wheel of large enough diam to bring the shaft speed and linear speed of vanes within the required limits (not used in any American turbines). (*b*) A single wheel of small diam, connected to the main shaft by gearing. Fig 4 *A* shows the principle of the simple impulse turbine. (*c*) A single-pressure stage with 2 or more wheels in it, each of which removes part of the energy (Fig 4 *B*). (*d*) A number of pressure stages, in which, by dividing the press drop, the development of energy is divided and hence the velocity in any stage is reduced. Each of these stages contains either a simple impulse wheel or is velocity staged. (*e*) Pressure staging and the utilization of the reaction principle; commonly known as the Parsons type. Fig 4 *C* shows its operation. Guide vanes, taking the place of nozzles, extend around the entire circumference. The pressure, as shown, is caused to drop in both guide and wheel vanes. The object is to generate only such velocity as can be utilized at the point where it is generated. The veloc in the guide vanes is sufficient to allow the steam to enter the moving vanes without shock. This gives a certain impulse effect. Expansion in the wheel vanes adds to the veloc, and the steam gives a reaction effect on leaving them. Only 3 stages are shown in the diagram, but commercial machines are made with 50 to 70 stages.

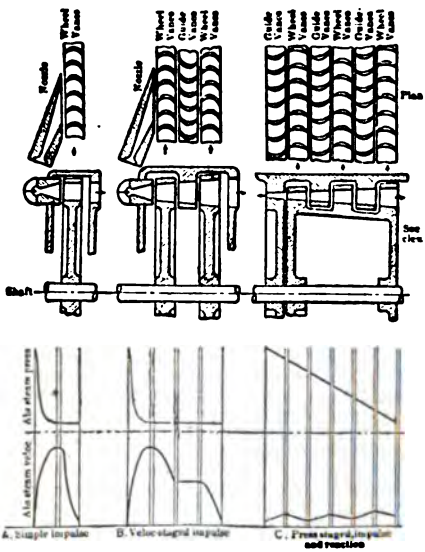


Fig 4. Diagram of Action of Steam in Different Types of Turbine.

The above types are divided among the various makes as follows: *DeLaval*, small size, 1 wheel with gearing; larger sizes, veloc or press-staged, 1 wheel in each stage. *Terry and Sturtevant*, velocity staged, with 1 press-stage. *Rateau and Zoelly*, press-staged, 1 wheel in each stage. *Curtis*, press-staged, and veloc staged with each stage; also pressure staged with the first stage, veloc staged, the remaining stages having a single wheel each. *Westinghouse*, small size, press-staged, with veloc staging in each. *Westinghouse-Parsons*, press-staged, utilizing reaction principle; also first stage impulse veloc-staged, and remainder press-staged reaction. *Allis-Chalmers-Parsons*, press-staged, utilizing the reaction principle.

6. Losses and Effect of Steam Pressure, Superheat, and Vacuum

Principal losses are: leakage losses past vanes, nozzle friction, impact on vanes, friction on vanes, and the friction of the wheels revolving in a steam atmos-

, the last being generally known as windage loss. Condensation, a large r in reciprocating engines, is unimportant in turbines, since there is practi- no variation of temp in any of the steam passages.

gh steam pressure is desirable and low back-press is particularly important. iprocating engine can not utilize expansion of steam down to a low back- as it would require too bulky a low-press cyl, but a turbine may be designed steam down to the highest vacuum. For relative work of steam at high ow press, see Sec 39, Art 4, 10. The use of superheated steam in turbines ises the friction losses, but the extent to which the superheat temp may be d depends upon cost of superheating, which varies according to conditions.

thods of governing. All low-power turbines, to approx 300 hp, maintain their under varying load conditions by throttle governing. From Art 5 it is seen that ethod is uneconomical, since for a given vane veloc the steam veloc for best condi- s fixed. In a few small turbines, some of the nozzles may be shut off by hand, when ing at low loads, thus allowing the remaining nozzles to operate at more nearly the steam press. The larger sizes of impulse turbines are governed by automatically g out and cutting in some of the nozzles. In the pure Parsons type, as steam is ad- over the entire periphery, the whole steam supply is interrupted periodically, the s being controlled by governor.

w-pressure turbines. A number of turbines have been installed to operate am at approx atmos press, with a high condenser vacuum. Some take the st from non-condensing, reciprocating engines, deriving more power from it the latter engines could, because turbines work with very low back-press.

he 59th St power station of the Interborough Rapid Transit Co, N Y, low-press es are attached to 7 500-kw reciprocating engines. Nearly all such installations een made to increase the capacity of an old plant, and there is rarely any economy an arrangement over a turbine installation.

re are several low-press turbine installations in which the turbine is supplied with from exhaust of engines, or other steam apparatus, which operate intermittently. cal case is at the Wisconsin Steel Co's works, South Chicago. The steam for the comes from the exhaust of rolling mill engines, which, when operating, deliver team than is needed by the turbine. Between engine and turbine there is a "re- tive accumulator," essentially a large horiz cyl, partly filled with water. When the of exhaust steam exceeds the turbine's need, the excess condenses and raises ap of the water. When the exhaust falls low or ceases, the turbine demand reduces ess in the accumulator, and steam is evaporated from the hot water. An auto- valve admits line steam to the turbine when the accumulator press falls below a ermined point. The accumulator is guaranteed to supply the turbine at its full 500 kw for a 5-min total interruption of steam supply. This device might ad- eously be used in connection with mine hoists.

7. Cost, Steam Consumption, and Weight

t of turbines varies with type and conditions of service. This is especially f the smaller sizes. The figures in Table 6 are average or approx. They e cost of generator, and of freight and erection, within 600 miles from s' shops, but not of auxiliary apparatus.

am consumption depends upon type, size, steam press, initial superheat, acuum. For any given case a much closer result than in Table 6 is ob- le by working out the Rankine cycle hypothetical water rate for the actual ions and modifying it by the effic *vs* Rankine cycle, as in Sec 39, Art 16.

ght of turbines is less than for equivalent reciprocating engines. This fact, coupled lmost total absence of vibration, permits large saving in cost of foundation, and the turbine to be placed on structures which could not support a reciprocating The space occupied is much less than for reciprocating engines.

Most of the data in the following table are from curves by H. G. Stott, R. J. S. P and W. S. Gorsuch (Amer Inst Elec Engs, June, 1914).

Types of service. Turbines operate at best effic when driving electric gener centrifugal pumps or fans, and other constant-speed machines. They are not econ for furnishing po variable speed; mo factory for a direct for intermittent c tion, as for hoistin can not be revers marine work, revers done by a separate bine). Turbines m indirectly used for i ing, with an electr driven by a tur erator. While ste

Table 6. Steam Turbine and Generator Sets

[illegible] kw	Wt per kw, lb	Steam per kw-hr, lb	Actual effic rs Rankine cycle, %	Cost per kw, erected
[illegible]	120–150	35–45	35–45	$45
[illegible]	120–150	20–30	40–50	35
[illegible]	60–100	16–20	50–60	16
[illegible]	40–120	13–16	55–70	8–12
[illegible]	35–40	12 5–13	70–75	7–8

[illegible] operate over a wide range of load, with little variation in steam consump [illegible] small engines show large increase in consumption when run much belo [illegible] load.

BOILER PLANT

8. Types of Boilers (1, 2, 3, 4, 5, 6)

General design must be such that natural circulation will draw the fe [illegible] through the cooler portion of the boiler, where it is gradually heated, a steam generated has free ac- cess to the space above water level.

No part of the boiler in con- tact with the hottest gases should form a pocket, as the steam generated in it would tend to force the water away and per- mit the metal to become over- heated, with danger of rupture. Boilers must withstand [illegible] [illegible] without dis- [illegible]

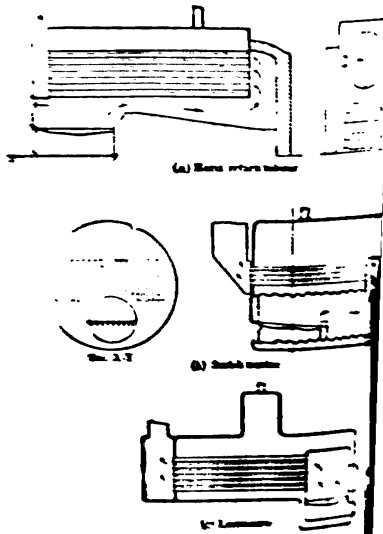


Fig 5. Fire-tube Boiler

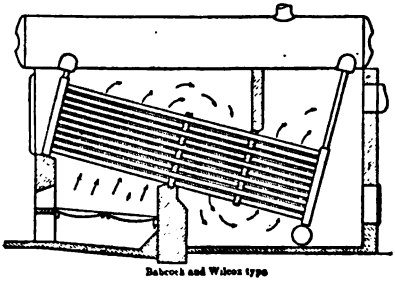
[illegible] large longitudinal flues, and [illegible] [illegible] contents. All these are now classed

dern **boilers** are of 2 general classes: fire-tube, in which the gases pass through expanded into headers forming part of the shell; water-tube, in which the water is tubes, the gases passing around them. Of the numerous designs only a few charactic types are herein described.

e-tube boilers are either internally or externally fired. Fig 5 *a* shows TERNALLY-FIRED, horiz, return-tubular boiler. The heads are connected to 4-in tubes, expanded into them. First cost is low and they are widely for moderate steam press and capacities of less than 200 boiler horsepower . The Scotch marine and locomotive types, Fig 5, *b* and *c*, are examples of NALLY-FIRED fire-tube boiler.

motive type is good for small temporary plants; having no setting, it is portable. nerally made in sizes of less than 100 bhp and for steam press of 70 to 125 lb gage. marine boiler has a corrugated firebox, extending through the shell, the gases ng through the tubes. The firebox corrugations relieve expansion strains due to s of temp. These boilers are very satisfactory in moderate size plants. Coms and absence of radiation losses characterize internally-fired boilers.

ter-tube boilers. Fig 6 shows 3 representative types; Babcock and Wilcox ably the best known. It meets practically all essentials of good design, like most other tube boilers, is transported, since t is of great weight. ade in sizes up to hp. Many other resemble the B & neral form. Some riz instead of veraffling. Stirling is a well-known rather cheaper e B & W, and has nade in units as as 2 350 bhp. y boiler is of same s the Stirling. It mparatively small s, each consisting umber of tubes ing 2 drums. As ections as necesay be connected r in a setting to single boiler, up s of over 2 500 Wickes boiler is r well-known ADVANTAGES of be over fire-tube they are faster s, may be obtained in much larger units, are easily transported, and a is less likely to cause disastrous explosion.



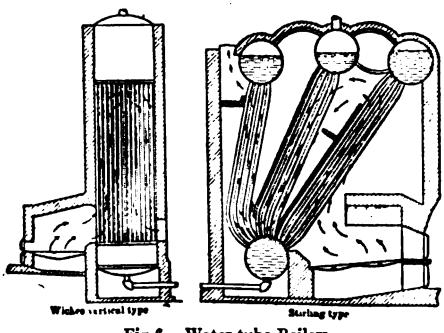


Fig 6. Water-tube Boilers

9. Boiler Attachments

offs. A blow-off pipe and cock must be connected to the lowest part of the This is for drawing off the water, when necessary, chiefly for blowing out sedi-

ment and scale. The blow-off is sometimes attached to a MUD DRUM connected to t bottom of the boiler. Surface blow-offs may be connected with the boiler at the wa surface, for blowing off scum caused by impure feed water or presence of oil.

Water column is a metal chamber with a GAGE GLASS on one side (for observing t height of water in the boiler) and 3 TRY COCKS on the other, for finding the water leve the gage glass is broken or stopped. Some water columns automatically blow a whis if the water is too high or too low.

Safety valves are essential. Legal requirements as to their design and method connection vary in different states and should be ascertained.

Fusible plugs, required by many state or municipal boiler laws, consist of a h plug with a fusible metal core, screwed into the boiler below normal water line and in path of the hot gases. While covered with water the temp of the plug is below t melting point of the core; if the water level falls below the plug the core fuses, allow steam to escape and press to fall. The plug is placed above the expanded joints of t tubes, which would be weakened by high temp and might rupture.

10. Ratings, Efficiency, and Costs (see also Sec 39)

Boiler horsepower (bhp) is defined as the transfer in 1 hr from the gases to t water of a quantity of heat sufficient to evaporate 34.5 lb water at temp of 212° to steam at sa temp. This is equ alent to 33 479 B per hr. *Note.*—Th is no relation wh ever between 1 h and an engine which is 33 000 ft of work per min bhp, under norm conditions of fe temp, and stea press, is approx equ alent to 30 lb of wa evaporated per a figure comm used for rough calculations, and accepted as a "commercial" bhp. Rat of boilers as to the hp they will develop is arbitrary; Table 7 shows th commonly used, including data as to relative dimensions of grates and rates combustion. Rate of combustion varies considerably with kind of coal, dra and mode of firing.

Table 7. Approx Proportion of Heating and Grate Surface per HP

Type of boiler	Sq ft of heating surf per hp	Coal per sq ft of heating surf per hr	Relative rapidity of steaming	Heating surf per sq ft of grate	Lb coal per sq ft of grate per hr	Lb water per lb coal
Water-tube...	10 to 12	0.3	1.00	35 to 50	12 to 20	9 to 12
Cyl tubular...	14 to 16	0.25	0.60	25 to 35	10 to 15	8 to 11
Vertical tube.	15 to 20	0.25	0 60	25 to 30	10 to 15	8 to 10
Locomotive...	12 to 16	0.275	0.55	50 to 100	20 to 40	8 to 11
Flue.....	8 to 12	0.4	0.25	20 to 25	10 to 20	8 to 10
Plain cyl.....	6 to 10	0.5	0.20	15 to 20	15 to 25	7 to 9

Efficiency. For losses in boilers and furnaces, see Sec 39. Effic of th boiler is not 100%, since the temp of the furnace gases can not be r below the steam temp. A large number of tests on different types of bo grate, with different ratings and modes of firing, show a combined b furnace effic of 60 to 80%; aver about 70%. In ordinary operation approaching the aver may be maintained in large plants, with strict supervision, but in small plants the aver effic is probably not much over 55

Costs. Following formulas are approx (*Engineering*, Nov 15, 1902).
Horizontal water-tube. Cost (dollars) = 500 + (9.2 × rated hp).
Vertical water-tube. Cost (dollars) = 500 + (8.5 × rated hp).
Horiz tubular. Cost (dollars) = 100 + (6.5 × rated hp).
Small vert fire-tube. Cost (dollars) = 100 + (5 × rated hp).
Cost of the water-tube boilers is based on 125 lb steam press and re

ɛ surface per bhp; that for horiz return-tubular boilers on 12 sq ft heating
. Scotch marine boilers may be estimated on same basis as the water-
aking 8 sq ft per bhp.
of plain settings are roughly:
iz water-tube. Cost (dollars) = 400 + (0.8 × rated hp).
iz return-tubular. Cost (dollars) = 300 + (0.7 × rated hp).

11. Feed-water Purification (6)

rious substances occur in practically all water. They are detrimental either oilers or to their proper operation. Troubles from impure feed water are: forma- scale, corrosion, and priming or foaming. Solid matter may be precipitated as it, or may form scale on the heating surface. SEDIMENT causes little trouble; ularly drawn off at the blow-off (Art 9). SCALE seriously retards heat transfer, llowed to accumulate the heat may weaken the metal sufficiently to cause rupture. ION causes general weakening of the boiler, finally resulting in a breakdown at oint. PRIMING causes steam to carry with it more or less water, which, if it enters ine cyl in sufficient quantity, may blow-out the cyl head. Substances causing oubles may be natural constituents of the water, or be due to waste from a plant ory at a higher point on the stream from which the water is drawn. Where the sed steam from surface condensers is pumped back to the boilers the oil should be ely separated. Condensed steam from turbines does not give trouble, as oil is ily in the bearings.

tment. Scale-forming substances may be precipitated by heating, or by adding hemicals to the water supply and eliminated by filtration; or chemicals may be to the feed water to cause the solid matter to form as a sediment instead of in- g the heating surface. Anti-scale chemicals are known as BOILER COMPOUNDS. ces causing corrosion or priming are treated in a similar manner. Electrolytic sometimes causing pitting and corrosion, may be neutralized by hanging zinc plates he boiler.

Table 8. Feed-water Purification

ouble	Cause	Remedy
	Sediment, mud	Filtration
	Readily soluble salts	Blowing off. Heating feed water and precipitating caustic soda, lime, and magnesia
	Bicarb of magnesia.	
ation	Lime, iron	
	Organic matter	See below
	Sulphate of lime	Sodium carbonate Barium chloride
	Organic matter	Precipitate with alum or ferric chloride and filter
	Grease, oil	Add slaked lime or sodium carb and filter
on	Chloride or sulphate of magnesium	Sodium carbonate
	Sugars, acid	
	Dissolved CO_2 and O	Slaked lime, caustic soda; heating
	Electrolytic action	Zinc plates
	Sewage	Precipitate with alum or ferric chloride, and filter
s	Alkalies	Heat feed water and precipitate
	Sodium carbonate in large quantities	Barium chloride

ire feed water should be analyzed, to determine what it contains and the remedy, o the wt of solids in a given wt of water and hence the proper wt of chemicals to be for neutralizing. Boiler compounds, if used in excessive quantity, may them- ause boiler troubles. Avoid all secret preparations. They are either worthless ain ordinary chemicals for which a higher price is charged under the name of a

special preparation. In the latter case, it is only a matter of chance that the chem are suited to the water to be treated. A number of concerns make a business of tre impure water for boiler feed and industrial purposes. In case of bad feed water often satisfactory to put the problem in the hands of one of these companies. importance of the proper treatment of impure water is shown by the extensive pur tion systems installed by many large plants and factories.

12. Furnaces for Solid, Liquid, and Gas Fuels

(For the theory of furnace operation, see Sec 39, Art 13; for details of large subject, see Bib.) The best efficiency is the result of supplying just s cient oxygen to combine with all of the combustible. With solid fuels this impossible in practice, the greatest loss being frequently due to excess air ra than imperfect combustion.

Bituminous coal. The combustible is partly volatile matter and partly fixed ca the relative proportions varying widely (Sec 35). The volatile matter is driv rapidly by heat as hydrocarbon vapors. To effect combustion these vapors m intimately mixed with enough air while at a high temp. Contact with a relatively surface, as the boiler tubes, causes formation of soot and prevents combustion. M forms of furnace have been designed to overcome the difficulty. Some provide a s ate combustion chamber, where complete combustion is supposed to take place be the gas passes to contact with the boiler. Others, by natural draft or steam jets, duce air at the bridgewall or the furnace sides, and brick piers are sometimes arrange baffles thoroughly to mix the air and gases. Some of these designs are effective, resulting high temp often produces rapid deterioration of the brickwork. This has especially marked when certain substances in the ash form a flux for the fire prevented by analyzing the ash and using a firebrick with which it will not flux.

While special furnaces may assist in effecting good combustion, the best results obtained by intelligent and experienced firemen. Some concerns spend a large am on apparatus for improving efficiency, while refusing to pay wages that will attract firemen, or to hire a sufficient number. The fire should be kept even and without h Aver depth of bed for bituminous coal is 8 to 10 in, but this depends on the avail draft. The fire should not be allowed to become dirty, a dark ash pit being an evid that cleaning is necessary. An important rule is that the coal shall be spread ever thin layers. In adding a heavy bed of fresh coal much of the volatile is driven off burning, since the temp above such a bed is too low. Instead of spreading the fresh on the fire, it may be piled on a dead plate in front, where the volatile is distilled remaining coke being then pushed over the fire. This is satisfactory only when fire is being carried.

Anthracite coal (Sec 34), containing comparatively little volatile matter, is hand differently from bituminous. Steam sizes form a dense bed, presenting con resistance to the passage of air. Hence the can not be thicker than 3 or 4 in. and the tion of holes or thin spots must be a This is more important than with bitumi coal and efficient firing is more difficult.

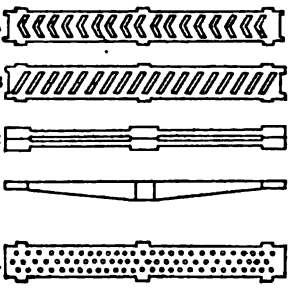


Fig 7. Boiler Grate Bars

Grates for hand-fired furnaces are stati or rocking. STATIONARY GRATES requ the ash be pulled out through the fire make it difficult to keep air spaces thr ash and clinker. ROCKING GRATES are numerous forms, worked or dumped by at the front of the boiler. Grates are di into 2 or more sections, so that the co be piled on one part while cleaning an Fig 7 shows ordinary forms of grate bars the herringbone, *B* is a Tupper, and common straight bar, all in plan. *D* sec elevation. Grate bars are generally about 3 ft long, and when used in a rocking this makes 1 dimension of a section, the other dimension depending on size of ma

mber of grate sections. Size of air space must be suited to the kind of coal; if too nuch fuel may be lost in the ash. *E* is a PINHOLE GRATE, used for burning saw- ıd other small sizes of fuel. Hand-fired furnaces should not be longer than 6 ft ıminous coal; beyond that, it is difficult to slice the fire. Furnaces for anthracite : 10 or 12 ft long, as no slicing is required.

chanical stokers. Advantages over hand-firing: (*a*) coal is fed continu- instead of intermittently, permitting a constant air feed of proper propor- whereas with hand-firing the air supply should be large directly after firing nstantly decreased until time for next firing; (*b*) the fire is kept even, with ıuent reduction in amount of excess air passing through thin spots and (*c*) the gases maintain a high, even temp, while with hand-firing, the g of a door often reduces the temp as much as 200 or 300°; (*d*) labor of is greatly reduced. But, it can not be too strongly emphasized that nical stokers will not give good results unless they are constantly super- ɔy an experienced man. Hence, while they are practically essential for a ılant, they may not show any net saving in a small plant (6).

sification of mechanical stokers: step grates, chain grates, and underfeed. STEP are arranged in a series of steps. Each grate (in some cases, alternate grates) ed from a horiz to a forward inclined position. A rocker arm connects all the and is actuated from front of furnace. The rocking action keeps the fire broken the coal continually moving forward. Coal is fed to the top of the grate from a . Rate of combustion is determined by rapidity of the rocking action, thickness bed, and intensity of draft. The relation between these 3 factors must be so main- that all the fuel will just be consumed when the coal reaches the end of the grate, ı plate or dumping grate receives the ash. With some step grates the air is drawn n the bars; others use hollow bars, into the ends of which air is forced and admitted fire through narrow slits in the sides of the bars. Some grates incline from front t of furnace; others consist of 2 sections, each feeding from the side toward the . CHAIN GRATES are a continuous chain formed of grate bars, mounted on 2 Grate moves from front to back of the furnace, and is either horiz or slightly l. Coal is fed from a hopper in front. Depth of coal and speed of grate are regu- t will. General conditions of operation are practically the same as for the step When bituminous coking coal is used, special devices are installed to keep the ɔperly broken up. UNDERFEED STOKERS consist of a horiz trough or retort, into he coal is forced by a ram or screw. Air enters through tuyeres along the top edges retort. On each side are dead plates or grates, the bed of the fire lying above. ɔen coal in the retort is reduced to coke, the volatile matter passing up through the l in close contact with the air supply. As fresh coal is forced into the end of the the coke is driven up through the bed and over on the plates or grates. Each ıas 1 or more stokers, depending upon its capacity. Underfeed stokers require ıace and work well even upon poorest grades of coal. They are probably the best r a small or medium-size plant.

t of mechanical stokers, of different types, installed under a 350-bhp :k and Wilcox boiler, ranges from $1 200 to $1 500. In large plants the hoppers are fed from overhead bunkers, receiving coal by conveyers; in ılants it is shoveled directly into the hoppers. Cost of handling coal up to ilers must be considered aside from the stokers for each individual case.

ces for burning powdered coal have long been on the market, but have been ıed. Since pulverized coal can be injected into the furnace in intimate contact with , very efficient combustion can be maintained. The lack of popularity of this is probably due to the objection of plant owners to installing the pulverizing ery and the special feeding apparatus.

nel is too large a subject to be treated fully here. In the numerous successful oil- : systems, the aver boiler effic is much higher than for coal. Conditions essential ess are: all the oil must be consumed before reaching the boiler surface, where tion would be retarded or arrested. Hence, the oil particles must be small, to the duration of burning. The oil spray must be thoroughly mixed with air before

it is raised to the furnace temp; otherwise, combustion will be poor. While fixed baffles facilitate good combustion, they deteriorate rapidly.

The oil is generally heated outside of the burner to about 150° F, to make it sufficiently fluid to flow readily through the supply pipes. In the burner it is atomized, by forcing it through a special nozzle under high press, or by a steam or air jet. Air for combustion is supplied outside the burner.

Gas burners are simpler than those for oil, as gas mixes more readily with air. The injection of a perfect mixture of air and gas may cause difficulty by localizing combustion in the furnace. This is overcome by the system of surface combustion (invented by C. E. Lucke), which permits such mixtures to be burned in a bed of refractory material without visible flame.

12. Superheaters, Economizers, and Feed-water Heaters

Steam superheaters. For the advantages of superheated steam, see Sec. Art 12, 15. Superheaters are placed in the boiler setting, or in a special setting where they are fired as a separate unit. It is generally impracticable to place a superheater in the waste gases from the boilers, because the difference between the steam and gas temperatures at this point is so small that, to transfer enough heat, the area of superheater heating surface would be prohibitive.

In boilers of the Babcock and Wilcox type, the superheater is generally placed under the drum that the gases leaving the front pass flow through the superheater before entering the second pass. In the Stirling boiler it is similarly placed, so that the gases reach the superheater after passing the first section of boiler tubes. In some boilers of the B & W type, but with less space between drum and tubes, the superheater is placed in a separate chamber and heated by by-passing part of the furnace gases. In horiz return-tubular boilers, the superheater is set in the rear connection. Separately fired superheaters either superheat all the steam from a battery of boilers, or only a part of it, as for a particular use, when it is desired to have the remainder saturated. Such superheaters have been built in units of over 3 000 bhp.

Superheaters generally consist of inlet and outlet heads, connected by banks of wrought-iron tubes through which the steam flows. Sometimes 2 or more sections are arranged in series. The tubes of the Foster superheater are protected from injury by the hot gases by C-I rings shrunk on, thus making a surface like that of an air-cooled gas engine. This increases the heating surface, and permits more compact construction.

Cost of superheaters varies widely with the steam press, amount of superheat, gas temp, and type of boiler for which they are used. Aver cost under ordinary conditions is between $2 and $4 per bhp.

Economizers recover part of the waste heat in boiler gases by using it in heating the feed water. They are generally of C-I tubes about 10 ft long, set vertically between headers, and placed in the breeching near the boiler. Each tube has a scraper, which is slowly moved up and down by an outside mechanism to keep the tubes clear of soot and dust.

Economizers show a slight net saving in operating cost. By feeding the boiler hot water they reduce the strains caused by cold feed water, thus saving in boiler repairs. In considering the installation of economizers in a plant operating on natural draft, it must be remembered that the draft will be reduced by friction of the gases through the economizer tubes, and by reduction of temp of the stack gases. Aver cost ranges from $5 to $6 per bhp.

Feed-water heaters heat boiler water by steam, usually the exhaust from engines or pumps. They are made in great variety, but are comprised in two classes, open and closed.

The OPEN HEATER, in which the steam comes in direct contact with the water, consists of a casing filled with horiz trays or pans. Water is pumped in at the top and flows from tray to tray. The exhaust steam fills the entire casing. Advantage of the open heater is that scale-forming solids in the water depusit on the trays, whence they are easily

If the steam contains oil, the feed water must be filtered before going to the
As open heaters are generally run at approx atmos press (the ordinary exhaust
for auxilliaries) the boiler-feed pump must be placed between heater and boiler.
ften requires a second pump to force the water into the heater. The feed pump
ave its suction lower than the bottom of the heater, because water approaching
np of steam at atmos press will not lift to a pump suction. Open heaters are some-
ised under press of live steam, to purify the water when it contains solids which
t precipitate at the temp of exhaust steam. CLOSED HEATERS, in which the water
am are kept separate, consist of a C-I shell forming the steam space, and contain-
es, through which the water is forced. The tubes are either straight or coiled,
eaters using a special form of corrugated tube to facilitate heat transfer. The
re strong enough to withstand the full feed-line press, and the heater is generally
between feed pump and boiler. As the steam does not come in contact with the
oil does not get into the feed, but it often adheres to the tubes and reduces effic of
ansfer. When the water velocity through the tubes is high, there is little trouble
rmation of scales. Such scale as is precipitated is forced into the boiler, where it
and is easily blown off. Scale is difficult to remove from most types of closed heater.
nnecting heaters with exhaust systems, all the exhaust steam is sometimes piped
eater, and what is not condensed escapes to the exhaust pipe or some other heating
. Or, the heater may be connected as a dead end to the exhaust main, the steam
being controlled by the press in the heater.

14. Condensers

cts: to reduce back-press on steam-driven apparatus, and to save boiler water
ood feed water is scarce.

condensers. In these, condensation is effected by bringing the cooling and steam exhaust into intimate contact. Fig 8 to 11 diagrammatically representative types.

g 8, the discharge is directly connected to the suction of a pump, which removes
ted water and condensed steam. The water is sprayed into the steam through a
aped valve, which also adjusts
antity. Fig 9 is the SYPHON
SER, similar to the so-called
TRIC CONDENSER. Instead of
pump to remove the discharge,
ndenser is placed at least 34 ft
the surface of the HOT WELL, so
e column of water in the dis-
pipe (connected to the bottom of
denser) will be sufficient to over-
mos press and prevent flooding
ondenser chamber. The cooling
dmitted around the lower edge
lindrical sleeve, as indicated, is
d in amount by raising or lower-
sleeve by a hand wheel. The
art of the sleeve passes through
ng box, which prevents the water
wing around its upper edge. The
asses through the sleeve and is
ed by coming into contact with
r surface of the hollow cone of
Fig 10 shows the KOERTING
R CONDENSER. Water enters

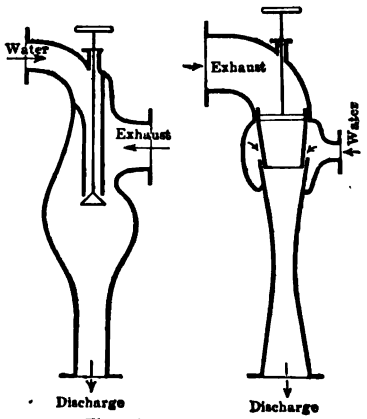


Fig 8, 9. Jet Condensers

nozzle at the top and flows down through the central jet tube. The steam is
ed by contact with the water through a large number of orifices, connecting the
er chamber with the jet tube. The orifices are drilled obliquely, so that the
nters the jet in the direction of the flow and its velocity tends to increase the
of the jet. The largest sizes have a series of water jets. Velocity of the dis-
s sufficient to overcome atmos press, so that neither a pump nor a discharge pipe

tion of high velocity, the condensation may be 6 to 10 lb steam per sq ft per hr. T condensers have maintained vacuums of 25 in of mercury. Loss of cooling water by oration is approx 1 lb per lb of steam condensed.

Cost of condensers varies widely with capacity of the power unit ser vacuum required, and temp of cooling water. Following figures are for com equipments, including cost of installation (1).

Type of condenser	Cost per kw of power unit se
Syphon condensers	$2 to $3
Jet condensers	3 to 4.50
Barometric condensers with dry-air pump	4 to 6
Surface condensers for 26-in vacuum	3.50 to 5
High-vacuum surface condensers	3.50 to 10
Leblanc jet condensers and pumps	2 to 6

Though a separate condenser is desirable for each power unit, the steam saving by condensing will not always offset the fixed charges on the cost of the condens ment, when the power units operate intermittently, or for most of the time at l below the maximum. In such cases it may be profitable to connect to one several power units; as hoisting or rolling mill engines, where all the machines operate simultaneously at full power. Power consumption of the condenser auxil varies from 1 to 10% of the rating of the main power unit.

15. Cooling Water for Condensers

Temperature of condensate corresponds hypothetically to the steam pres the condenser chamber, and in an ideal condenser the cooling water should at the same as the condens Hence, the wt water for co sing 1 lb steam $L \div (t_2 - t_1$ t_1 = temp of ing cooling t_2 = temp of cooling water. latent heat of correspondin and corrected moisture. I sumed that t_3 = temp of densate. Fig shows the rel between wt ing water and of steam cond calculated by a equation. B practice the ence of air raises the ab in the cond

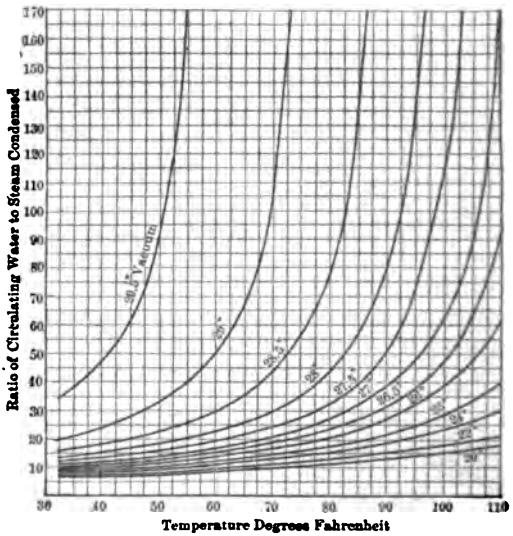


Fig 14. Condenser Water Ratio Chart

above that corresponding to the temp of the condensate.

In jet condensers the condensate and cooling water leave together and at the temp, but in surface condensers the cooling water leaves at a temp lower than

condensate. The chart (Fig 14) may be used to find approx amount of cooling water uired for given conditions, by assuming that the temp of the cooling water is 5 to 10° her than is actually the case. This correction covers the HEAT DROP for ordinary densers. The chart shows graphically how rapidly the cost of condensing equipment s with increase in vacuum. In surface condensers the surface must be greatly in-ased, since the mean temp difference is less, and a greater vol of steam must be handled oth surface and jet condensers. Also, the circulating pumps must be much larger to dle the increased amount of water, with consequent increase in operating cost.

Cooling towers, sprays, and ponds are useful when there is insufficient water the ordinary operation of condensers. The condensing water is cooled for use by bringing the hot water into intimate contact with air. Part of the heat removed by raising the sensible heat of the air, and part by evaporation of ter. Normally, 85% to 90% of the cooling is done by evaporation.

Cooling pond. The governing factor is surface area, depth of water being immaterial. the rate of cooling depends upon air temp, relative humidity, and aver wind velocity, difficult to give a rule for area of cooling pond surface. 100 to 200 sq ft per hp of main am unit are suggested. If there are prevailing winds, it is best to make the pond long narrow, with the long axis perpendicular to normal direction of the wind. Thus the is blown away from the pond, as soon as it becomes warm and saturated and unable do any further cooling.

Spray nozzles require much less space than cooling ponds. They are placed in the m over a pond. Water is pumped to them at 5 to 10 lb press per sq in, and sprayed into air where it is cooled by evaporation and conduction. The spray falls into the pond, ence it is drawn by the condenser circulating pump. Makers state that the water s is not over 2%, and may be less than 1%. The nozzles are from 0.5 to 2 in diam; acity (at the supply press named), 15 to 300 gal per min.

Cooling towers, often of considerable height as compared to their ground area, are ed with some material (as wire mats or wood strips) that will afford large surface area hout seriously impeding the passage of air. The hot water, pumped to top of the er, trickles through the filling. One type of tower is placed on the roof of a building, on an open ground space, and air is allowed to circulate between the wet surfaces. this, the cooling effected is largely dependent on the wind velocity; the size of the tower elatively large. Another type has closed sides and draws its supply of air in at the ttom by chimney action, since the hot moist air within is lighter than the surrounding nosphere. This may be placed wherever convenient, since it is independent of wind rents. A third type uses forced circulation of air, and is the most economical of ice, since the cooling capacity depends upon the volume of air brought into contact h the wet surfaces. Makers state that the power for operating a forced-draft tower bout 3% of the power of the main engines served, for moderate size plants and even s for large plants.

16. Chimneys and Mechanical Draft (7, 8)

Functions of chimneys: (*a*) to provide sufficient draft to draw through grates d fire bed enough air to support the required rate of combustion; (*b*) to carry e gases to such height that they will dissipate without being objectionable.

Design. Intensity of chimney draft depends upon the relative wt of the lumn of gases within the chimney and a similar column of atmospheric air. nce, the design of a chimney depends upon: the temp within and without the imney, barom press, height of chimney, and the physical characteristics of the ses. Having determined the gas velocity in the chimney, a cross-sectional area n be computed which will carry the required volume of gases.

Flow of gases through the furnace, boiler, flues and chimney, may be calculated from clet's formula, $h = (v^2 + 2g)\,[L + A + (BL + C)]$, where h = head producing the w, ft; v = veloc of gases in the chimney, ft per sec; A = coeff of resistance of passage rough the fire; L = total length of path of the gases, ft; B = friction coeff for the rfaces traversed by the gases; C = area of the chimney divided by its perimeter. For dinary conditions, $A = 12$ and $B = 0.012$. Normal stack temp, 450° to 650° F.

Empiric formulas f

Intensity of draft
Following figures are

Intensity of draft, in	
Bituminous run-of-	
Bituminous slack	
Anthracite pea	
No 1 Buckwheat	
No 3 Buckwheat	

Construction. C
chimneys, generally
expensive. They sh
double with an air sp
resulting in a higher
builds chimneys from
air passages to reduc
becoming common,
failures have occurr
chimneys are either
made of thin plate, u
4 ft diam and 75 ft
neys they deteriorat
inside is exposed to c
generally lined with f
are heavier than thos

Costs of chimneys
a chimney 150 ft hig

Common red brick	
Radial brick	
Steel, self-supporting	

Mechanical draft
carried off by the chi
by the draft press as
to be only 0.1 to 0.2
press by a blower or
comparison does not
cost of producing dra
plus depreciation an
gases by economizers
high temp of the gas
is the interest on init
deprec, and repairs,

In cities the laws con
cient height to produce
tion is whether enough
the operating cost of
interest and deprec. A
regulated, and a slight
operation gives sufficie

Draft, induced or for
forced draft, the air is
air needed to support t
of fire, boilers, and flue

:, or in the breeching. The press at the blower inlet is reduced below atmos press, the atmos press in the ash pit forces the air through the fire and boiler. The chief ntages claimed for induced over forced draft are that combustion is more even and gas press within the boiler setting is always less than that of the surrounding atmos, ice it is unnecessary to shut off the draft when cleaning fires or stoking. On the r hand, there is a considerable loss due to excess air which filters in through leaks ie boiler setting and through the doors, when open. Also, with induced draft, the ers must be larger, as the vol of hot gases is considerably greater, and the cost of ation is higher since power required to drive the fans for the same press-change is :tly proportional to the vol.

eam-jet blowers are generally in the form of a hollow ring, perforated with holes on face. Steam is admitted to the interior and the jets from the holes induce a draft llel with the axis. They are placed at the base of the stack, or in a hole in the ash- rall. They are generally uneconomical for constant service, tests showing that they 5 to 20% of the total boiler steam generated; but, as their cost is low, they may be rable for small plants which can carry their normal load with chimney draft, but h for short periods must meet peak loads far above the normal.

entrifugal blowers of many types, driven either by steam engines or electric motors, : been used. Small propeller fans set in the ash-pit wall, and direct-connected to m turbines, have found considerable favor.

STEAM POWER PLANT DESIGN

17. Factors Governing Choice of Type

)perating cost. The best plant is that which has the lowest total operating t, including fixed charges, while satisfactorily performing the required service. rrating expenses include cost of wages, fuel, water, oil, supplies, and ordinary airs. Fixed charges include cost of administration, interest on investment in chinery and buildings, ground rent of land occupied, insurance, taxes, and reciation. When fuel and labor are cheap it is uneconomical to invest a large n in fuel-saving or labor-saving devices. Cost of power should be figured as cost per actual hp hr, and not be based on the rated power of the plant, as is en done. LOAD FACTOR is the relation between the aver load of the plant for 24-hr day and the load which it is rated to carry. Electric-lighting plants must able to carry a peak load which may not last longer than 1 or 2 hr, and may rate the rest of the time under only a small fraction of the max load; that is, load factor may be only 10 or 15%.

While calculations for a given case may show that fuel-saving devices are warranted steady load conditions, they may not pay in a plant having a low load factor. Hence, leciding upon details of the equipment, the al cost per actual hp-hr delivered must be it in mind. In some cases reliability of vice is so important as to demand an elabo- ion and duplication of apparatus not war- ited by power-plant economics.

Hp of plant	First cost per hp, erected	Annual operating cost per hp
20 to 100	$150 to $200	$70 to $130
100 to 1 000	60 to 160	25 to 70
1 000 to 5 000	50 to 65	15 to 25

The factors influencing the design and op- ting cost of power plants are too numerous discussion here. In the accompanying le of average costs, first costs include buildings; engines for small plants assumed to simple, non-condensing; for medium size, compound condensing engines (or turbines); large plants, turbines. Operating costs include fixed charges, and are based on 10-hr y and coal at $3.50 per ton.

Economy of operation is poor when machines are run far below their rated load; hence, plant should be so chosen with reference to expected load fluctuations that, by throw- some of the machines in or out of service, the others will operate at somewhere near ir most economical conditions.

18. General Arrangement of Plant

While local conditions will decide the detailed arrangement of plant, certain general principles of design should be followed.

Boiler and engine rooms should be separated by a tight partition, to prevent the machinery from dust. Boilers should be connected to the steam main (header) by curved pipes to allow for expansion, and each boiler should have a stop valve between it and the main.

When boiler and engine rooms are side by side, a pipe is run through the dividing wall to each engine, with a valve at its junction with the main besides the stop valve at the engine. It is desirable, though not indispensable, to place a steam separator in the branch pipe just above the engine cylinder. In any case there must be a drain pipe at this point to remove water before the engine stop valve is opened. The above layout permits the use of a comparatively small steam main, since no portion of it carries the total supply for all the engines. When there is more than one bank of boilers, they are generally placed face to face with enough space between them to permit removal of ashes and proper firing. The mains from 2 banks of boilers are usually connected at both ends, forming a rectangle. Valves at different points in the mains allow steam to be turned in either direction, and permit cutting out of sections for repair without interfering with operation of the plant. A further guard against interruption of service is to install a separate steam header for the engines, divided by valves into sections and connected to the boiler mains at 2 or more points. The headers are sometimes in duplicate. When boiler and engine rooms are end to end, more and larger piping is required. Since part of the piping has to carry all the steam and a separate engine main is necessary. To insure continuous service there must be double mains for the engines.

Feed pumps may be in either the boiler or the engine room. They are generally direct-acting, but centrifugal pumps are being more and more used, driven by electric motor or steam turbine. The pumps deliver into a main feed line, with branches to each boiler. A regulating valve is placed in front of each boiler, with check valves in the line before or after the regulating valves. To avoid interruption due to accident, the feed system is sometimes in duplicate. In some plants the duplicate (auxiliary) system is the same as the main system; in others, an injector is connected to each boiler. The exhaust from the plant auxiliaries goes to a feed-water heater. With piston pumps, the heater must be of the closed type to prevent the oil in the pump exhaust from entering the boiler, but with turbine-driven auxiliaries either open or closed heater may be used.

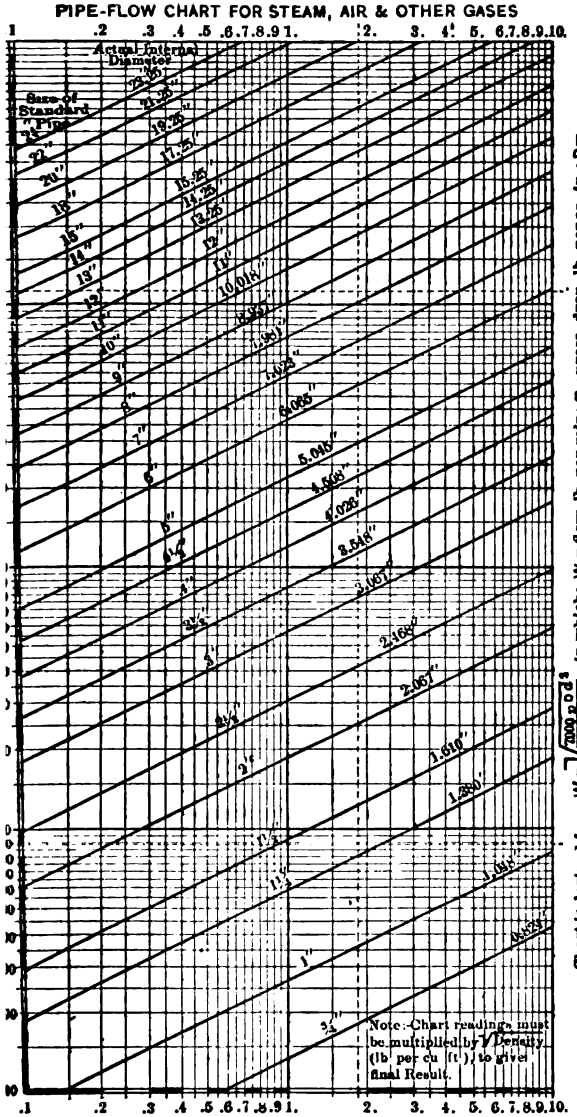
Condensers should be as close as possible to the units which they serve, not only to avoid pipe-friction loss, between engine and condenser, but also to reduce loss of vacuum due to air leakage at pipe joints by minimizing the number of joints. Where space is limited, the condensers are placed below the engine room floor.

19. Steam Piping (For tables of pipes, fittings, and valves, see Sec 41)

Pipe sizes should be properly proportioned to the amount of steam they are to carry. Too large a pipe increases heat loss and initial cost; too small a pipe increases pressure drop due to friction. In the past, 5 000 ft per min was considered the max allowable steam velocity, but the aver in new installations is nearer 10 000 ft, especially when superheated steam is used. Calculation of pipe size on basis of steam velocity is unsatisfactory, since it ignores length of pipe and friction drop. A widely accepted formula for flow in pipes of steam or air is

$$W = \sqrt{\left(7\,000\, p\, D d^5\right) \div \left(L + \frac{3.6\, L}{d}\right)}$$

where W = flow per min, lb; p = press drop, lb per sq in; D = density, lb per cu ft; d = internal diam of pipe, in; L = length of pipe, ft. Fig 15 shows the results of this formula for standard pipe from 0.75 to 24 in diam. In calculating press drop, friction of valves and bends are allowed for in terms of equivalent length of straight pipe. Loss in 1 to 3-in elbows or tees is approx equal to 5 ft of pipe, 3 to


PIPE-FLOW CHART FOR STEAM, AIR & OTHER GASES
Actual Internal Diameter
Size of Standard "I" Pipe
Note: Chart readings must be multiplied by √Density (lb per cu ft) to give final Result.
Chart is derived from W = √(2000 p D^5 / L(1+ 4f/D)); in which: W = flow, lb per min; p = press.-drop, lb per sq in.; D = density, lb per cu. ft.; d = internal diam, in.; L = length, ft

Allowable Press-Drop, lb per sq in, per 100 ft length of Pipe

Fig 15. Chart for Flow in Pipes of Steam or Air

5-in = 15 ft, and 5 to 10-in = 30 ft of pipe. Loss in a globe valve is about do that of an elbow.

Expansion joints. Pipes are supported by hangers, floor stands, or wall brac To allow for expansion and contraction, it is best to anchor the pipe at certain p and provide loops or U-bends. Pipe hangers will permit movement by swinging wall brackets should have rollers. Sleeve expansion joints may be used for large p where lack of space prevents long bends. To prevent leaks connections between ea or boilers and steam mains should always have long bends (or their equivalent

Drainage of steam pipes. To prevent dangerous accumulation of condensa pipe lines should have a slight slope in the direction of flow, so that the water will to the drainage points even if the steam velocity be high. Openings at drainage p should be wide, so that the water will not cross them even when carried along at velocity; they are connected with a chamber large enough to hold considerable conde tion. For small and medium-size pipe, DRAINAGE CHAMBERS are best made by conn ing to a tee in the line a short drop leg of the same size pipe. These legs are capped blank flanged at the bottom, with water gages at important drainage points. Drai chambers are connected to a STEAM TRAP, for removing condensation without allow leakage of steam. Each trap has a by-pass around it, normally closed by a valve the trap fails to work, the water may be drained through the by-pass. Traps can be depended upon to drain a cold line; the by-pass should be opened when steam being turned on. Condensed steam ("drips") may be returned directly to the boile it is from a high-press main, or to the hot well or open feed-water heater. Valve steam lines over 6 or 8 in diam should have a small by-pass around them. Steam through this allows the line to be warmed gradually; also, large valves are difficult open unless the pressure on both sides is equal. Large valves often include a by-pas

RECIPROCATING PUMPS

20. Classifications (17, 18, 19, 20)

Water-end classification comprises the simplex pump, having a single wa cyl, duplex, with 2 water cyls in parallel, and triplex, with 3 water cyls in paral The pump is single-acting when it takes water on one side of the piston only double-acting when it takes water during both strokes.

The water cyl has a piston (bucket) or plunger. A piston makes a loose fit in the and is packed with hemp or a metallic packing ring (Fig 16). Piston-and-ring p (Fig 17) is a cheaper construction, but is apt to score with gritty water. Both types permit undetected leakage past the piston. They may be used for heads up to 150 lb per sq in. For pressures of 150–300 lb, outside-packed plunger pumps (Fig 18, 19) are recommended; for pressures exceeding 300 lb they are necessary. With outside packing leakage is readily detected, and stopped while the pump is running. Scoring (cutting) of the plunger is also plainly visible. Plunger pumps cost more than piston.

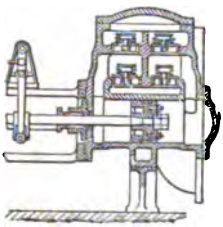


Fig 16. Water End of Piston Pump

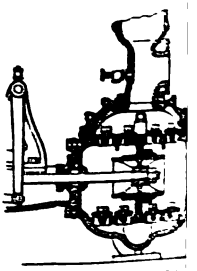


Fig 17. Piston-and-Ring Pump

When both suction and discharge valves are above the barrel (Fig 16, 18), the pump of the submerged type; when the suction valves are below (Fig 17), it is of the straight-way type. Submerged type is used for hot liquids and condenser service, the pump being surrounded by cool and de-aerated water, which protects the packing and keeps

action. But, as the flow of water in the cyl is reversed at every stroke, it is not ended for strokes over 14 in. Straight-way type costs more, but the water has direct flow through the and is therefore used for ater velocities.

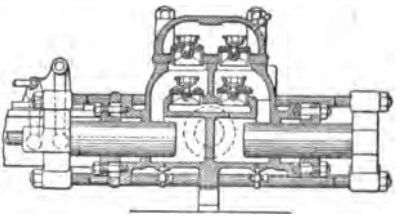
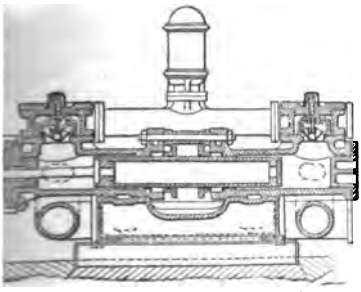


Fig 18. Outside, End-packed Plunger Pump

am-end classification. s are divided into simple, und, and triple expan- ccording as each water served by a single steam high and low-press cyl, high, intermediate, and ess cyl.

mmercial classification: ARLY-FITTED pump has a C-I plunger or piston working in a bronze ring, or sleeve; piston rod is of steel, and valve seats, springs, and all inside bolts ts, are of bronze; valves are of rubber or bronze. BRASS-FITTED. Besides onze of regularly-fitted pumps, the rods are of "tobin bronze," piston is solid bronze or bronze-lined, and stuffing-box throats and glands are bronze-lined. In ALL-IRON FITTED pumps, for ammonia or tar, no bronze nor brass comes into contact with the fluid pumped. Valve seats are of malleable iron, springs of steel, the C-I piston works in the solid-bore of the cyl, and plunger nuts are iron. ALL-BRONZE pumps, for acids that attack iron, are entirely of bronze, which in some cases must contain no zinc. LEAD OOD-LINED pumps. Large pumps, that would be too costly if of bronze, made of iron, with the whole interior lined with an acid-resisting material, , wood, or cement. By LARGE OPENINGS is meant that the valve seats are tructed by ribs. Ball or clack valves are of this type.



19. Outside, Center-packed Plunger Pump

ecial-fitted" pumps for different liquids:

r fitted"	Liquid pumped	"How fitted"	Liquid pumped
.....	Ammonia	Regularly-fitted.....	Calcium brine
.....	Bichloride of mercury	" "	Calcium chlorate
.....	Calcium chloride	" "	Citric acid
.....	Caustic soda	" "	Cyanide of potassium
tted.....	Brine	Lead-lined, no zinc...	Hydrochloric acid
"	Copper liquors	" " " " ...	Hydrosulphite acid
"	Glycerine; oils	All-bronze, or lead or wood-lined C-I, no zinc	Sulphuric acid
" (large ngs)	Sewage		

ıp size designations: For direct-acting pumps, 8 & 5 × 12, means 8-in cyl, 5-in water cyl, 12-in stroke. For pumping engines, 5 000 000 gal per eans that the pump's normal capacity is 5 000 000 gal in 24 hr. For a ugal pump, 6-in, single, 2 or more stages, means that the discharge pipe is am and that there are 1, 2, or more impellers.

21. Reciprocating Pumps, Design and Construction (17, 19)

Head. — Total head pumped against = $h_s + h_d + [(v_d^2 - v_s^2) \div 2g] + d$ where h_s = suction head, ft, measured by gage on suction side of pump; h_d = disch head, ft, measured by gage on column pipe; v_s = veloc, ft per sec, in suction at gage point; v_d = veloc, ft per sec, in discharge pipe at gage point; d = dist between gage centers, ft. SUCTION HEAD h_s = pipe entrance head lost + $v_s^2 \div$ pipe friction (including bends) + suction lift. DISCHARGE HEAD h_d = static head a pump + friction head in pipe line (including head lost in valves, elbows, etc).

Suction head. Suction action of a pump is produced by atmos pres surface of water in the pump well or sump, which forces the water into the p chambers after the plunger has created a void in them. Theoretical w column sustained by atmosphere at sea level is 33.9 ft, but this is not obta because under rea press water boils at than 212° F, and tains about 2% si solution, which is d ated. The theore value is also red by air leakage friction, and water locity. Fig 20 s suction lifts at var temps, based on a s suction pipe and bend, at a water v of 3 to 4 ft per Long suctions, or b and foot valves other obstructions in the pipe, reduce the actual lift. Fig 20 shows that if water is hotter than 160° F, it must flow to the pump under press. T liquids must also flow to pump under press.

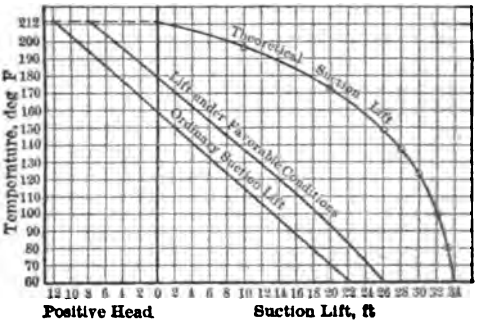


Fig 20. Suction Diagram

Air chamber, generally of C I, is attached to the discharge pipe. When on the tion pipe it is called a VACUUM CHAMBER. It is to provide an elastic element in the line, taking up shocks and inducing a uniform flow. Air chambers are essential in the single-acting, duplex single-acting, and crank-and-flywheel pumps, but may be omitted in other types. Volume of the air chamber should be 6 to 8 times the displacement simplex direct-acting or crank-and-flywheel pumps and 3 or 4 times in duplex pumps. When air chambers are used, they should have a gage glass and means provided to charge them with air when the water in them rises above the proper height. Vacuum chambers are placed on the dead end of a tee REPLACING AN ELBOW in the suction line. Air liberated from water during the suction period enters the vacuum chamber until it has displaced all the water; air will then enter the pump as if there were no vacuum chamber.

Cavitation. At excessive piston speed, the water is drawn from suction pipe faster than atmos pressure can force the water into it. Hence, a break occurs in the water column, known as CAVITATION. This may also be caused by liberation of air and vapor from water under a high suction head. Cavitation is more apt to occur at high temp, high lifts and high suction heads than with low suction and low temp.

Capacity and speed. Relation between capacity and cyl diam of pumps is given by $G(1 + \text{slip}) = 0.0408 d^2 P$

where G = gal per min for 1 double-acting plunger; d = diam of plunger, in; P = piston speed, ft per min. This is for a simplex, double-acting pump. For single-acting pump divide the right-hand member by 2, and for a duplex double-acting pump multiply it by 2. As the duplex pumps usually do not make

ıke, the formula applies only to the rated stroke. Actual stroke = 0.7 — 0.9 ated stroke. An allowance is necessary for the volume occupied by the pis- rod. It is often made by increasing the slip (see below) by 1 or 2%.

Pump efficiency is $E_p = \dfrac{\text{Wt of water pumped} \times \text{total head}}{\text{Ihp of steam end}}$ (3)

c depends upon piston speed and stroke; its value is shown in Fig 21, plotted inst the length of stroke. Speeds generally used with different strokes are shown.

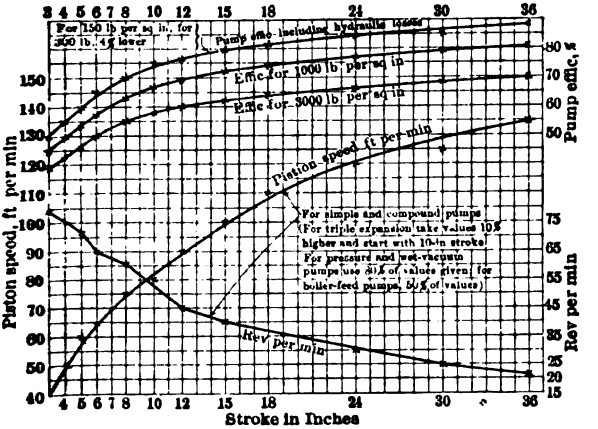

For 150 lb per sq in. for 300 lb 4 to 1 lever
Pump effic. including hydraulic losses
Effic for 1000 lb per sq in
Effic for 3000 lb per sq in
Piston speed ft per min
For simple and compound pumps (For triple expansion take values 10% higher and start with 10-in stroke)
For pressure and wet-vacuum pumps use 80% of values given for boiler-feed pumps, 60% of values)
Rev per min
Piston speed, ft per min
Pump effic, %
Rev per min
Stroke in Inches

Fig 21. Speed and Efficiency of Pumps

ote. — Data on which Fig 21 is based are from tests of commercial pumps. Piston d depends upon length of stroke and number of strokes per min. For boiler-feed pumps, taken as 0.5 that of ordinary service pumps, to allow for sudden capacity demand. triple-expansion pumps, it may be taken 10% higher, because stroke is long and a heel is often used. For pressure and mine pumps, where high water velocity is not rable, and in wet-vacuum pumps, where a high velocity in the system would break water seal at the valves, a piston speed of 80% of ordinary service speed is taken.

Pump slip is the shortage in delivery, due to loose-fitting or worn piston, cyl, valves, and failure of valves to close promptly as stroke reverses. Expressed % of piston displacement, and based on Eq 2:

$$\text{Slip} = \left(1 - \frac{\text{Gal per min delivered}}{[0.0408(d^2 - d_1^2) + 0.0408\,d^2]P}\right) \times 100 \qquad (4)$$

re d = plunger diam, in; d_1 = piston rod diam, in; P = piston speed, ft per . Slip is 2% or less in new pumps, with water reasonably free from air, and arge pumps may be even 0.5%. For calculations it is often assumed as 3%. nps showing 5–10% slip are in poor condition and should be overhauled.

Water velocity in the suction pipe of reciprocating pumps should not exceed ft per min; in valve seats, 220 ft per min; in discharge pipe, 300 ft per min. ; = gal per min, pipe diam is found as follows:

$$\text{suction diam, in} = \sqrt{0.1\,G}; \quad \text{discharge diam, in} = \sqrt{0.08\,G} \qquad (5, 6)$$

alve construction (17). For low heads, and where tightness is essential, e are of soft rubber; for heads up to 75 lb per sq in, medium hard rubber; up to 1

medium hard rubber or bronze; up to 300 lb per sq in, leather-face, conical-wing or flat seated metal valves are used. Hot water requires special rubber composition and for heads of 300 to 3 000 lb per sq in, conical-seated bronze valves are used. Thickness of a rubber valve depends on the unsupported opening and Eq 7 serves as a guide.

$$t = 0.05\, s \sqrt{p} \qquad (7)$$

where t = thickness, in; s = greatest unsupported span, in; p = water press, lb per sq in

Valve-spring pressure depends upon water press. Approximate rules are:

(*a*) Discharge valves. Spring pressure, lb per sq in on valve area = 0.5 to 1% of the discharge press; max, 5 lb per sq in.

(*b*) Suction valves. Spring pressure on valve area = 0.25 to 0.5 lb per sq in, if the water is lifted by suction; for pressure suction, 0.5 to 1 lb per sq in.

Valve area. If A_v = valve-grid opening area, sq in; V_v = veloc through valves (= 220 ft per min); A_p = piston area, sq in; V_p = piston velocity (= 100 ft per min

$$Q = A_v V_v = A_p V_p; \quad A_v = 0.45\, A_p \qquad (8)$$

that is, valve-grid opening area is 45% of piston area for general service. 50 and 60% valve area is often demanded in water works specifications, and pays for itself in good service and durability. (Above velocities are assumed.) Largest size valve for good service is considered to be 4.5 in. Clear opening of valve seat will be 50 or 60% of disk area.

Valve lift. Assume discharge head = 200 lb per sq in, water end 12 × 18, with 4-in valves and 45% valve area.

$$\text{Number of valves} = \frac{\text{Required valve area}}{(\pi 4^2) \div 4 \times 0.6} = \frac{A_v}{7.52} = \frac{0.45 \times (\pi 12^2) \div 4}{7.52} = 7 \text{ (approx)}.$$

From valve-spring pressure rules (above), the spring tension = 0.005 × 200 = 1 lb per sq in, which equals 2.3 ft, and $v = 8.02\sqrt{2.3} = 12.2$ ft per sec = 729 ft per min. Then from Eq 8.

$$\text{Area valve opening} = (V_p + V_v)\, A_p = (100 + 729)\, 113 = 15.5 \text{ sq in.}$$

Total periphery of seven 4-in valves is $7 \times \pi 4 = 88$ in, and lift = 15.5 ÷ 88 = 0.176 in (say 3/16 in); and allowing for a coeff of contraction of 0.5, the lift will be 3/8 in. Hence spring load on the 4-in valve, when raised 3/8 in, is [(π × 4²) + 4] × 1 = 12.6 lb.

Pressure-pump valves should be of metal, with conical seating surface, which can be ground to a perfect seat more readily than a flat surface. But, to pass same volume of water, they must lift 45% higher than flat valves. Pressure valves should be guided to a perfect seat. One method is to have on the under side of valve disk 3 or 4 guide wings. For impure mine water, a flat-seat, leather-faced wing valve is used.

Structural formulas for water end: $d \times p = 2\, t \times s$; t (for cyl heads) $= 0.1\, d \sqrt{\frac{p}{200}}$

$B = d + 2\,(b + \frac{1}{4}$ in); $f = b + \frac{1}{4}$ in; $D = B + 2\,(b + \frac{1}{4}$ in), where B = diam of bolt circle, in; b = diam of bolt or stud, in; D = diam of flange, in; d = diam of cyl, in; f = minimum thickness of flange, in; p = internal press, lb per sq in; s = allowable stress of metal, lb per sq in; t = thickness of metal, in.

22. Pump Tables, Duty, and Steam Consumption (19)

Water head vs steam pressure. The head under which a pump will work depends upon the relation, $P_s A_s \times E_p = P_w A_w$; where P_s = steam press, lb per sq in; A_s = steam piston area, sq in; E_p = pump effic (see Fig 21); P_w = water press, lb per sq in; A_w = water piston area, sq in.

By transpositions of this formula can be found the proportions of steam and water cylinders for different steam pressures and heads. When steam press is increased for pumping under a greater head, the required structural strength of cyl and cyl heads must be determined. For a given vol of water, the dimensions of water end depend on piston speed and head pumped against. Thus, Tables 9, 10, 12, 13 show several different water and steam cyl for the same length of stroke. These tables are of end

ntially the same as built by a number of makers. Larger sizes are obtainable on spe-ation, or, when very large, are built to order.

Table 9. Cameron Simplex Piston Pump for General Service

iam eam l, in	Diam water cyl, in	Stroke, in	Gal per stroke	Capacity at ordinary speed, gal per min	Steam pipe, in	Discharge pipe, in	Floor space, in	Wt, lb	Price, iron water cyl, steel piston rod	Price, with water-cyl lining, piston and rod of composition
3.5	2	4	0.054	8	⅜	1.	32×9	136	$ 80	$ 85
4	2	6	0.081	12	⅜	1	40×10	210	120	125
5	2.5	6	0.12	18	½	1.25	40×11	260	140	150
6	3	7	0.21	28	¾	1.5	47×13	418	165	175
6	3.5	7	0.29	38	¾	2	47×15	435	185	200
7	3.5	7	0.29	38	¾	2	47×15	459	210	225
7	4	7	0.39	50	¾	2	51×16	457	235	250
7	3.5	12	0.5	50	1	2.5	58×17	820	275	290
7	5	13	1.10	100	1	3	63×20	1 117	325	350
8	4	12	0.65	65	1	2.5	58×18	864	325	340
8	5	13	1.10	100	1	3	63×20	1 160	350	375
0	5	13	1.10	100	1¼	3	64×21	1 345	375	400
0	6	13	1.59	150	1¼	3.5	64×21	1 411	400	425
2	7	13	2.16	200	1½	4	66×24	1 928	470	520
4	8	13	2.83	261	2	5	73×26	2 548	540	575
4	9	18	4.96	330	2	5	81×30	3 126	....	.....
6	10.5	18	6.75	450	2½	6	90×37	4 920	....	.....
8	12	20	9.80	587	3	8	103×41	6 080	....	.....

he above pumps are for filling tanks and other light duty.

Table 10. Deane Compound Duplex Pump
(Outside end-packed, double-plunger, pot-valve pattern)

am -p l, n	Diam l-p cyl, in	Diam water cyl, in	Stroke, in	Gal per stroke, 1 plunger	Strokes per min, 1 plunger	Gal per min displacement of 4 plungers	Steam pipe, in	Discharge pipe, in	*L'gth, ft and in	Width, ft and in
0	16	8½	18	4.42	40 to 80	354 to 708	2	6	18 11	4 0
2	18½	8½	18	4.42	40 to 80	354 to 708	2½	6	18 11	4 0
4	20	8½	18	4.42	40 to 80	354 to 708	3	6	18 11	4 6
6	24	8½	18	4.42	40 to 80	354 to 708	3½	6	19 5	6 0
0	16	9	18	4.96	40 to 80	397 to 794	2	6	18 11	4 5
2	18½	9	18	4.96	40 to 80	397 to 794	2½	6	18 11	4 5
4	20	9	18	4.96	40 to 80	397 to 794	3	6	18 11	4 6
6	24	9	18	4.96	40 to 80	397 to 794	3½	6	19 5	6 10
0	30	9	19	4.96	40 to 80	397 to 794	5	6	19 5	6 10
2	18½	10	18	6.12	40 to 80	489 to 978	2½	8	20 0	6 10
4	20	10	18	6.12	40 to 80	489 to 978	3	8	20 0	6 10
6	24	10	18	6.12	40 to 80	489 to 978	3½	8	20 6	6 10
0	30	10	18	6.12	40 to 80	489 to 978	5	8	20 6	6 10
6	24	12	18	8.81	40 to 80	705 to 1410	3½	10	21 6	6 10
0	30	12	18	8.81	40 to 80	705 to 1410	5	10	21 6	6 10

Dimensions are figured with plungers in extreme outward position.
hese pumps will stand 350 lb water press.

Table 11. Blake-Knowles, Simplex Horiz Boiler-Feed Piston Pumps (For 250 lb max working steam and wat

Size, in			Gal per stroke	Max capac per min			Boiler hp pump will feed at slow speed	Steam pipe, in	D ch p
Diam steam cyl	Diam water cyl	Stroke		No single strokes	Piston speed, ft	Gal			
3½	2¼	4	0.068	150	50	10	45	¾	
4½	2¾	6	0.15	130	65	20	70	¾	
5½	3¼	7	0.25	124	72	31	100	1	
6½	4⅛	8	0.46	118	78	54	160	1¼	
7½	4½	10	0.69	108	90	75	300	1	
8	5	12	1.02	100	100	102	500	1	
10	6	12	1.47	100	100	147	750	1¼	
12	7	12	2.00	100	100	200	1 000	1½	
14	8	12	2.61	100	100	261	1 300	2	
16	10	18	6.12	67	100	410	3 500	2½	

Table 12. Cameron Vertical Plunger Sinking Pu

Diam steam cyl, in	Diam plunger, in	Stroke, in	Capacity per stroke, gal	Capacity at ordin'y speed, gal per min	Steam pipe, in	Discharge pipe, in	Space occupied in shaft, in
7	3½	12	0.85	50	1	2	12X18
8	4	12	0.65	65	1	2½	21X21
10	5	13	1.10	100	1¼	3	21X30
12	5	13	1.10	100	1½	3	21X33
10	6	13	1.58	150	1¼	3½	21X30
12	6	13	1.58	150	1½	3½	21X33
14	6	13	1.58	150	2	3½	21X36
12	7	13	2.16	200	1½	4	21X33
14	7	13	2.16	200	2	4	21X36
16	7	16	2.66	200	2½	4	21X40
12	7	16	2.66	200	2	4	21X33
14	8	13	2.83	261	2	5	21X36
16	8	16	3.48	261	2½	5	21X40
16	9	16	4.40	330	2½	5	21X40
16	9	18	4.40	330	3	5	21X40

Priming and operation. To facilitate starting, steam should be pu
the pump with water and to prevent it from acting as an inspirator
the discharge pipe filled, there is no danger of knocking out the head
pump up. If the suction pipe has a foot valve, the pipe may be
steam to fill it and the pump chambers, and drive out the air through
or ¾ the size of the discharge pipe plus 0.5 in. to
chamber. If there is no foot valve, each chamber
a pressure supply. Another mode of priming is to
valve provided with a stop. Check valve is
stroke and closes on suction stroke. The
chamber. Drain cleaning opening to the cyl.
and bar steam end.

Operation. The Test Committee of
" means the lift of

water pumped × head) per million B t u supplied. The definition of duty viously used by pump makers, who naturally wish to make the best show- for their product, was the ft-lb of work per 1 000 lb dry steam. The duty nputed according to the committee's recommendation may give results as ch as 8% lower than on the old basis. Hence, the definition of duty should specified when using the term.

Table 13. Jeanesville Duplex, Pot-valve, Solid-end Pump
(Outside end-packed plungers)

iam eam l. in	Diam plung-ers, in	Stroke, in	Gal per rev	Max rev per min	Total capac	Steam pipe, in	Dis-charge pipe, in	Approx l'gth, ft and in	Approx width, ft and in	These pumps are good for lifts up to 900 ft.
10	4	12	2.63	40	105	2½	3	11 0	2 10	
12	4	12	2.63	40	105	2½	3	11 0	2 10	
12	5	12	4.14	40	165	2½	4	11 2	3 10	
14	5	12	4.14	40	165	3	4	11 2	3 10	
14	6	12	5.9	40	235	3	5	11 4	4 4	
16	6	12	5.9	40	235	3	5	11 4	4 4	
18	6	12	5.9	40	235	3	5	11 4	4 4	
16	6	18	8.8	33.3	295	3	5	15 0	4 4	
18	6	18	8.8	33.3	295	3	5	15 0	4 4	
18	7	18	12.0	33.3	400	3	6	15 6	4 6	
20	7	18	12.0	33.3	400	4	6	15 6	4 6	
20	8	18	15.6	33.3	525	4	6	15 6	4 8	
22	8	18	15.6	33.3	525	4	6	15 6	4 10	
24	8	18	15.6	33.3	525	4	6	16 0	4 10	

le 14. Duty and Steam Consumption of Direct-acting Pumps (Figures correct only for effics given. For other effics reduce figures proportionately.)

Simple, Non-condensing Steam Cylinder
Non-jacketed, but lagged; wire-drawing = 4.7 lb; back press = 16 lb per sq in

oiler pressure	50	70	90	100	110	120	150
bsolute initial press	60	80	100	110	120	130	160
e p	44	64	84	94	104	114	144
ard duty, million ft-lb	45	50.5	53.5	55	56	57	58.5

oke, in	Mech effic, %	Steam effic, %	Total effic, %	Actual duty, million ft-lb per 1 000 lb dry steam = upper fig. Lb dry steam used per water hp per hr = lower fig						
4	55.0	37.5	21	9.5 208	10.6 187	11.3 175	11.6 171	11.8 168	12.0 165	12.3 161
6	65.0	40.0	26	11.7 169	13.1 151	13.9 143	14.3 139	14.6 136	14.8 134	15.2 130
8	70.0	42.5	30	13.5 147	15.2 130	16.1 123	16.5 120	16.8 118	17.1 116	17.6 113
10	75.0	45.0	34	15.5 128	17.2 115	18.2 109	18.7 106	19.1 104	19.4 102	19.9 100
12	77.5	47.5	37	16.6 119	18.7 106	19.8 100	20.4 97	20.7 96	21.0 94	21.7 91
15	80.0	50.0	40	18.0 110	20.2 98	21.5 92	22.0 90	22.5 88	23.0 86	23.5 84
18	82.5	52.5	43	19.4 102	21.7 91	23.0 86	23.7 84	24.0 83	24.5 81	25.2 79
24	85.0	55.0	47	21.0 94.0	23.7 83.5	25 1 79	25.9 76	26.5 75	26.9 74	27.5 72

Table 14. Duty and Steam Consumption of Pumps (*Continued*)

Compound, Non-condensing Steam Cyl
Non-jacketed, but lagged; wire-drawing = 4.7 lb; back press = 16 lb per sq in

Boiler pressure	50	70	90	100	110	120	130
Absolute initial press	60	80	100	110	120	130	140
Ratio of cyls	1.94	2.24	2.5	2.62	2.74	2.85	2.9
M e p on area of h-p cyl	58.0	88.5	120.0	136.0	152.0	168.8	185.0
Card duty, million ft-lb	60.0	69.5	76.5	79.5	82.0	84.0	85.0

Stroke, in	Mech effic, %	Steam effic, %	Total effic, %	Actual duty, million ft-lb per 1 000 lb dry steam = upper fig. Lb dry steam per water hp per hr = lower fig						
6	65.0	40.0	26	15.6	18.1	19.9	20.7	21.4	21.8	22.1
				127	110	99	95	92	91	90
8	70.0	42.0	30	18.0	20.8	22.9	23.8	24.6	25.2	26.7
				110	95	86	83	80	78	74
10	75.0	45.0	34	20.4	23.6	26.0	27.0	27.9	28.5	30.3
				97	84	76	74	71	69	65
12	77.5	47.5	37	22.2	25.7	28.3	29.4	30.4	31.1	33.7
				89	77	70	67	65	64	60
15	80.0	50.0	40	24.0	27.8	30.6	31.8	32.8	33.6	35.6
				83	71	65	62	60	59	57
18	82.5	52.5	43	25.8	29.9	32.9	34.2	35.3	36.1	38.3
				77	66	60	58	56	55	52
24	85.0	55.0	47	28.2	32.6	36.0	37.4	38.4	39.5	41.4
				70	61	55	53	52	50	48
36	87.5	57.5	50	30.0	34.0	38.2	39.7	41.0	42.0	44.5
				66	58	52	50	48	47	45

Compound, Condensing Steam Cyl
L-p cyl jacketed and lagged; wire-drawing = 4.7 lb; back press = 6 lb per sq in

Boiler pressure	70	90	100	120	150	170	190
Absolute initial press	80	100	110	130	160	180	200
Ratio of cyls	3.65	4	4	4	4	4	4
M e p on area of h-p cyl	116.2	151	168.5	203.5	256	291	326
Card duty, million ft-lb	91.0	96.5	98	101.5	104.5	106.5	107

Stroke, in	Mech effic, %	Steam effic, %	Total effic, %	Actual duty, million ft-lb per 1 000 lb dry steam = upper fig. Lb dry steam per water hp per hr = lower fig						
10	75	55	41	37.4	39.6	40.2	41.6	42.9	43.7	44.7
				53	50	49	48	46	45	43
12	77.5	57.5	45	41.0	43.4	44.1	45.5	47.1	48.0	48.9
				48	45	45	44	42	42	42
15	80.0	60.0	48	43.7	46.4	47.0	48.7	50.1	51.1	51.3
				45	43	42	41	40	39	38
18	82.5	62.5	52	47.4	50.0	51.0	52.8	54.3	55.4	55.9
				42	40	39	37	37	36	35
24	85.0	65.0	55	50.0	53.0	54.0	55.9	57.6	58.6	59.2
				40	37	37	35	35	34	33
36	87.5	67.5	59	53.8	57.0	58.0	60.0	61.7	62.8	63.1
				37	35	34	33	32	32	31
48	90.0	70.0	63	57.2	60.8	61.9	64.0	65.8	67.1	67.4
				35	33	32	31	30	30	30

Triple-expansion, Condensing Steam Cyl. Jacketed and lagged; wire-drawing = lb; back press, 5 lb per sq in; no reheater; no cutoff; cross-exhaust valves open

oiler pressure	100	110	120	150	170	190	200
atio $\frac{L-p}{I-p} = \frac{I-p}{H-p} = R$	2.8	2.88	2.96	3	3	3	3
atio $\frac{L-p}{H-p} = R_1$	7.85	8.32	8.78	9	9	9	9
e p on area of h-p cyl	212.2	235.2	258.2	328.3	375	421.7	445
ird duty, million ft-lb	123.6	126.1	128.5	134	136.8	139.0	140.2

roke, in	Mech effic, %	Steam effic, %	Total effic, %	Actual duty, million ft-lb per 1 000 lb dry steam = upper fig. Lb dry steam per water hp per hr = lower fig						
10	75.0	67.5	50.5	62.4	63.7	64.9	67.7	69.1	70.2	70.8
				32	31	30	29	29	28	28
12	77.5	70.0	54.0	66.7	68.1	69.4	72.4	73.9	75.1	75.7
				30	29	29	27	27	26	26
15	80.0	72.5	58.0	71.7	73.1	74.5	77.7	79.4	80.6	81.3
				28	27	27	25	25	25	25
18	82.5	75.0	62.0	76.6	78.2	79.7	83.1	84.8	86.2	86.9
				26	25	25	24	24	23	23
24	85.0	77.5	66.0	81.6	83.2	84.8	88.4	90.3	91.7	92.5
				24	24	23	23	22	22	21
36	87.5	80.0	70.0	86.5	88.3	90.0	93.8	95.8	97.3	98.1
				23	23	22	21	21	20	20
48	90.0	82.5	74.0	91.5	93.3	95.1	99.2	101.2	102.9	103.7
				22	21	21	20	19	19	19

Triple-expansion Condensing Steam Cyl. Normal speed, cross-exhaust valves used; jacketed and lagged; back-press = 4 lb; no reheaters, no cutoff; first intermediate space = vol h-p cyl; second = 0.75 intermediate-press cyl

iler pressure	100	110	120	150	170	190	200
atio $\frac{L-p}{I-p} = \frac{I-p}{H-p} = R$	3	3	3	3	3	3	3
atio $\frac{L-p}{H-p} = R_1$	9	9	9	9	9	9	9
e p on area of h-p cyl	248	276	303	384	438	492	519
rd duty, million ft-lb	144	147.2	150	156.4	159.6	162	163

oke, n	Mech effic, %	Steam effic, %	Total effic, %	Actual duty, million ft-lb per 1 000 lb steam = upper fig. Lb dry steam per water hp per hr = lower fig						
10	75	67.5	50.5	72.7	74.3	75.7	79.0	80.6	81.8	82.3
				27	27	26	25	25	24	24
12	77.5	70.0	54.0	77.8	79.5	81.0	84.5	86.2	87.5	88.0
				25	25	25	23	23	23	23
15	80.0	72.5	58.0	83.5	85.4	87.0	90.8	92.6	94.0	94.5
				24	23	23	22	21	21	21
18	82.5	75.0	62.0	89.3	91.3	93.0	97.0	99.0	100.4	101.0
				22	22	21	20	20	20	20
24	85.0	77.5	66.0	95.0	97.2	99.0	103.3	105.0	106.9	107.6
				21	20	20	19	19	19	19
36	87.5	80.0	70.0	100.8	103	105.0	109.5	111.7	113.4	114.1
				20	19	19	18	18	17	17
48	90.0	82.5	94.0	106.6	109	111.0	115.8	118.1	119.9	120.6
				19	18	18	17	17	16	16

For compound condensing steam end, cyl non-jacketed but lagged, with 6 lb bac press, the card duty is the same as for compound condensing cyl. Steam effic is 5 lower than above, and total effic is 4% lower. That is, the actual duty is found by tak the card duty above (for example, 91.0 for 70 lb press, and 12-in stroke) and multiply it by total effic less 4% (45 − 4 = 41 for 70 lb press, and 12-in stroke). Hence, du = 91 × 41 = 37.4. Steam consumption in this case is found by dividing the steam co sumption (for example, 48 lb for 70 lb press and 12-in stroke) by the new total effic and multiplying by the total effic as above, 45. Hence, 48 × 45 ÷ 41 = 53 lb.

Steam effic, as used in the tables, accounts for press lost in wire drawing, steam lo by leakage condensation, etc. Slow-speed, triple-expansion steam ends, equipped a outlined above, give duties of 12 to 15 million ft-lb per 1 000 lb dry steam less than stat in the table. Tables are for pumps in good working condition and operating at norma length of stroke.

Cost of reciprocating pumps. Some costs of general service pumps, sinkin pumps and station plunger pumps for mine service, are given in Tables 4, 12 For cost of large or special pumps, apply to makers.

CENTRIFUGAL PUMPS

23. Classification

Turbine pump. The IMPELLER vanes are encircled by stationary distributing vanes discharging into the spiral casing, to convert velocity head into pressure head. In VOLUTE PUMPS, the impeller discharges direct into the casing. Effic of recent designs is improved by a "whirlpool chamber," between impeller and outer spiral casing. Centrifugal pumps comprise SINGLE-STAGE, for low heads, and MULTI-STAGE, for high heads (Art 24). Multi-stage pumps comprise a series of impellers on a common shaft, each having its own casing. Each impeller delivers water with an added increment of head to the following impeller, the last one delivering into the discharge pipe at required head.

If, viewed from the power end of impeller shaft, the rotation is clockwise, the pump is right-hand; if anti-clockwise, it is left-hand.

24. Structural Relations (21, 22, 23)

Capacity, water velocity, and size. Capacity is closely related to size and speed. Water velocity at the discharge flange varies between 8 and 10 ft per sec in American pumps, and 6 to 8 ft per sec in European. By a pipe reducer the velocity is brought down to 3 to 6 ft per sec in the pipe line. With given capacity and above velocities at the discharge flange, the diam of discharge pipe and size of pump are found.

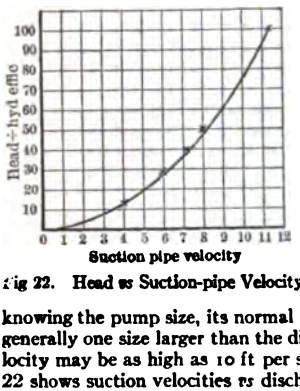


Fig 22. Head vs Suction-pipe Velocity

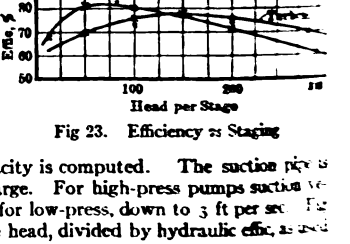


Fig 23. Efficiency vs Staging

knowing the pump size, its normal capacity is computed. The suction pipe is generally one size larger than the discharge. For high-press pumps suction velocity may be as high as 10 ft per sec; for low-press, down to 3 ft per sec. Fig 22 shows suction velocities vs discharge head, divided by hydraulic effic, as used in design work.

Efficiency is a maximum for the normal or rated head and discharge. Fig 23 shows that normal head per stage for volute pumps should be less than 130 ft;

urbine pumps, less than 225 ft. For single-stage pumps, the best effic is l at one speed only. For speeds above and below the best, effic is slightly than the maximum; that is, there is a speed where the sum of water friction earing friction is a minimum (Fig 24). For a homogeneous series of pumps

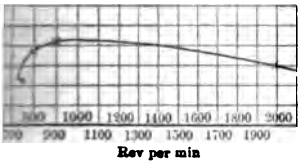


Fig 24. Efficiency *vs* Rev per Min

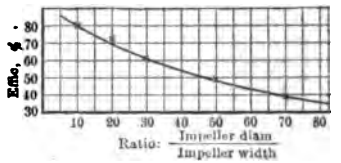


Fig 25. Effic *vs* Ratio of Impeller Diam to Width

re the areas and dimensions of one are proportional to those of any other), fic increases with capacity; that is, the water friction is proportionately less ger pumps.

: greater the speed the higher the effic for a given head and discharge, until the s reached; because the ratio of diam to width of impeller becomes smaller. But, if e carried too far to secure speed, certain desirable features in design must be sacrificed 5). Effic varies with specific (Fig 26). For good effic high heads, the volume de- l should be not less than 75 r min. For STAGED PUMPS, tages of less than 130 ft, the type gives the better effic; stages are greater than 130 e turbine pump is superior 3). For a discharge of less than 600 gal per min, the turbine pump gives the better for larger capacities, the volute (Fig 27).

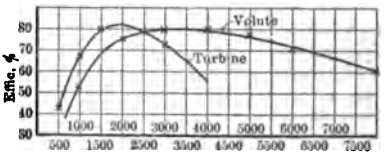


Fig 26. Effic *vs* Specific Speed

above statements refer to the head developed by a single stage, not the total head ulti-stage pump. If the specific speed is unfavorable to good effic, the conditions e improved by using several stages.

ripheral speed of an impeller is approx determined by the head, and in the peripheral speed determines diam and rev per min. Small diam and high rotative speed give a low-priced driver, but the peripheral speed might give a higher head than desired (as in case of a direct-connected steam turbine without velocity or pressure staging). There would also be danger of cavitation in the suction pipe. Suction-pipe velocity should not exceed 12 ft per sec rably 6 to 8 ft). But, the speed must not be so low in proportion to eral speed that the water may slip back through the impeller, as when w-speed engine is direct-connected to a small pump. In good design, eral speed may reach 90 ft per sec, and rev per min may vary from 4 000; but both are sometimes higher. By staging, high heads and iable speeds (commercial speeds of electric motors) may be adopted ut cavitation and vibration.

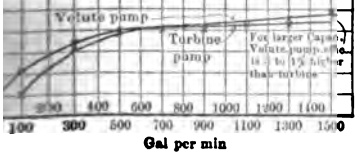

Volute pump
Turbine pump
For larger Capac... Volute pump... is 5 to 15% hi... than turbine
300 400 600 800 1000 1200 1400
100 300 500 700 900 1100 1300 1500
Gal per min

Fig 27. Effic *vs* Capacity

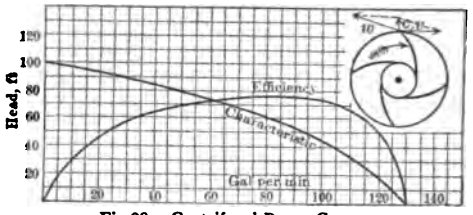


Fig 28. Centrifugal Pump Curves

Pump characteri These are the head-c city curves when th pacity coordinates horiz. Pumps ar signed with falling and rising character Fig 28, 29, 30 giv cal curves for the m shapes shown in t of each cut. In the peller shapes, v = v veloc of tip of vanes

veloc of water relative to that of vane; c = absolute veloc of water leaving in The angle between v and w varies with shape of vane.

Short relations for centrifugal pump calculations. The approx size pump for a given capacity is found by:

$$Q = Ac = (\pi d^2)\, c \div 4 \times 144, \text{ where } c = K\sqrt{2gH}, \text{ and}$$

$$\text{Gal per min} = K_1 d^2 \sqrt{H}, \text{ whence, } d = \sqrt{\text{Gal per min} \div K_1 \sqrt{H}}$$

where A = discharge-flange area, sq ft; c = velocity at discharge flange, ft sec; d = discharge-flange diam, in; K = experimental factor in $c = K\sqrt{}$ $K_1 = (\pi K \sqrt{2g}) \div (4 \times 144 \times 449)$; H = total head pumped against, ft.

Proper pump speed for best economy is determined by the pump size and head; pump size fixes the impeller diam within a small limit and the b with the pump characteristic, fixes the speed. The relation between sp diam, and head is

$$v = \pi DN \div 12 \times 60 = \phi\sqrt{2gH}; \quad \text{whence } N = (\phi\, 1\,840\sqrt{H}) \div D$$

where v = peripheral velocity of impeller, ft per sec; N = number rev per min; D = impeller outlet diam, in; ϕ = experimental factor; H = total head, ft.

The following relations are true as long as ϕ is constant for a homogeneous series of pumps (that is, a series in which each impeller is merely an enlargement or reduction of another having the same proportions): H varies with N^2; rev per min vary with $\sqrt{H}$; gal per min vary with $\sqrt{H}$ and with N; hp in the water pumped varies with $H^{3/2}$ and with N^3. The brake hp input does not vary as $H^{3/2}$ or N^3, because effic varies with pumps of different size of the same series; that is, friction loss in glands and bearings is not a fixed percentage the input for different size pumps.

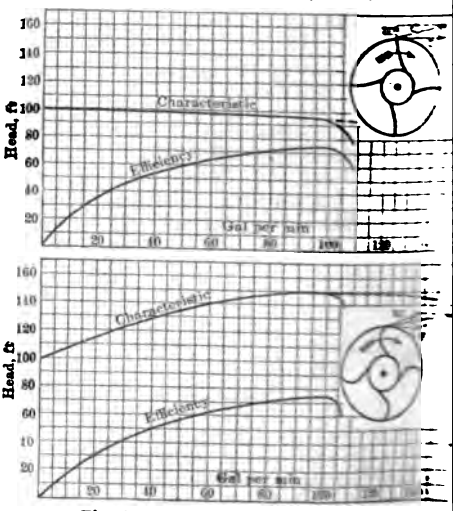


Fig 29, 30. Centrifugal Pump Curves

cific speed, defined as the rev per min at which any impeller of a series would run ere of a diam to discharge 1 gal per min against 1 ft head, is $N_s = (N \sqrt{\text{gal per min}})$ $^{3/4}$. This expression involves the head, gal per min, and speed, which are essential cting a centrifugal pump, and which effect the effic, as shown in Fig 23 to 27. It only practically constant for a single impeller at any speed, but for a whole series of lers of homogeneous design. Thus, specific speed is an index to type; it varies en 500 and 8 000.

25. Performance of Centrifugal Pumps (22, 23)

r single-stage, the usual limit of head is 100 to 150 ft; in rare cases, up o ft or more. Generally, for heads over 150 ft, staging is adopted, and can ntinued up to any head within the limit of strength of pump and pipe. bles 15, 16, furnish examples of approx centrifugal pump performance.

ble 15. Standard Type "*S*" Centrifugal Pumps (Allis-Chalmers M'f'g Co) = pump size (= diam of discharge); *B* = rev per min; *C* = hp of driving motor

Head, ft		20	30	40	50	60	80	100	120	140	160	
er min												
	A	6	6	6	4	4	4	4	4	4	....	
300	*B*	680	860	860	1 150	1 750	1 750	1 750	1 750	1 750	....	
	C	5	10	15	15	20	25	30	30	40	....	
	A	6	6	6	6	6	4	4	4	4	....	
400	*B*	680	860	860	1 150	1 150	1 750	1 750	1 750	1 750	....	
	C	7½	10	15	15	20	25	30	35	40	....	
	A	6	6	6	6	6	4	4	4	6	6	
700	*B*	680	860	1 150	1 150	1 150	1 750	1 750	1 750	1 750	1 750	
	C	7½	10	15	15	20	30	30	40	65	75	
	A	8	6	6	6	6	4	4	4	6	6	
800	*B*	680	860	1 150	1 150	1 150	1 750	1 750	1 750	1 750	1 750	
	C	7½	15	20	20	25	30	35	40	65	75	
	A	8	6	6	6	6	6	6	6	6	6	
900	*B*	680	860	1 150	1 150	1 150	1 750	1 750	1 750	1 750	1 750	
	C	7½	15	15	20	25	40	40	50	65	75	
	A	8	6	6	6	6	6	6	6	6	6	
000	*B*	680	860	1 150	1 150	1 150	1 750	1 750	1 750	1 750	1 750	
	C	7½	15	15	20	25	40	50	50	65	75	
	A	8	8	8	6	6	6	6	6	6	6	
250	*B*	680	860	860	1 150	1 750	1 750	1 750	1 750	1 750	1 750	
	C	10	15	20	25	35	40	50	65	75	85	
	A	8	8	8	8	8	8	8	8	8	8	
500	*B*	860	860	1 160	1 160	1 160	1 750	1 750	1 750	1 750	1 750	
	C	15	20	25	30	35	65	65	85	100	125	
	A	10	10	10	10	10	8	8	8	8	8	
000	*B*	680	860	860	860	860	1 750	1 750	1 750	1 750	1 750	
	C	15	25	30	40	55	65	85	100	125	125	
	A	12	10	10	10	10	10	10	8	8	8	
500	*B*	275	860	860	1 160	1 160	1 160	1 160	1 750	1 750	1 750	
	C	21	25	30	50	50	75	100	125	125	150	
	A	14	12	10	10	10	10	10	10	10	10	
000	*B*	250	325	1 160	1 160	1 160	1 160	1 750	1 750	1 750	1 750	
	C	27	38	40	50	65	85	125	125	150	175	
	A	14	14	12	10	10	10	10	10	10	10	
500	*B*	250	300	350	1 160	1 160	1 160	1 750	1 750	1 750	1 750	
	C	29	48	58	65	75	100	125	150	150	175	
	A	16	16	16	12	12	Use pump		10	10	10	
000	*B*	230	260	300	400	425	of special		1 750	1 750	1 750	
	C	34	47	95	85	95	design		150	175	200	

For higher heads, multi-stage pumps gives best effic

te. — Figures above the stepped line are for 60-cycle motor speeds; those below, for engine speeds. *C* should not be used to find the pump effic, it merely indicates the r commercial size of motor.

Lines of standard side-suction centrifugal pumps are made for heads fr 20 to about 180 ft, and capacities from 30 to 500 gal per min, operating at cycle motor speeds. For heads above 110 ft and capacities below 150 gal min, small multi-stage or 2 pumps in series, are recommended. Boiler-f pumps of from 2 to 6 stages, driven by motor or steam turbine at 1 750 rev min, are made for press of 150 to 250 lb and capac of 200 to 1 250 gal per m

Table 16. Multi-stage Centrifugal Pumps, 60-cycle, Motor-driven (Allis-Chal

Size and capacity, gal per min	No of stages	2-stage	3-stage	4-stage	5-stage	6-stage	Appr pe
3 in, 200– 400 gal	Head, ft	100–180	180–260	260–340	340–440	440–520	
	Rev per min	1 750	1 750	1 750	1 750	1 750	50
4 in, 400– 700 gal	Head, ft	180–220	220–320	320–420	420–520	520–600	
	Rev per min	1 750	1 750	1 750	1 750	1 750	55
5 in, 700– 900 gal	Head, ft	180–220	220–320	320–420	420–520	520–600	
	Rev per min	1 750	1 750	1 750	1 750	1 750	61
6 in, 900–1 200 gal	Head, ft	108–240	240–360	360–460	460–600	.....	
	Rev per min	1 750	1 750	1 750	1 750	.....	65
8 in, 1 200–2 000 gal	Head, ft	180–280	280–420	420–540	.....	.....	
	Rev per min	1 750	1 750	1 750	.....	.....	74
10 in, 2 000–4 000 gal	Head, ft	200–260	260–390	390–520	.....	.....	
	Rev per min	1 160	1 160	1 160	.....	.....	74

Example. To find approximately a pump and motor to deliver 720 gal per against a head of 380 ft. Table 16 shows this capacity to be within the limits of pump, of either 3 or 4 stages, running at a motor speed of 1 750 rev per min. To the hp, use formula: $\text{Hp} = \frac{\text{Gal per min} \times \text{head} \times 0.000253}{\text{Effic}} \times K$, where $K = 1.1$, a const to take care of the small over-capacity which the pumps are designed to give.

$$\text{Substituting, Hp} = \frac{720 \times 380 \times 0.000253}{0.61} \times 1.1 = 125.$$

28. Construction Details and Costs (21)

Casings are usually of C I; for high press, they may be of steel. Liners be used to with acid action or abr Casing is of either s barrel type. Sp ing divides the and bearing housi the horiz plane shaft. This pr ready access to the interior, as the pi nections are in the half of the pum need not be di Barrel casing has movable side pla 31, No 14) at the fo end, so that the im and separators m slipped out of the ing, for disassembl pump. The barrel type is the cheaper to build. Fig 31 is a section of a typical bronze-fitted pump.

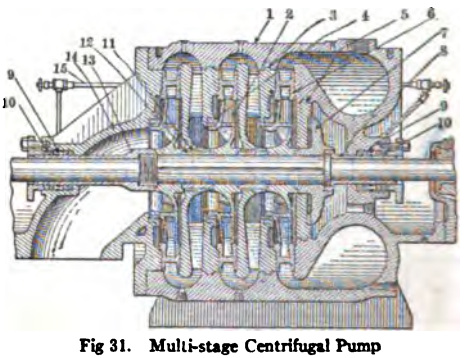


Fig 31. Multi-stage Centrifugal Pump

The bronze wearing ring (Fig 31, No 4) prevents leakage around the runner thro

be a loose-running fit, or fitted to the impeller, with a small clearance between d casing. The clearance must not be too small, else vibration or end play of shaft bring the surfaces in contact. The water-ring seal in the packing (Fig 31, No 9) t- air from leaking into the suction. The flange coupling to motor should be made by boring the bolt holes large and providing rubber bushing.

thrust. Impellers, even when placed back to back, have an unbalanced thrust the suction, which must be resisted. It may be taken up mechanically by a hrust-bearing. The balancing disk (Fig 31, No 7) is of the hydraulic type. High- ater leaking past the shaft presses against the disk and pushes the impellers and o the right. This slightly opens the leakage joint at the disk periphery, allowing of water faster than it leaks past the shaft; thus reducing the press against the d allowing it to move to the left. Then the leakage joint is closed and the cycle es until the disk is in equilibrium. Hydraulic balance is obtained by these means, ails varying in detail in different makes of pump.

r space occupied by a centrifugal pump and its driver is less than that for a cating pump, including its power end, except in sizes having a 3-in discharge and , where the reciprocating pump has the advantage.

ble 17. Over-all Dimensions of Single-stage, Double-suction Gould Centrifugal Pump (without driver)

., discharge, in	2	3	4	5	6	8	10	12	15
space, in.....	28×22	37×26	44×32	44×36	45×42	68×48	72×56	94×65	98×78

18. Over-all Dimensions of Allis-Chalmers Stage Pumps (without driver)

Length in ft and in					Width, all stages, in	Diam discharge, in	Length in ft and in					Width, all stages, in
2-stage	3-stage	4-stage	5-stage	6-stage			2-stage	3-stage	4-stage	5-stage	6-stage	
3 11	4 2	4 5	4 8	5 0	24	8	6 0	6 8	7 4	8 0	8 7	48
4 6	4 10	5 3	5 7	5 11	28	10	7 1	7 9	8 5	9 1	9 9	48
5 5	6 0	6 6	7 0	7 7	36	14	8 5	9 2	9 10	...	...	66

of a centrifugal pump is affected not only by the size or capacity, but head pumped against, speed, number of stages, quality of construction, mmercial conditions.

Table 19. Average Cost of Iron Single-stage Pumps

ty. r	A	B	C	D	Capacity, gal per min	A	B	C	D
	$ 55	$ 80	$150	$200	1 500	$290	$420	600	$700
	90	135	200	250	2 000	340	500	710	790
	150	215	335	400	5 000	660	900	1 190	1 300
	220	325	510	580	10 000	1 300	1 900	.....	.....

— *A* = **low-head** pumps, with iron impellers, single suction, belt-drive, and no **ance guarantee.** *B* = double-suction pumps, belt or direct-drive, but otherwise o *A*. *C* = **pumps** of better construction and heads up to 150 ft. *D* = pumps **good construction,** suitable for direct motor-drive.

of 4-in, single-stage pump. Single-suction: iron, $100; brass impeller **ss sheathed** shaft, $160; bronze ditto, $212. Double suction: iron, **brass impeller** and brass sheathed shaft, $250; bronze ditto, $350.

Cost of 2-stage pumps is about 3 times that of single-stage, for same charge and speed, but double the head. As the number of stages increases cost approaches the number of stages times the cost of a single-stage pump.

For similar conditions of head, speed, and construction, cost varies dir as capacity. Water-supply pumps cost about $1 000 per million gallons capacity (DeLaval Steam Turbine Co).

For pumps suitable for high heads (as for mine drainage), the design is sp and cost varies widely. Examples:

2 500 gal per min	600-ft head	$ 5 000
5 560 " " "	280-ft "	26 000
5 500 " " "	300-ft "	27 800

Cost per lb wt is nearly constant, decreasing slightly as size of pump incre for small iron pumps, about 20¢ per lb; higher for brass and bronze-fitted

27. Installation and Operation (24)

Location. Pumps should be accessible, set on rigid foundations, with base sli above the floor. Care must be taken in setting, not to exceed proper suction he (including losses in suction pipe or fittings). If bolted to foundations instead o grouted in, the base must be in perfect alinement when drawing up on the bolts.

Lining up. In setting the pump, see that its shaft is alined with that of the dri This can be done by putting a steel scale on the coupling flanges. If the machine is up correctly, the scale will rest on both flanges all around the surface. Also see that clearance between the 2 half couplings is equal at all points of the circumfere If bolts instead of grouting are used, make sure after the bolts are tight, that the pu and driver are still in line.

Piping. Piping should have as few bends as possible, and these should be of radius. Introduce no unnecessary bends or other sources of friction in the suction Suction line must be free from air leaks, especially where it is long, or the pump is at height above water level. Avoid air pockets in the suction line; if any exist, pro means to get rid of the air.

Check valve. In multiple-stage and in single-stage high-press pumps, a check v should be placed in the discharge line. This should be put between gate valve and pu Where there is a foot valve and possibility of water hammer, the discharge valve sh be closed before shutting off the power for stopping the pump.

Foot valve. Where the suction lift is not high, a foot valve is often advisable simplifies the priming of the pump. There must be a strainer to keep the foot va becoming choked by foreign substances. A foot valve is not advisable for a pump against a high static head; on shutting off the power, the pump would stop and the water rushing back might close the foot valve before the check valve could producing heavy water hammer. The foot valve should be of flap type, and of size to minimize friction.

Bearings must be cleaned before starting the pump for the first time, as dirt foreign substances may get in during shipment or erection. They should then be with pure, clean mineral oil, which must be changed when it becomes dirty and the bea thoroughly cleaned at same time.

Stuffing boxes and packing. Before starting the pump, the stuffing boxes should carefully cleaned and packed. There must be enough packing back of the water ring so that the inlet water for sealing is brought in at the ring and not at the pa Piping supplying the water seal should fit tightly so that no air can leak in; air entering at this point might cause the pump to lose its suction. If the water is dirty, or gritty, the sealing water must be obtained from some clear source. The valves for the water seal must be open before starting up.

If packing is too tight, it may cause burning of the packing and cutting the s Packing-ring joints should be at angle of 45°, and the joints of the several rings should be staggered. On starting the pump, it is better to have the packing a loose (without causing an air leak); instead of putting too much pressure on

heavy oil in the gland until the pump works properly and then gradually tighten and. Since packing boxes are water sealed, see that the valves are open before ig; a slight leakage of sealing water through the gland is desirable.
in water may be liberated and collect in the pump passages. Hence, the air cocks) of the casing should be opened occasionally; or, if much air, these vents may be artly open during operation, connecting them to drain pipes if desired. If the ty falls off without apparent reason, it is well to open the pump and examine its r and the impeller passages, as pieces of wood and other foreign bodies are sometdrawn into the pump.

ning. Unless the water supply flows to the pump with sufficient head to fill the , the pump must be primed. PRIMING BY EXHAUSTER or ejector is convenient steam or compressed air is available. The discharge pipe must have a tight valve o the pump, to allow the air to be exhausted from casing and suction line. A foot s desirable in connection with a steam exhauster, for priming the first time, or when, reason, the suction line has been emptied. PRIMING BY FOOT VALVE can be done there is a supply of water available under pressure, by merely allowing the suction nd pump casing to fill. All pet cocks should be opened during the filling to allow of air. PRIMING BY VACUUM PUMP. Where steam is not available and it is imable to fill the suction line with water, a hand or power-operated air pump may be A valve must then be placed on the discharge line.
riming multiple-stage pumps, the air must be exhausted thoroughly from each if primed by exhauster, it should be connected to each stage.

ting. The pump casing must be entirely filled before starting; if run empty, arance rings and shaft sleeves, which have very small clearances, will bind, heat, t. When first starting the motor, be sure that its direction of rotation agrees with the pump; pumps must not run in a direction opposite to that for which they are d. This is usually stamped on the casing. After the pump is primed, the shaft be turned over 1 or 2 rev, to allow all air to free itself from the impeller vanes.

WATER TURBINES

ssification. *A*. According to the action of the water: as (*a*) reaction turbines h the water is under press in the runner buckets, the energy being derived by ge the path of the water in passing through the runner, or by the reaction of the n the restricted bucket discharge opening; (*b*) impulse turbines in which the rmation from potential to kinetic energy takes place in the guide apparatus, and ter is received by the runner at the full veloc due to the head. *B*. According to n of flow of the water through the runner: as (*a*) radial flow (outward or Fourneyard or Francis turbine); (*b*) axial flow (Jonval type if reaction, Girard, if impulse); ed flow (American, in which flow is first inward radial and then down parallel); gential flow, where the jets strike cups on the periphery (Pelton and spoon wheels). ording to whether the shaft is horiz or vertical.

28. Reaction Turbines (30, 31)

bols used in following discussion:

uantity of water, cu ft per sec	1 = subscript, indicating exit from guide passage, and entrance to runner
et head, ft	2 = ditto, turbine wheel
orsepower	3 = ditto, entrance to draft tube
v per min	4 = ditto, exit from draft tube
iam of turbine, in	p = suffix, indicating supply pipe
iam of turbine jet, in	a = angle between c_1 and v_1
iic, per cent	d = angle between c_2 and v_2
cel due to gravity, 32.2 ft per sec	f = coeff, indicating loss of press by friction
solute veloc, ft per sec	s = subscript, indicating 1 ft head conditions
lative veloc, ft per sec	
ripheral veloc of runner, ft per sec	
riph veloc of runner + veloc of jet	

ut available. $P = (QH + 8.8) \times E$; when $E = 80\%$, $P = QH + 11$.

Velocity of water (25). HEAD RACE. In low-press plants, as little head as possi should be lost through friction and eddies. Water is led to the turbine in an open cha at a low veloc, never exceeding 2 ft per sec. In the TURBINE PIT or flume, the veloc may be even lower, to avoid eddies on entering the guides. Head-water level must not be less than 1.25 diameters above guide-vane openings; otherwise air may enter as eddies, and destroy suction action in the draft tube. If this rule can not be observ eddies may be reduced by placing wooden floats in the head race just above the turbi TAIL-RACE veloc should be such that the energy lost is less than 1%. This is difficult attain in low-head installations, but it should be considerably less than 1% with h heads. PENSTOCK (if any); with favorable conditions and short pipe line, the veloc may be $c_p = 0.1\sqrt{2gH}$, but with very high heads it is impossible to take ad vantage of this rule. It is fairly safe for heads between 35 and 180 ft, but for low lines the veloc must be reduced to save friction loss. In the distributing pipe near turbine the veloc may be increased to $0.13\sqrt{2gH}$, and in the inlet pipe, close to bine casing, to $0.17\sqrt{2gH}$, or $0.2\sqrt{2gH}$ for low specific speed and spiral-cased turbi but should be less than $0.1\sqrt{2gH}$ for barrel-cased turbines. DRAFT TUBE must be dimensioned that water enters it from wheel without change of veloc. It s be tapered outwards and downwards, as a diffuser for regaining part of the resi kinetic energy, and the outlet veloc should be slightly more than that in the tu (see above). It is inadvisable to use draft tube more than 1 ft in diam t length, otherwise the water column broken. TURBINE GATES. "Gate" part of the turbine which opens and cl to regulate entrance of water in acc with load. In Francis turbines Fig best makers use guide blades swi parallel axes. Each is coupled to moved concentrically about the tur by the governor, thus opening and cl the blades. This gives good effic at gate openings; the cyl gate, placed be guides and runner, gives poor effic load. GUIDE AND RUNNER VANES must enter guides and runners eddies or impact, change its veloc and continuously, move along smooth curves, and leave the tu the draft tube as nearly devoid of ene possible. Observing these rules, the veloc diagrams will be closed triangles Vane angles and sizes of channels must be designed so that the resulting velocities in Fig 33.

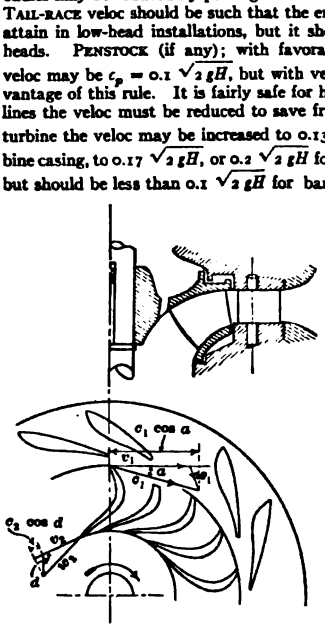


Fig 32. Guide and Runner Vanes (Francis Turbine)

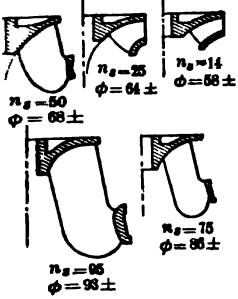
Turbine theory. The energy of the water on entering the turbine runner is $(W + g)\, c_1 \cos a \times v_1$ = available energy in a tangential direction at entrance (entrance diagram, Fig 32). Energy at discharge from the runner is $WH_2 = W(\cdots$ $\cos d \times v_2$ = available energy in a tangential direction at discharge (see discharge dia Fig 33). Hence, the work done is $W(H - H_2) = (W + g)(c_1 \cos a \times v_1 - c_2 \cos d \cdot$ WH_2 representing work lost. Other losses are due to impact at entrance to runner friction, water friction, water leakage, and eddies. These are considered collectiv percentage of the total available energy, and are subtracted from 100% to fi hydraulic effic. Then the above expression reads $gEH = v_1 c_1 \cos a - v_2 c_2 \cos d$. Substituting for $\cos a$ and $\cos d$ their values from the triangle relations for the entrance discharge diagrams (Fig 33),

$$EH = \frac{c_1^2 - c_2^2}{2g} + \frac{v_2^2 - w_1^2}{2g} - \frac{v_1^2 - v_2^2}{2g}.$$

Both forms of the fundamental equation are highly useful.

iciencies. Hydraulic effic of a turbine is the ratio of the energy delivered to r shaft to that of the water at the turbine entrance. It ranges from 80% to 95%, ding upon size and design of turbine. To get the commercial effic, from these s deduct 2 to 5%, for bearing friction and operation of the governor. The r the turbine the larger the mechanical loss. Thus, the over-all effic is 75 to Effic to be expected depends largely on the cost of the turbine. Reputable s guarantee over 80% over-all effic for stock turbines, higher guarantees being for special designs.

eds of Francis turbines. In considering the relation between output and speed, ntity called the SPECIFIC SPEED, n_s, is used, which is the speed at which a wheel run to develop 1 hp under 1 ft head. $n_s = n\sqrt{P} + H^{5/4} = n_x\sqrt{P_x}$, subscript x ng to 1 ft head conditions. n_x and P_x are found from $n_x = n + \sqrt{H}$ and $P_x = P +$



3. Types of Francis Runners

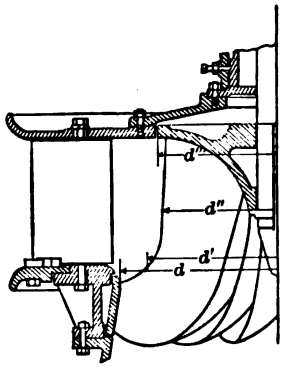


Fig 34. Modes of Measuring Runner Diam

esigning Francis turbines, n_s may be varied from 10 to 100; that is, a turbine can igned to develop 1 hp under a head of 1 ft, and by altering angles and dimensions ain any speed between 10 and 100 rev per min without loss of effic. Francis turbines e grouped according to unit speed into: Group 1, n_s = 10–25 (low reaction); 2, n_s = 25–50; Group 3, n_s = 50–78; Group 4, n_s = 78–100 (high reaction). 1 are for high heads, to about 600 ft. They are narrow axially and work nearly as e wheels. Group 4 are broad axially, are for low heads, and work with a high degree :tion. Fig 33 indicates the difference in design of these groups. This flexibility gn explains why the Francis turbine has superseded other types for reaction work.

ice of speed depends upon the nature of machinery to be driven. In the case generators, the speed must be regulated to suit the frequency. RUNAWAY SPEED. load suddenly ceases, the governor may not operate, and, when running at max the safety of the machinery being driven must be considered. Table 20 gives the ıy speed in percentage of the normal rev per min which may be expected for ıt makes of runner.

ct of head upon speed. From the formula for peripheral speed of a given tur- $r = (\pi\, dn) \div (60 \times 12) = \phi\sqrt{H}$, or $n = \left(1\,840\,\phi\sqrt{H}\right) \div d$. Thus, when ϕ) is constant, the rev per min vary with $\sqrt{H}$. But, commercial work requires a ıt speed; hence, as the head changes, ϕ must change from its best value. That is, ic for a given turbine is obtained at one head only.

meter of the runner is measured at the entrance edge. Fig 34, in con- ɔn with Table 20, gives the practice of different makers. "HAND" of the looking at the turbine from the shaft coupling, the runner is r' f it rotates clockwise; left-hand, if anti-clockwise.

	Maker
	1
I	Allis-Chalmers Co
II	Dayton Globe Iron Works Co
III	Holyoke Mach Co
IV	Rodney Hunt Machine Co
V	Hydraulic Turbine Corpor
VI	J. & W. Jolly Co
VII	James Leffel & Co
VIII	I. P. Morris Co
IX	Pelton Doble Water Wheel Co
X	Platt Iron Works Co
XI	S. Wiggin Smith
XII	Young Mig Co
XIII	Wellman-Seaver-Morgan Co

te to Table 20. — *a* and column 8 are from (29), and *b* from paper No 1 138, Am Soc 1 Eng, by C. W. Larner. Value of ϕ, col 4, is for best effic as given in col 6. Col 5 tained by using the rev per min corresponding to ϕ at best effic, the hp at best reduced thus, $P \times (80 \div$ Best effic), and the actual head upon the turbine. Tur- are never guaranteed to give their best effic, but usually 3 to 8% lower, depending the setting, price, and keenness of competition. The foregoing shows that the table educed the value of n_s for the different turbines to an 80% guarantee basis, and the table is used the turbine selected has a slight margin of power. Where not wise specified, information in the table is from makers' literature, or received from by letter. Where incomplete, no information was obtainable.

ample: to show how turbine selection is made.
ven: $H = 16$; $P = 350$; $Q = 241$ cu ft per sec; current, 3 phase a c, 60 cycles.
rmal head, or difference between head water and tail water levels, is 16 ft. Then, al discharge being known from records, $P = [(241 \times 16) \div 8.8] \times 0.8 = 350$.
owing the rev per min desired (as 138.5 or 180), $n_s = n \div \sqrt{H} = 138.5 \div \sqrt{16} =$ or $n_s = 180 \div \sqrt{16} = 45$. Also, $P_s = P \div H^{3/4} = 350 \div 16^{3/4} = 5.46$; and $n_s = P_s = 34.6 \sqrt{5.46} = 80.7$, or $n_s = 45 \sqrt{5.46} = 105$.
ble 20, column 5, shows that makers number I, II, VII, VIII, X, and XI supply es which have values of n_s approximating 80.7. The I. P. Morris, Type O, is t, and other makers could meet the requirements by slightly modifying their de-

Diam of the Morris turbine is found thus: $\phi = nd \div 1\,840 \sqrt{H}$; whence, $= 138.5\, d \div 1\,840 \sqrt{16}$, and $d = 41.75$ in.
180 rev per min, the Leffel turbine is nearest. If the rev per min were not definitely d upon, the selection might be made thus: with the low head, the turbine would bly be a Francis high-reaction wheel, with $n_s = 78$–100. Then, using the same of H, P, and Q as above, and trying $n_s = 80$:

$$n_s = n \sqrt{P} \div H^{5/4} = n_s \sqrt{P_s}, \text{ and } P_s = P \div H^{3/2}.$$

tuting, $P_s = 350 \div 16 \sqrt{16} = 5.47$, and $n_s = 80 \div \sqrt{5.47} = 34.4$. Hence, $\sqrt{H} = 34.4 \times \sqrt{16} = 137.6$ rev per min. Also, $n_s = 138.5 \times \sqrt{350} \div 16^{5/4} = 80.7$. enerator for this speed would have $60 \times 60 \div 137.6 = 26.2$ pairs of poles (say es), corresponding to a speed of 138.5 rev per min, and a slightly different n_s. Other of n_s could be tried until a turbine was found which suited all conditions.

nmary of short expressions for turbine formulas:

$\sqrt{H} = n_s \div \sqrt{H_s}$;	that is,	n	varies with	$\sqrt{H}$,	when	ϕ	and	D	are	constant.
$D_s = n_s \div D$;	"	"	n	"	"	$1 \div D$,	"	ϕ	"	H	"	"
$\sqrt{H} = Q_s \div \sqrt{H_s}$;	"	"	Q	"	"	$\sqrt{H}$,	"	ϕ	"	D	"	"
$D^2 = Q_s \div D_s^2$;	"	"	Q	"	"	D^2,	"	ϕ	"	H	"	"
$H^{3/2} = P_s \div H_s^{3/2}$;	"	"	P	"	"	$H^{3/2}$,	"	ϕ	"	D	"	"
$D^2 = P_s \div D_s^2$;	"	"	P	"	"	D^2,	"	ϕ	"	H	"	"

$n_s = n \sqrt{P} \div H^{5/4} = n_s \sqrt{P_s}$, and $\phi = nD \div 1\,830 \sqrt{H}$

9. Impulse Wheels

ign is much simpler than that ction turbines. A jet of water the peripheral buckets tangen- to the wheel, with a velocity ponding to the head, less friction nozzle (Fig 35) (27, 28). Thus $\sqrt{2gH}$, where f is about 0.97. of runner = diam of pitch circle ; centers of buckets. Peripheral y is approx $v_1 = 0.45 \sqrt{2gH}$. ghr effic (80% or above), the D of jet or jets (in) must not diam d of the wheel (ft); if

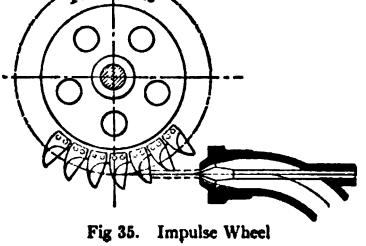


Fig 35. Impulse Wheel

5% lower effic be permitted, it may be up to $D = d + 9$. For high-pres turb with horiz shafts, it is possible (though undesirable) to use 2 or even 3 jets per wh For vertical-shaft wheels, 4 jets may be used. More than 2 wheels should not be f on one shaft. From what precedes,

$$Q = \frac{\pi D^2}{4 \times 144} \times c_1 = \frac{\pi D^2}{4 \times 144} \times 0.97 \sqrt{2gH}.$$

If $D = d + 12$, then $Q = \frac{\pi d^2}{4 \times 144 \times 144} \times 0.97 \times 8.02 \sqrt{H} = 0.000295\, d^2 \sqrt{H}$.

Example. Given: $H = 600$ ft; $P = 580$ hp; $Q = 10.6$ cu ft per sec.

$Q = 10.6 = 0.000295\, d^2 \sqrt{600}$; hence, $d = \sqrt{\frac{10.6}{0.000295 \sqrt{600}}} = 38.4$ in (say, 3 inches). Diam of jet $= 38.5 \div 12 = 3.21$ in. Since $n_1 = 0.45 \sqrt{2gH} = (r$ da (60 × 12), $n = 0.45 \frac{1840 \sqrt{H}}{d} = 525$ rev per min.

30. Automatic Governors for Turbines

This subject can not be dealt with here. For details of modern practice, and consid ations respecting effects of changes of load and pressure, see Bib 25, 27, 28, 29, 30.

31. Turbine Costs (31)

As nearly every installation is special, the price varies widely. Per lb wt of tu price may range from 8 to 30¢, depending upon the efficiency and regulation guara workmanship, metal used, style of setting, and whether or not a stock design is used specially designed turbine is required. Cost of the turbine itself generally varies tween 2 and 6% of total cost of the power plant. Cost of the turbine decreases w increase in head. Following figures are from (31):

Head, ft	Cost per hp	Cost of building per hp	Cost of a 16-in turbine		
			Head, ft	Hp	Cost per hp
60	$30–$7	$30–$4	30	52	$19 30
100– 600	12– 2	7– 2	60	158	6 75
500–2 000	8– 2	7– 2	100	318	3 14

Results of proposals for 4 turbines to give the same power under approx the same co ditions are as follows:

Hp	Rev per min	Head	Specific speed	Wt, lb	Total cost	Cost per lb, ¢	Effic guar	Design	Remarks
550	600	134	31.4	11 500	$1 750	15	None	Stock	26-in double discharge
500	514	138	24.3	....	4 000	....	"	Special	Governor included
550	600	142	28.8	33 000	6 000	18.2	Yes	Modified special	Guarantee
550	600	142	28.8	38 000	8 740	23.0	Yes	New design	Governor included

For other cost data, see Bib (29, 30, 31).

INTERNAL COMBUSTION ENGINES AND GAS PRODUCERS. (See also Sec 39, Art 5)

32. Classification and Characteristics

ommon basis of classification is the type of cycle employed, as the Otto or Diesel. cycle may be completed in either 2 or 4 strokes, resulting in a division into 2- e or 4-stroke cycle engines; abbreviated into 2-cycle and 4-cycle. Another classi- on is: gas engines and liquid-fuel engines, or gas, gasolene, and oil engines. The of governing, igniting the charge, and degree of compression (whether low, medium, gh) is a basis of classification; as is also the cylinder ar- ement (single or double-acting, single or multicylinder, or vertical).

ur-cycle engine indicator card for an Otto cycle is shown g 36. A new charge is drawn into the cyl on the first or on stroke AB, and compressed into the clearance space on econd or compression stroke BC, at or about the end of h ignition occurs and the charge instantly burns, causing press rise CC', at approx dead center. The high-press formed by combustion expand during the third or expan- stroke $C'D$, driving the piston before them; on this stroke is power delivered to the engine, enough being stored in the heel to run the engine until the next power stroke. On the exhaust stroke DA, burned gases are forced out. This type of engine has 2 valves, inlet and exhaust; latter is always mechanically operated, the former may be either mechanically oper- or automatic.

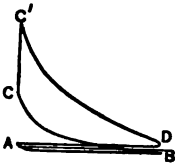
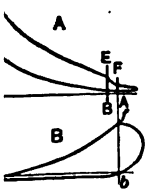


Fig 36. Otto 4-cycle Indicator Card

wo-cycle engine. A power stroke occurs once in each rev, the transfer of the charge out being made during a small part of the expansion and compression strokes. Fig 37 shows cards for this type of Otto engine, A being from the engine cyl, B from the compressor. At C ignition occurs, followed by expansion DE, as in the 4-cycle; at E the exhaust port is uncovered by the piston, causing the press to fall nearly to atmos. A little later, at F, the piston uncovers a second opening (transfer port); or an inlet valve is opened, allowing new mixture previously compressed to 6–10 lb above atmos press to flow into the cyl and drive out the remaining burnt gases. To facilitate this action, and prevent escape of new mixture through the exhaust, the 2 openings to the cyl are as far apart as possible and the piston end has a lip (Fig 38). The transfer port is closed at A (Fig 37), and the exhaust at B, permitting compression during the rest of the stroke. During compression in the engine cyl, suction bc occurs in the compressor cyl, followed by compression cf, while expansion takes place in the engine. In large engines a separate compressor or possibly 2 cyls, 1 for air and 1 for gas), are always provided, but in small es the charge is usually compressed in the crank case by the piston. This type of e is divided into 2-port or 3-port, depending on whether the inlet to the crank case ntrolled by a valve or by a third port, uncovered at the end of crank-case suction by ain piston. Card B, Fig 37, is from the 2-port engine in Fig 38, A. A 3-port engine wn in Fig 38, B.



37. Otto 2-cycle ndicator Cards

2-cycle engine having twice the cycles for the same rev per min gives greater hp e same size cyl, but not twice as great, since the m e p is less than for a 4-cycle, as o the effic. Chief advantages of the 2-cycle are: decreased piston area per hp and smaller wt and cost, decreased flywheel wt due to greater frequency of impulse, and ation of exhaust valves and gear (or, in small sizes, of all valves) hence further de- e in cost.

s and gasolene engines (32, 33, 34, 35). Gas engines may include engines run- on readily vaporized liquid fuels, as benzene, gasolene, and alcohol. To adapt a gas e to these fuels merely requires replacing the MIXING VALVE (which proportions the

quantities of air and gas) by a CARBURETER (see below). Hence, most engines listed as gas engines may be operated on gasolene and to some extent on alcohol. As alcohol permits higher compression, engines using it have somewhat smaller clearance spaces. The higher compression is not necessary, but tends to better economy. At present [illegible] the cost of alcohol in the U S prevents its competition with gasolene, but abroad it is largely used.

Oil engines use oils which can not be vaporized without adding heat. In general such fuels are injected into a projection of the cylinder termed the VAPORIZER, which, having no water jacket, is kept at a high enough temp for vaporizing the oil by the heat of the explosion. Prior to starting the engine the vaporizer is heated by an external source. There are 3 types of oil engine: (a) those injecting the fuel before the end of compression, all operating on Otto cycle and with relatively low compression, as the Hornsby-Akroyd and Mietz and Weiss; (b) those injecting fuel after a high compression and operating on Otto cycle, as the Franchetti; (c) those injecting fuel after compression and operating on Diesel cycle (Sec 39, Art 5). Types (b) and (c) are confined to large sizes and have about equal efficiency which is high due to high compression. They first compress air above the Diesel to press of 400–600 lb. the Franchetti to 200–300 lb. In the Diesel, oil is then injected at a rate required to keep the press constant by combustion, ignition taking place by the high temp of the air due to high compression. In the Franchetti, the proper amount of oil is suddenly blown into the cyl in a finely-atomized form, is ignited by a hot bulb in the cyl head, and burns with explosive rapidity. Both engines can produce 1 brake hp-hr on 0.5 lb of oil, or slightly less.

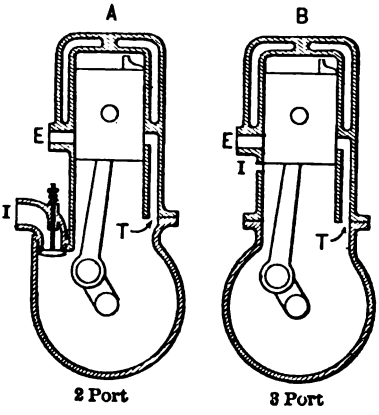


Fig 38. Two-cycle Engines, 2 and 3-port

33. Details of Operation (35, 37)

Governing. There are several modes; not only is the m e p changed, as in steam engines, but the number of cycles also. One of the commonest systems is the HIT AND MISS in which fuel is admitted only when speed is at or below normal, the engine running by the flywheel energy when speed is above normal. The mechanical means employed are numerous, the commonest being to arrange the push-rod operating the inlet-valve stem so that when speed rises it may be pulled to one side by the governor action, thus missing instead of hitting the valve stem. On engines having automatic inlet valves the exhaust may be held open, thus preventing creation of a sufficient vacuum in the cyl to open the inlet, or the inlet may be mechanically held shut. This economical form of governing is confined chiefly to small, cheap engines, where close regulation is not essential. Governing by throttling is usual on large engines; a butterfly valve or other restriction in the mixture pipe is operated by the governor, or the latter varies the lift of the inlet valve. Also, the time of closure of the inlet valve may be altered, giving a cutoff effect. See QUANTITY GOVERNING. Changing the amount of fuel per cycle, or QUALITY GOVERNING, is usual for oil engines, the quantity being varied by changing the length of the pump stroke, or by-passing some of the oil pumped.

Ignition. At present (1916) the hot-tube and flame ignition systems have been superseded by electrical means, except in oil engines where the hot bulb used for vaporizing the fuel also serves for ignition, and, in the Diesel engine, where the temp due to compression is high enough to make separate ignition devices unnecessary. Electric systems comprise the

H-SPARK and the MAKE-AND-BREAK. In the former, the secondary circuit of an in-ion coil is led into the cyl through the insulated terminal of a spark plug. At the er moment the primary circuit of the coil is closed and a spark (or series of sparks in better forms) jumps a short gap in the secondary circuit at the end of the plug. Make-break system has a single low-tension circuit; just prior to the moment of ignition the uit is closed, and then suddenly broken within the cyl at the proper instant, causing ark at the point of breaking. To increase the spark intensity, an inductance coil is ully placed in the circuit. Breaking the circuit may be done mechanically, or in the er forms electrically; the latter is preferable in requiring no moving element to pass ugh the cyl wall. Any source of current, as primary or storage batteries, magnetos, ther generators, may be used for ignition. Frequently, for greater reliability, there 2 sets of ignition apparatus, supplied from different sources.

tarting devices. Large engines are started by admitting compressed air to one or e cyl at the proper time, until a mixture has been drawn into and fired in one of them, reupon the air is shut off. Small engines are started by turning them over by hand l an explosion occurs, or by electric motors. Automobile engines nearly always have tric starting motors, which become generators and supply current for ignition, stor-battery, and lighting, when the engine is in operation.

ooling is accomplished by water jacketing the cyl, except in very small sizes where, dding ribs on the cyl, enough radiating surface is obtained for cooling. In larger s, the exhaust valves, and in double-acting engines, the pistons and rods, are also er-cooled. Cooling water must be clean and not deposit much salt on heating; dirty ard water fills the jackets with mud or scale, causing overheating. Water may be ved to go to waste, or artificially cooled (Art 15). From 3 to 5 gal per hr per hp are cient, if a reasonably cool source is available.

arbureters prepare explosive mixtures from the lighter liquid fuels, as gasolene or hol. Strictly speaking, the term carbureter is limited to apparatus where no external is applied, those requiring heating being called vaporizers (Art 32). But, with the of heavier and heavier grades of fuel, many so-called carbureters are now equipped i means for applying heat, as from the engine-jacket water or exhaust. Carbureters ortion the fuel and air, vaporizing the fuel to some extent and mixing the two. This ne by allowing the fuel to flow from a reservoir, in which the level is kept constant, the air-inlet pipe. As the press in this pipe drops with increase of air flow, more fuel flow. If air and fuel followed the same law of flow with press drop, a carbureter sted for any condition of load or speed would be correct for all conditions. But, the of fuel increases faster than that of air, tending to cause too rich a mixture at heavy s or high speeds. The numerous carbureters on the market differ chiefly in the mode rrecting this defect; most of them use a manual or automatic device, or a combina-of these, for introducing extra air when the conditions normally cause a rich mixture. fuel reservoir is called the float chamber, when the liquid level in it is regulated by a . This device is limited to engines in which the supply is under a gravity or press l. For stationary engines insurance regulations usually forbid its use, in which case mp forces fuel to the reservoir in excess of the engine demand, the surplus returning he tank. The fuel flows from the reservoir through a spray nozzle, controlled by a le valve.

r light fuels the heat in the atmosphere usually suffices for vaporization; for heavier ls, heat is applied by drawing the air partially or wholly from a jacket on the exhaust , or by allowing the engine-jacket water to pass through a jacket on the carbureter. essive heating is bad, as the engine power is proportional to the density of the charge. mplete vaporization, especially in multicyl engines, is apt to cause a non-homogeneous ture. For mixing action, carbureters depend chiefly on the throttle and valve ob-ction, and on bends in the supply pipe (37).

34. Power, Efficiency, and Cost

xpressions for calculating power, thermal effic, and fuel consumption of internal bustion engines are given in Sec 39, Art 5, and Tables 21, 22 show values of com-sion ratio, diagram factor, and effic, for different sizes of engine and kinds of fuel. ir brake hp is the product of i h p and mech effic, values for which are in Table 23.

Table 21. Internal-combustion Engine Compression Pressures

Fuel	Comp press, lb per sq in, abs	Fuel	Comp press lb per sq in, abs
Gasoline	60–120	Coal gas	75–125
Kerosene	40–100*	Natural "	100–175
Alcohol	100–180	Producer "	100–150
Oil injected prior to comp'n	30– 60	Blast furnace "	125–175
" after comp'n	250–280†	Coke oven "	100–175
Carbureted water gas	75–120		

* If water injection is used. † Pranchetti; 500 to 600 Diesel.

Table 22. Diagram Factors and Efficiency of Otto-cycle Engines

Make of engine	Approx comp ratio	Authority	Compression	Diagram factor	Eff.
[illegible]	[illegible]	Hubert	9.2	0.49	22.9
"	[illegible]	"	10.4	0.51	25.5
[illegible]	[illegible]	Witz	8.0	0.61	27.1
[illegible]	[illegible]	Allaire	11.2	0.51	25.7
[illegible]	[illegible]	Mathot	11.6	0.61	30.4
[illegible]	[illegible]	"	11.3	0.71	35.2
[illegible]	[illegible]	"	5.8	0.69	27.4
[illegible]	[illegible]	Burstall	7.0	0.49	21.5

Table 23. Mechanical Efficiency of Gas Engines (Lucke)

Type of engine		4-cycle	2-cycle
[illegible] multiple-cylinder, single-acting		0.75	
[illegible] stationary, single-acting		0.87	
" " "		0.84	
" " "		0.80	
[illegible] single-acting		0.90	[illegible]
" "		0.86	[illegible]
" "		0.84	[illegible]
[illegible]		0.83	[illegible]
[illegible]		0.81	[illegible]
[illegible]		0.77	[illegible]

Table 24. Heat Balance for Internal-combustion Engines

Engine [illegible]	% i h p	% exhaust	% jacket
[illegible]	22.1	35.5	43.2
[illegible]	29.5	36.3	34.2
[illegible]	46.0	24.0	32.0
[illegible]	32.0	41.0	27.0
[illegible]	33.0	37.0	30.0
[illegible]	43.0	24.0	33.0
[illegible]	27.0	23.0	50.0
[illegible]	24.0	51.0	25.0
[illegible]	21.0	29.0	50.0
[illegible]	40.0	30.0	27.0

the effic of internal combustion increases with load, the tendency is to rate such es at nearly their max capacity; while steam engines, though also rated near their of best economy, have large overload capacity. Hence care should be taken to

Table 25. Standard Sizes of Vertical Gas Engines (Nash)

Hp		Rev per min	Diam fly-wheel, in	Wt, lb	No of cyl	Hp		Rev per min	Diam fly-wheel, in	Wt, lb
A	*B*					*A*	*B*			
6	4	350	30	2 500	3	85	60	250	60	24 000
10	6.5	300	48	3 300	3	100	70	250	66	31 000
12	8.5	300	48	3 500	3	125	90	250	72	34 000
20	14	300	48	5 000	3	150	105	250	72	36 000
25	17.5	300	60	5 500	3	185	135	250	72	39 000
30	21	275	60	9 000	4	250	175	250	72	50 000
40	28	275	60	10 500	4	300	210	225	78	75 000
50	35	275	60	14 100	4	425	350	225	84	90 000
65	45	275	60	15 000	....	....	...	....	....	.....

ote. — *A* on city gas, natural gas, gasolene, and distillate; *B* on producer gas.

Table 26. Standard Sizes of Otto Horiz Gas Engines

o of yl	Hp		Rev per min	Diam flywheel, in		Approx floor space, ft	Wt, lb	
	A	*B*		(1)	(2)		(1)	(2)
	4	.....	340	16 or	46	5 by 2	1 800 or	2 100
	6	.....	340	16 "	46	5 " 2	1 800 "	2 300
	8	.....	300	22 "	52	6 " 3	2 400 "	3 500
	10	.....	300	22 "	52	7 " 3	2 400 "	3 700
	12	.....	260	30 "	64	8 " 3	3 500 "	5 800
	15	.....	260	32 "	64	8 " 4	3 800 "	6 000
	21	.....	240	40 "	68	8 " 5	5 100 "	9 000
	25	.....	240	40 "	72	9 " 5	6 600 "	10 500
	30	.....	240	44 "	78	10 " 5	8 000 "	12 000
	40	30	240	48 "	80	11 " 5	9 800 "	14 500
	50	40	230	54 "	86	12 " 7	13 000 "	21 500
	60	50	220	58 "	96	12 " 7	16 000 "	26 000
	80	70	210	62 "	108	13 " 7	22 000 "	30 000
	100	90	190	66 "	120	15 " 9	25 000 "	36 000
	115	105	190	66 "	120	16 " 9	33 500 "	42 000
	140	125	190	72 "	120	17 " 10	40 000 "	52 000
	170	150	190	72 "	120	18 " 10	49 000 "	60 000
	80	60	240	58 "	80	11 " 10	16 500 "	21 500
	100	80	230	62 "	86	13 " 12	21 500 "	26 000
	120	100	220	66 "	96	14 " 12	26 000 "	33 000
	160	140	210	70 "	108	15 " 15	36 500 "	47 000
	200	180	190	72 "	120	15 " 16	45 500 "	58 000
	230	210	190	71 "	120	16 " 17	50 500 "	62 500
	280	250	190	80 "	120	18 " 18	60 000 "	75 000
	340	300	190	84 "	120	18 " 19	71 000 "	87 000
(3)	4	.....	340	16		7 " 2	2 100	
(3)	6	.....	340	16		8 " 2	2 400	
(3)	8	.....	300	22		8 " 2	3 100	
(3)	10	.....	300	22		9 " 2	3 300	
(3)	12	.....	260	30		9 " 3	4 700	
(3)	15	.....	260	32		10 " 3	5 100	

ote. — *A* on city gas, natural gas, or gasolene; *B* on producer gas; (1) commercial e, (2) electric type, (3) on skids.

avoid the common error of purchasing gas engines of too low a power for their work. Brake hp of automobile engines is found by the empiric formula, brake hp = $D^2N \div 2.5$, where D = diam of cyl, in, and N = number of cyls.

Tables 26 to 30 give commercial sizes of different types of internal-combustion engines.

Table 27. Standard Sizes of Hornsby-Akroyd Low-compression Oil Engines

No of cyl	Hp	Rev per min	Diam fly-wheel	Approx floor space, ft	Wt, lb	No of cyl	Hp	Rev per min	Diam fly-wheel	Approx floor space, ft	Wt, lb
1	10	290	54	8 by 4	5 000	1	125	145	144	21 by 13	75 000
1	15	260	57	9 " 5	7 500	2	40	220	84	11 " 9	17 000
1	20	220	60	10 " 6	10 500	2	50	220	84	11 " 9	22 000
1	25	220	72	11 " 7	13 600	2	70	220	94	13 " 11	27 000
1	35	220	72	12 " 7	16 000	2	100	180	108	15 " 13	39 000
1	50	180	108	15 " 9	24 000	2	170	160	132	17 " 13	66 000
1	85	160	132	17 " 10	41 000	2	250	145	144	21 " 17	115 000

Note. — Sizes 10–35 hp have 2 flywheels.

Table 28. Standard Sizes of Franchetti High-compression Oil Engines

No of cyl	Hp	Rev per min	Diam flywh'l, in	Approx floor space, ft	Height, ft	Wt, lb
1	60	225	102	14 by 11	7	33 500
1	100	200	115	17 " 12	8	48 000
1	140	164	132	20 " 15	9	70 000
1	200	150	153	23 " 17	10	110 000
2 (*a*)	120	225	102	14 " 16	7	55 000
2 (*a*)	200	200	115	17 " 17	8	75 000
2 (*a*)	280	164	132	20 " 20	9	110 000
2 (*a*)	400	150	153	23 " 23	10	170 000
2 (*b*)	120	215	102	14 " 13	7	55 000
2 (*b*)	200	200	115	17 " 15	8	75 000
2 (*b*)	280	164	132	20 " 16	9	110 000
2 (*b*)	400	150	153	23 " 20	10	170 000
2 (*c*)	200	200	115	17 " 16	8	70 000

Note. — (*a*) = pulley between cyls; (*b*) = no pulley, flange on end of shaft; [illegible] = 3 bearings.

Table 29. Standard Sizes of Small Gasolene or Kerosene Engines

No of cyls	Hp	Rev per min	Diam fly-wheel, in	Approx floor space, in	Height, in	Wt, lb
1	1	800	12	36 by 18	15	135
1	1	650	14	29 " 19	18	150
1	1	600	17	42 " 22	21	270
1	1.75	550	20	48 " 25	25	350
1	2.5	500	22	54 " 29	28	500
1	4	450	30	51 " 36	32	1 470
1	6	425	33	56 " 38	36	1 900
1	8	400	37	63 " 40	40	2 740
1	10	375	41	68 " 42	44	3 420
1	12	350	45	73 " 48	48	3 850
1	15	325	50	79 " 51	54	4 600
.....	20	350	48	75 " 72	63	6 300
.....	25	350	56	80 " 72	70	7 410
.....	50	350	51	120 " 96	.....	10 500

Note. — Sizes 1–2.5 hp on skids; all throttle-governed, make-and-break ignition.

Table 30. Standard Sizes of Busch-Sulzer Diesel Engines

of cyls	Hp	Rev per min	Diam flywheel, in	Approx floor space, ft	Height, ft
4	120	300	84	15 by 7	9
4	160	276	96	16 " 8	11
4	240	257	108	19 " 9	13
4	350	225	120	22 " 10	15
4	500	200	132	25 " 11	16

st of internal-combustion engines varies greatly, due to competition and other s. The writer has had 2 bids by different makers for the same contract, one 40% r than the other. Kent quotes following expressions by Wickenden, as applying gines suitable for electrical work: Cost in dollars for 50–125 hp = 400 + (40 × 125–250 hp = 2 500 + (20 × hp); above 250 hp = 32 × hp. A similar expression is engines in general from 10–300 hp is (33.6 × hp) − 115. For gasolene engines, ring expressions by Prof A. A. Potter represent the aver of a large number of cases: 75 hp, throttle-governed, cost (dollars) = 310 + (36 × hp); up to 100 hp, hit-and-governed, 141 + (25 × hp). For a number of gasolene-operated irrigation plants, 8–30 hp, price per hp was about $100. For oil engines, up to 400 hp, cost is 400 + hp). Diesel 4-cycle, up to 600 hp, $60 per hp; 2-cycle, 750–1 500 hp, $42 per hp. st per hp, Franchetti engines: 100 hp, $61; 140 hp, $64; 180 hp, $68; 200 hp, $60; p, $61. Hornsby-Akroyd: 20 hp, $58; 35 hp, $46.

35. Gas Producers and Auxiliaries (33, 38, 41, 42)

nstruction. A gas producer consists of a metal shell, usually cylindrical, lined with rick and having provision for supporting the fuel bed and blasting it with air or a ure of air and steam. Fuel bed is 4 ft or more thick, comprising a zone of ash at ottom, next a high temp zone of burning fuel and finally a layer of fresh coal. The nay be supported on a grate, fixed or revolving; or on a concrete pier rising from a of water, into which the sides of the producer extend, thus forming a water seal igh which the ashes are raked out. Fuel is charged from time to time through a hopper that of a blast furnace, to prevent an outrush of gas or inrush of air.

peration. The process consists in roasting off volatile matter of the fuel, and then izing the carbon to CO or CO_2 by the O in the air and steam; H of the steam is set and the ratio of CO to CO_2 depends on temp and thickness of the bed. Steam is to reduce the temp, the exothermic reaction of the oxidation of the carbon being ly neutralized by the endothermic reaction of breaking down the steam. Without e temp is usually high enough to fuse the ash, forming clinker and preventing flow of hrough the bed.

uels. Though any fuel rich in carbon may be used, those containing much volatile ter are disadvantageous due to variation in gas quality and difficulty in removing and soot formed by reduction of the heavier hydrocarbons. Coking coal is unsatis-ory, as it cakes and tends to block the flow of gas like clinker. Some producers have hanical pokers to break up the cake. The following fuels are used in producers: racite or bituminous coal (non-caking), coke, lignite, charcoal, oil, wood, peat, and te products.

last may be forced through the bed by building up pressure before the bed is reached, n PRESSURE PRODUCERS, or by creating a deficiency of press beyond the bed, as in TION PRODUCERS. In pressure producers (commoner in the larger sizes), the blast aused by a steam jet or positive blower (usually the former, as steam is required in blast and the blower serves as a good mixing device). In the suction type, the ine piston draws the blast through the producer (a small auxiliary blower being used starting up). Though this type requires no gas holder, it is found disadvantageous employ the engine for producing blast, and a rotary blower or fan is generally used draw the gas from the producer and deliver it to the engine. The suction producer imited to the better grades of fuel, as anthracite and coke, but is safer, because, as ts are inward only, the burning of some of the gas in the producer is all that ca- --

sult; whereas any leak in a pressure producer allows escape of very poisonous gas and may result in forming an explosive mixture in the room or building.

The draft may pass upward through the bed, UP-DRAFT PRODUCER; downward, DOWN-DRAFT PRODUCER; or both ways, COMBINED-DRAFT PRODUCER. In the two latter the volatile matter passes through the combustion zone and is either reduced to simpler forms or is completely burned.

Producer auxiliaries. The term PRODUCER SET covers producer, auxiliaries, and engine. The most important adjunct is the cleaning and purifying device, usually comprising a scrubber and filter or tar extractor. The SCRUBBER is a chamber filled with coke or similar material, through which the gas rises while water passes down; sometimes it is a simple pipe, up which the gas travels, passing water sprays at short intervals. FILTERS, of saw dust, excelsior, or mineral wool, extract tar and other liquids, but are only partially successful. TAR EXTRACTORS, other than filters, are centrifugal fans or beaters for churning the gas and water, causing the tar particles to collect in large drops which then float away. A recent filter seems to depend on static electrical action causing minute particles of tar to coalesce into drops large enough to fall by gravity in an ordinary separator; tar so recovered is water-free, and has value as a by-product. Suction producers must also have apparatus to supply steam for humidifying the air. As no press is required, the steam may be made in the producer jackets, or in a boiler or evaporator heated by the gas on its way to the scrubber. The proportions of steam and air must be controlled; best done by using saturated air, but of different temp. For a pressure producer a separately fired boiler generates steam for the blower.

Mond producers recover ammonia from the coal. An excess of steam is supplied to the producer, reducing the temp of the reaction and hence increasing the CO_2 content but preventing the breaking down of the ammonia. After purification the gas passes to a scrubber supplied with dilute H_2SO_4 instead of water, which absorbs the ammonia as sulphate, and upon evaporation the solid sulphate is obtained.

Rating of producers is more uncertain and difficult than that of boilers; the term horsepower of a producer is incorrect, as no mechanical work is done. However, a producer is usually designated as being of a certain hp, meaning that it will supply gas for an engine of that hp. On the basis of 1 to 1.5 lb coal per hr per engine hp, and a rate of combustion of 7 to 10 lb per sq ft of producer grate surface (figures realized in average use of both types), a producer will supply about 7 hp per sq ft of grate.

Table 31. Producer Dimensions (Smith)

Hp rating	Coal per hr, lb	Space required, ft			Hp rating	Coal per hr, lb	Space required, ft		
		Length	Width	Height			Length	Width	Height
50	65	12	7	20	175	220	15.5	9.5	22
75	95	12.5	7.5	20	200	250	16	10	22
100	125	13	8	20*	250	315	17	11	22
125	160	13.5	8.5	20	300	375	18	12	22
150	190	14	9	22	....	.....	.....	.....	.....

Producer costs. The following figures are from actual installations:

Suction producers, up to 300 hp,	$252 + (14 \times hp)$	Prof A. A. Potter
Pressure " " " 300 "	$860 + (15 \times hp)$	
Suction " and auxiliaries, 0–200 hp,	$570 + (46 \times hp)$	
Suction producers and auxiliaries,	$150 + (11 \times hp)$	Prof W. S. Wicksenden (kent)
Pressure " " " to 200 hp,	$1\,000 + (16 \times hp)$	
" " " " over 200 hp,	$2\,000 + (15 \times hp)$	

Producer performance. For information, see Bib 5, 33, 38, 41.

APPARATUS FOR TESTING POWER PLANT

(Codes governing the testing of power-plant machinery are published by Amer Soc Mech Engs and may be purchased from the society's Secretary, 29 W 39th St, N Y. See also Bib 13, 14, 15, 16. Following suggestions, relating to methods and procedure may be of assistance in laying out and running tests.)

36. Preparations. Pressure and Temperature Measurements

Preparations. Define the object of the test, and decide upon the limit percentage of allowable in the results. Determine the quantities to be measured, by analysing xpressions denoting the final results (which are generally compound units) into ities of a single dimension, since there are very few methods or instruments for uring quantities of more than one dimension. Thus, the determination of water alls for the wt of dry steam per hr per ihp, and the measurements necessary are: steam used, dryness factor, length of run, m e p in the cyl, length of stroke, area of ., and number of strokes per min.

Choice of instruments and methods of measurement for any particular investigation ds upon the methods available and the limit percentage of error allowable. If an ate error of 20% is permissible, it is unnecessary to use apparatus of extreme acy; but, if the results will be worthless unless accurate within a very few per cent, eless to proceed unless accurate methods are available.

After deciding upon the test methods, a DIAGRAMMATIC SKETCH should be made of the ne to be tested, with all instruments and measuring apparatus shown in their proper re positions. This sketch serves as a guide in setting up the apparatus. A log of the necessary readings is then drawn up, with a column for each variable and for each constant. The log should also contain spaces for description of apparatus , date of test, names of observers, and remarks concerning unusual occurrences the test. The completeness of the log and sketch of instrument arrangement l be proved by seeing that there is a place for every instrument reading, and that the las to be used for calculating final results can be fully satisfied for the log readings.

Pressure measurements. The ordinary press units are: lb per sq in, inches of mercury, of water, and ft of water. 1 lb per sq in = 2.0359 in mercury at 32° F = 27.71 in 69 ft water at 62°. Lb per sq in is used for all ranges of press, except vacuum and up to 10 or 15 lb above atmos, which are generally expressed in inches of mercury. ce draft, press in ventilating ducts, and other small pressures, are expressed in inches ter. For hydraulic work, ft of water are used almost exclusively.

Pressure gages of the Bourdon tube type, with scales ranging from 30 in vacuum (15 lb to several thousand lb, are of either the indicating or recording form. Diaphragm are not widely used. Gages should be CALIBRATED before and after use. The reliable gage tester is the "dead-weight" tester. Measurements by mercury meter, of either the U-tube or cistern form, are standard if the mercury be pure. ty of mercury is altered by dissolving tin in it; evidence of such adulteration is by subjecting a mercury column and water column to the same press, and com- the ratio found with that for the pure liquids. Pure water should be used in manometers, but the ordinary impurities in fresh water will not materially affect gs. For measuring very small press at temp below 32°, other liquids may be used nometers. Such readings should be transposed into inches of water, by direct com- n with a water manometer, or by measuring the sp gr of the liquid by a hydrometer. mall press may be measured by a water manometer with tube inclined at a consider- ngle to the perpendicular, thus causing large longitudinal travel of the surface of uid for small change in elevation.

Temperature measurements are made by the Fah scale by engineers, though chemists e Cent scale. 1° F = 5/9° C. Temp in °F = temp in °C × 9/5 + 32. (For con- n table, see Sec 37.) The mercury thermometer, generally used for measur- np, is made with a range of scale from −40° to 1 000° F. Indicating and recording ments are also made, in which the actuating portion is a bulb filled with liquid , connected by a flexible tube to a Bourdon tube which moves the pointer or record- n. Electric pyrometers, indicating or recording, are of either the thermo-couple stance type, for all ranges of temp to 3 000° F. Optical and radiation PYROMETERS measuring temp from 1 000° F up. High temp is sometimes gaged by the fusing erials which have well-defined melting points.

Temperature measurement of gases is difficult. The temp of a gas flowing through may vary considerably in different parts of the cross section, and the true aver is o determine. Also, the instrument must not be exposed to radiation from some at a temp higher than that of the gas. Failure to shield the instrument from su- ion may cause errors of several hundred deg. As gases are transparent they n

not radiate heat, and, if optical or radiation pyrometers are used for measuring ... temp, a solid object must be placed in the gases at the point where the temp is ... measured and upon which the instrument is sighted. Such an object should be dull colored; all optical or radiation pyrometers are calibrated for a black object.

37. Measurement of Weight, Dimensions, Speed, and Power

Measurement of weight is made by either lever or spring balance. Spring balances are often more convenient, but their length of scale is limited and their percentage error relatively high. After long use their accuracy is more questionable than that of lever balances. Wt may be determined indirectly by measurement of the volume of an object when its sp gr is known (Sec 1, Art 3).

Measurement of length, area, and volume. Small irregular areas on drawings may be measured directly by a planimeter. Irregular volumes may be determined by weighing in air and immersed in water, the difference being the wt of water displaced, from which volume is computed. Differences in altitude may be determined by measuring the difference in press in a water pipe running from one point to the other. The water must not be flowing when the measurement is made.

Measurement of speed. Many types of stroke or revolution counters are used. Some are attached to the machine; others are for hand use. They are the most reliable for general service, especially when the speed varies. TACHOMETERS indicate or record directly rev per min. They operate by the centrifugal force of solids or liquids, voltage of an electric generator, or synchronous vibration. They should be carefully calibrated before use in important work, and it is wise to check their readings by a revolution counter. They are unsatisfactory for variable speeds, because averages must be estimated. Chronographs make a graphical record of each revolution, with simultaneous time record.

Power measurements. Standard unit in the U S is the horsepower = 33 000 ft-lb per min = 0.7457 kilowatts. Power developed in cyls of engines, pumps, or compressors is indicated horsepower (ihp). It is measured by an engine INDICATOR, a special form of pressure gage which records on a piece of paper wrapped on a drum, caused to follow the motion of the piston. In nearly all indicators the press-recording mechanism is driven by a spring-opposed piston, the cyl of which is connected with the engine cyl. Springs are furnished of a wide range of strength, and the scale marked on the spring indicates that the recording pencil will rise 1 in for that unit press. Mean effective press (m e p) is the aver height of the recorded diagram, inches, multiplied by the scale of spring used. The indicator drum, usually 1.5–2 in diam, is generally driven from the engine crosshead, with the necessary reducing motion. The indicator is connected to the cyl through a special cock, which has a standard 0.5-in pipe thread on one end, the other end being special for each make of indicator. Each indicator must have its own cock. Indicators are often used for recording rapidly fluctuating press in pipes, the drum being designed so that a continuous roll of paper is passed over it.

Power output of machines may be measured by absorption or transmission dynamometers. The absorption dynamometer or PRONY BRAKE is easily made in any shop, the details of its design being governed by the work to be done. Fig 39 shows a Prony brake made of wood blocks held together by steel bands. The arms are of pipe or iron. Friction is adjusted by handwheel. Bhp = $2\pi LN \times$ rev per min $\times W \div 33\,000$, where W = net wt registered on the scale, less unbalanced wt of the brake and support, and L = length of brake arm, ft. Prony brakes are rarely used to absorb more than 200 hp, the limitation lying in the difficulty in removing the heat generated. This difficulty may be overcome by water-cooling the brake. This is done by using a brake pulley with an internally-flanged rim, which when running will hold a body of water in contact with the pulley rim. Another absorption dynamometer is the Alden brake, made with a rotating steel disk surrounded by

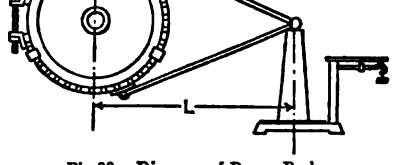


Fig 39. Diagram of Prony Brake

g. Within the casing are copper plates, pressed against the disk by water press. r is determined by measuring the torque of the casing, the heat generated being d off by the water.

umber of HYDRAULIC BRAKES are made on the principle of a centrifugal pump, with s on casing as well as on rotor. Power is absorbed by the eddy-current friction, and asured by the torque of the casing. These brakes have been made to absorb over hp. At low speeds their capacity is relatively low, but increases about as the cube e speed.

electric generator may be used for measuring the power of a machine, if its effic is n. ELECTRIC BRAKES are made on this principle, with the generator stator hung ll bearings, so that the torque can be measured as for any type of brake. By run- them as motors they will also accurately measure power input to a machine.

8. Measurement of Water and Flow of Steam. Calorimeters

asurement of water or other liquids. The most accurate mode of weighing liquids anks on scales. When measured volumetrically, by change of level in a calibrated the area of the water surface should be as small as possible at the points where the urements of depth are taken, so that a small error in depth will not make a large in volume. Flowing water is measured by weirs, Venturi meters, pitot tubes, s, and nozzles (Sec 38). Piston and disk meters, also used for small and intermittent should always be set up so that they can be easily calibrated. Disk meters as usu- onstructed will not handle hot water.

asurement of steam flow should be made in the form of water, either before it is the boiler or after it leaves a condenser. STEAM METERS operate on the pitot tube, ri meter, orifice, or float principle. Chief source of error is in the recording or in- ig device, which, for accuracy, should correct for press and moisture content or eat. A steam meter may be calibrated by placing it in a steam line from a boiler, d water being weighed, or in the line to an engine exhausting to a surface condenser.

asurement of gas flow is more difficult than that of steam, as the meters are not ly calibrated. Small quantities of gas can be measured in gasometers or gas holders, et or dry gas meters give fair accuracy for small flow. But, large quantities of ist be measured by a RATE-FLOW METER. Pitot tubes, Venturi meters, orifices of s forms, and anemometers, are generally used. The standard meter for measur- is the Thomas electric gas meter, in which the gas is heated by an electric resistance d the amount of current measured. The wt of gas flowing can then be computed he rise in gas temp and the specific heat of the gas. The apparatus is expensive, generally used for calibrating the simpler and less costly forms of meters.

um calorimeters. For most work a throttling calorimeter is best. It consists of a ghly lagged metal chamber (often made of pipe fittings), with a thermometer cup top and an opening to the atmos at the bottom. It is connected close to the steam d the steam flows into it through a small orifice. Total heat in the steam is com- from the amount of superheat indicated by the calorimeter thermometer. As nciple of operation depends upon the difference in total heat, the range of this ient is limited, about 4% moisture being the max at 100 lb steam press and 5.5% lb press. For very wet steam a separating, or combination separating and throt- lorimeter should be used. The chief source of error in steam calorimetry lies in the y of obtaining a truly representative sample of steam; hence care should be taken ing the sampling nipple.

calorimeters are of 2 classes, those which operate continuously and those operat- single charges. Continuous calorimeter is used for gas and light fuel oils. The n form resembles a small fire-tube boiler, set vertically, with an internal fire box. el burner is placed within the fire box, the air for combustion being drawn in at tom. Water is circulated around the tubes, and the heating surface is sufficient the gases of combustion to be cooled down to atmos temp. The relative quan- water and fuel are so adjusted that the rise of water temp is from 20 to 30° F. u per lb or cu ft of fuel can be computed from measurements of the amount of ned, wt of water heated, and the use in water temp. The discontinuous type of eter, generally known as the bomb calorimeter, is for solid fuels and heavy oils. ed amount of fuel is placed in a heavy chamber or bomb, together with oxy-

gas under press, or a chemical containing oxygen to support combustion. The bomb placed in a fixed wt of water, in an insulated chamber, and the charge is fired by an electric spark. The rise of water temp measures the heat units set free.

39. Contracts and Acceptance Tests

When contracts for power machinery contain a clause guaranteeing the performance the wording of the guarantee should be such that its meaning can not be mistaken, and that the operating conditions are clearly specified under which the performance test is made. Unless otherwise agreed upon, the acceptance test is run by the engineer of the purchaser. The contractor is allowed to have a representative present, who shall have access to all readings, though without right to interfere in the running of the test, that he may see that the machine is properly adjusted before the test is begun, and in the case of boiler tests he is generally permitted to direct the firing, but not to furnish the fuel. The engineer in charge is responsible for maintaining the operating conditions for which the guarantee is made. When there may be difficulty in maintaining such conditions some agreement should be reached by the contracting parties as to allowances to be made in fulfillment of the guarantee on account of such variations of conditions. When it is necessary to make measurements by other than generally accepted methods, a mutual agreement on the matter should be reached before the test begins. All readings should be entered upon the official log sheet (Art 36), which should be retained in its original form because, in case a lawsuit develops upon the results of the test, that sheet alone has legal standing in court as evidence.

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SECTION 41

MECHANICAL ENGINEERING MISCELLANY

BY

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1. Gearing

Characteristics. The term as here used means tooth gearing; friction gears and other special forms will not be considered. According to the shape of the teeth gears are divided into involute and epicycloidal; former has many advantages and the latter is little used. PITCH CIRCLE (Fig 1) is the base to which gear calculations are referred. The pitch circle diameters of 2 intermeshing gears are equal to the diameters of 2 equivalent cylinders, rolling in contact without slipping. CIRCULAR PITCH is the distance measured on pitch circle from center to center of teeth: hence

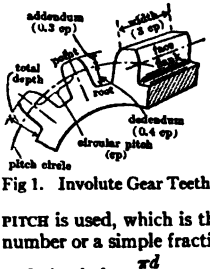
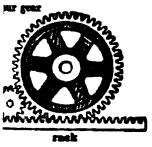


Fig 1. Involute Gear Teeth

$$\text{cir pitch} = \frac{\text{pitch circumference}}{\text{no of teeth}} = \frac{\pi d}{N}, \text{ where } d = \text{diam}$$

of pitch circle and N = number of teeth in the gear. Circular pitch is not convenient for determining number of teeth or pitch diam; hence, the DIAMETRAL PITCH is used, which is the number of teeth per in of diam, and is either a whole number or a simple fraction. Then, no of teeth = pitch diam × diametral pitch, and cir pitch $= \frac{\pi d}{N} = \frac{\pi d}{d \times \text{diametral pitch}} = \frac{\pi}{\text{diametral pitch}}$. In this way proportions are readily calculated. Thus, for a 2 diametral pitch, if the wheel has 20 teeth the pitch diam is 10 in; or, if a wheel has 60 teeth and the pitch diam is 20 in, the diametral pitch is 3, and the circular pitch is

ar gears are gears in one plane, with their axes parallel. If the radius of pitch circle nite, the circle becomes a straight line, resulting in the RACK. Spur gears having ame pitch are interchangeable (Fig 2). HERRINGBONE GEARS are used when end t on shafting is to be avoided. Their teeth are cut at an angle with the axis (Fig 3).



Spur Gears and Rack

Fig 3. Herringbone Gears

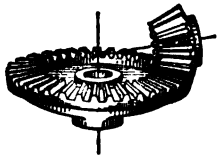


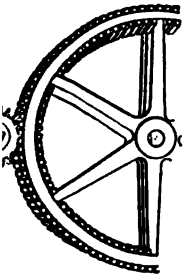
Fig 4. Bevel Gears

rel gears are those of which the axes are in one plane and intersect; they are called gears if both are of the same diam. Bevel gears are not interchangeable for the pitch, but must be made in pairs (Fig 4).

rm and worm-wheel are gears with axes at right angles n parallel planes (Fig 5). They must be made in sets.

Fig 5. Worm and Worm-wheel

nstruction. Metal gears are usually of C I or steel; brass, metal, raw-hide, paper, and cloth, are used for special Teeth cut in the solid rim give the most perfect form oth, and are known as CUT GEARS. If cast in a sand l the tooth outline is less accurate than the cut gear; ration they are noisy, noise increasing with speed. Small are cast in steel moulds under pressure, known as die-g. They are accurate in form, but can be made only of metal and commercially must be duplicated in large quan- because of the expensive dies. Gearing is also made of serted tooth type. The teeth, of maple wood, are inserted -I rim; one wheel of a pair has wooden teeth, the other, ary metal teeth (Fig 6). Advantages: they run quietly and in case of accident the n teeth, which are the ones to be damaged, are readily replaced. Wooden teeth are made in coarse pitches only. They are wider on pitch circle than the iron tooth, in the proportion of 0.6 circular pitch for the wooden to 0.4 circular pitch for the iron, making strength and wear more uniform. Gears of large sizes are made in halves or sections, bolted together, and can therefore be placed anywhere on a shaft.



). Wooden-tooth Gear

Spur-gear ratios for single reduction of speed seldom exceed 6 : 1; in some special cases as high as 10 : 1. When greater reduction is required, gear trains may be used; in that shown in Fig 7 the driver has 20 teeth and meshes with

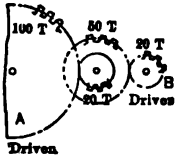


Fig 7. Compound Gear Train

tooth wheel, giving a 2.5 : 1 reduction; attached to the same shaft as the oth wheel is a 20-tooth wheel, forming a COMPOUND GEAR. The latter l meshes with a 100-tooth wheel, giving a 5 : 1 reduction, or a total of 5 = 12.5 : 1.

andy rule for calculating gear ratios in compound trains is $\frac{\text{Product of the drivers}}{\text{Product of the driven}} =$ ratio; in which diameters, numbers of teeth, or rev per min may be used. Where reductions of 3 or 4 : 1 are made, the teeth on small gears wear much faster than

those on the large. To equalize wear, the smaller or pinion gear is made of harder ma as steel, while the larger is of C I. When the number of teeth are exact multiples tooth on the pinion will always mesh with the same tooth on the large wheel, and case there were opposed hard and soft spots in the teeth, wear would be rapid. In where exact gear ratios are unnecessary, as for triplex pumps, the larger gear has an ex tooth called a HUNTING TOOTH; thus, if the pinion has 20 teeth and the large wheel teeth, every tooth on the pinion will come in contact with every tooth on the la gear, thus distributing the wear more evenly over both wheels.

Strength of gear teeth. The weakest part of a tooth is at the root in the sm wheel, as at *a*, Fig 8, and in practice, the root of the tooth is bounded by the ra lines, as shown. Upon this basis have been calculated the formulas for the streng teeth, the smallest number of teeth being 12. As the number of te increases, the root width *a* and the strength also increase. Wid face for good contact is generally assumed to be 3 times the circ pitch, but this is sometimes exceeded. Spur and bevel gears are tainable of almost any size, catalogues giving stock sizes in pi from 0.5 to 46 diametral pitch. Selection of proper pitch de upon size of gear and power transmitted; in general, the larger gear the greater the power transmitted and the coarser the pi

Fig 8.

WORM-GEAR REDUCTIONS are much higher than for other types of ing. If the worm is a single thread, for each revolution of the w the worm-wheel will move 1 tooth; for a double, triple or quadruple thread, the wh will advance 2, 3, or 4 teeth. In practice, the wheel should have not less than 2 and as the diams of worm and worm wheel have no influence on the gear ratio, the cor proportions must be carefully worked out to secure efficient operation. In ma catalogues, ratios of 15 : 1 for quadruple threads, or 129 : 1 for single thread, are quot In all gearing, positive ratios are obtained; there is no slipping as with belting.

2. Belting

Principles of pulley and belt transmission. Since slippage always oc exact speed ratios can not be obtained. Resistance to slippage depends o the coeff of friction, area of surface in contact, and the press on pulley fac the belt pull. The effective pull is equal to the difference between the tens on the tight and slack sides of the belt. The angle over which contact ta part is the angle of wrap. Coeff of friction depends upon the materials surfaces and the condition of the two surfaces. For ordinary iron or wooden pulleys and leather belting, the coeff is 0.3 to 0.4 aver for dry surfaces, and say 0.15 to 0.2 for wet or greasy surfaces.

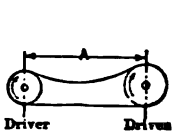


Fig 9. Favorable Belt Drive

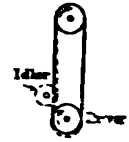


Fig 10. Unfavorable Belt Drive

For most operation the coeff must be as high as possible. It is evident that the best drive is that shown in Fig 9, that is, the tight side below, since any stretch in the belt increases the angle of wrap, one of the power factors. The worst drive is shown in Fig 10. The lower pulley being the driver, any stretch in the belt causes excessive wear. In practice this form of drive requires an idler pulley and lower tension.

Power transmitted by a belt depends first upon the difference in ten between tight and slack sides, and second upon the belt speed. With surf a 200 coefficient, and 180° angle of wrap, the ratio of tension on tight side (T_t) to tension on slack side (T_s) averages about 2.5, that is, $T_t \div T_s = 2.5$. This ratio varies from 2 in small sizes to 3 in large, the pulley diams being equa practically so. To solve this formula either T_t or T_s must be known. T_t is obtai by dividing the breaking strength of a piece of belting, from which is determin

ife **working stress.** Since belting varies in thickness, the tension per in lth is generally given, rather than tension per sq in of cross sec.

t **speed.** Increase in speed, other conditions unchanged, increases power delivered, a certain limit. But, as the belt travels over the pulley, centrifugal force throws it from the pulley face and partly neutralizes the driving force. The limit is reached peed of about 5 000 ft per min; any further increase of speed decreases the power nitted. Hp = [pull of belt (lb) × veloc (ft per min)] ÷ 33 000. In practice, to 4 500 ft per min are aver speeds. Velocity must sometimes be sacrificed, due to ge diam of pulleys required for given rev per min of shafting, and the belt width is ed to transmit the required hp. Belting is usually of leather, or several plies of covered with rubber and pressed together. Woven belts, or thin steel bands, so used.

ather belts are of single, double, or triple thickness. Makers' catalogues ables of the hp transmitted by different kinds of belting at given speeds, gures being based upon an arc of contact of 180°, and an aver working ten- per in of width found satisfactory in practice, viz: 40 lb for single, 70 lb ouble, and 100 lb for triple thickness. Thus, if a belt running at 3 000 ft in is required to deliver 60 hp, the total pull would be $P = 33\,000 \times 60 \div$ = 660 lb. For a double belt a pull of 70 lb per in of width is allowable, g 660 ÷ 70 = 9.5, or say a 10-in belt. It is good practice to design a belt capable of transmitting 25% excess power for small sizes and 5% excess arge drives. One maker recommends the following rule, which includes ve power: for single-thickness leather belt, the hp per in of width = belt l in ft per min ÷ 800; for double thickness, divide the speed by 500.

ting should be properly cleaned and dressed with a reliable compound to keep it nd pliable, and be systematically inspected. Belts exposed to dampness or acid require special treatment in manufacture. If a belt is stretched to its limit to mit a given hp, its life is short. Since tension produces a heavy pull on shafting earings, causing friction and wear, it should be a minimum for the work to be done. tension gives long life and increased first cost; high tension, a short life and lower ost, but with more stoppages for belt repairs. Shafts can be driven at almost any , if provided with idler or guide pulleys. If the distance between centers of shafts iple (Fig 9), the weight of belt on slack side produces the necessary tension, and h increases the angle of contact, which is also favorable. If distance between centers rt, the belt must be stretched to produce necessary tension. Slippage, which will after a short time, due to increased stretch, can be overcome by adjustable idler ys placed on the slack side, close to the driving pulley, so as to increase the arc of act (Fig 10). If the distance between centers is excessive, the tight side will sag, easing the angle of contact, and the belt is liable to flap, ng the faces to rub on each other. To get proper tension putting on new belting some makers recommend cutting in of belt for every 10 ft of tape line measurement. For ing shafts in opposite directions, crossed belts (Fig 11) sed, but should be avoided if possible, because the belt rubs at point of crossing. other types of belt drive, the makers should be consulted. Leather belting can be from 0.5 in to 72 in wide; widths from 0.5 to 1 in vary by ⅛ in, from 1 to 4 in by in, 4 to 7 in by 0.5 in, 7 to 26 in by 1 in, 26 to 40 in by 2 in, and above 26 in by 4 in. e figures in Tables 1, 2 are based on first-class short-lap belting, made from backs of oak-bark tanned leather, run under ordinary conditions and on pulleys of medium and l size. When the driven pulley is smaller, the power transmitted will be as much than the tabulated values as that part of the surface of the driven pulley which is red by the belt is less than half its surface.

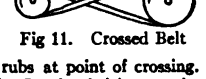


Fig 11. Crossed Belt

ubber belts consist of several thicknesses (plies) of canvas covered with ber. They can be used in damp or wet places, or in drives exposed to the osphere or to acid fumes.

uality of rubber determines grade of the belt, new rubber being better than reclaimed luct. Strength depends upon the wt of canvas used, which is given in oz per sq yd of

Table 1. Hp Transmitted by Heavy Single Leather Belts

Width of belt, in	Velocity, ft per min								
	400	600	1 200	1 800	2 400	3 000	3 600	4 200	4 8
	Hp transmitted								
1	⅜	1	2	3	4	5	6	7	
2	1½	2	4	6	8	10	12	14	
3	2	3	6	9	12	15	18	21	
4	2⅝	4	8	12	16	20	24	28	
5	3⅛	5	10	15	20	25	30	35	
6	4	6	12	18	24	30	36	42	
7	4⅝	7	14	21	28	35	42	49	
8	5⅛	8	16	24	32	40	48	56	
10	6⅜	10	20	30	40	50	60	70	
12	8	12	24	36	48	60	72	84	
14	9⅛	14	28	42	56	70	84	98	
16	10⅜	16	32	48	64	80	96	112	
18	12	18	36	54	72	90	108	126	
20	13⅛	20	40	60	80	100	120	140	
22	14⅜	22	44	66	88	110	132	154	
24	16	24	48	72	96	120	144	168	

Table 2. Hp Transmitted by Heavy Double Leather Belts

Width of belt, in	Velocity, ft per min								
	800	1 200	1 600	2 000	2 400	3 200	3 600	4 000	4 8
	Hp transmitted								
2	4½	6¾	9	11¼	13½	18	20¼	22½	
4	9	13½	18	22½	27	36	40½	45	
6	13½	20¼	27	33¾	40½	54	60¾	67½	
8	18	27	36	45	54	72	81	90	
10	22½	33¾	45	56¼	67½	90	101¼	112½	
12	27	40½	54	67½	81	108	121½	135	
14	31½	47¼	63	78¾	94½	126	141¾	157½	
16	36	54	72	90	108	144	162	180	
18	40½	60¾	81	101¼	121½	162	182¼	202½	
20	45	67½	90	112½	135	180	202½	225	
24	54	81	108	135	162	216	243	270	
30	67½	101¼	135	168¾	202½	270	303¾	337½	
36	81	121½	162	202½	243	324	364½	405	
40	90	135	180	225	270	360	405	450	
44	99	148½	198	247½	297	396	445½	495	
48	108	162	216	270	324	432	486	540	

canvas duck. Rubber belting is cheaper than leather. Makers advocate a differe... tension of 60 lb per in of width for 4-ply belts, made of new rubber, with minimum pu... diam of 24 in. Smaller pulleys can be used, but the life of belt will be shortened, ... bending over the pulleys, the outside layers being stretched, while the inner are ... pressed, thus causing working of the layers on one another. On this basis makers ... the following figures, 3 or 4-ply rubber belts being rated as the equivalent of ... thickness leather, and 5 or 6-ply the equivalent of a double leather belt transm... about 50% more power than a single. The arc of contact is taken as 180°.

4-ply rubber belt, 60-lb tension per in of width, min diam of pulley, 24 in
5 " " " 80 " " " " " " 30 "
6 " " " 100 " " " " " " 36 "
7 " " " 120 " " " " " " 42 "
8 " " " 140 " " " " " " 48 "

Some engineers estimate the aver value for tension at 20 lb per in of width per ply ... canvas, which is higher than the values given above. In laying out belt drives, am...

ace must be between ma- frame and the f wooden build- be belt rubbing t a timber may e enough heat ise a fire, be- destroying the If the outside coating is ted, moisture nes will pene- the canvas and y the belt. ation of resin- substances to faces to in- adhesion also ys belting by g the rubber. transmitted, upon previous la for 180° arc ntact, is $hp = V \div 33\,000$; if c is 240°, allow ; if 120°, allow if 90°, allow An allowance to 0.25 in per ft e line measure- for stretch in r belts is recom- ed by makers. AS BELTS are by sewing plies nvas together; n belts by a ing process as th, animal fiber used in some In using this

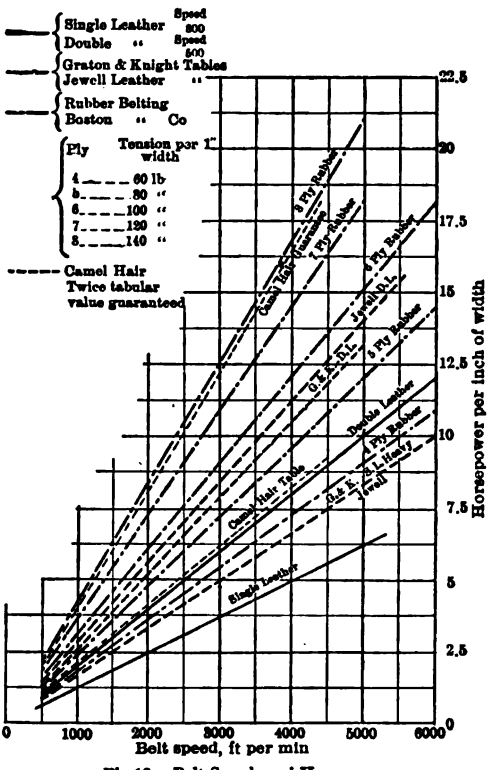
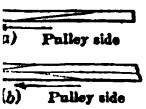


Fig 12. Belt Speeds and Horsepower

of belting the tension must be the same over the entire width, otherwise the belt ot run true. CAMELS-HAIR BELTING is guaranteed to transmit twice the power of a leather belt, based upon the increased coeff of friction and strength of belt.



13. Scarf Joints, Leather Belt

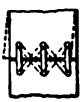


Fig 14. Belt Lacing

Belt fastenings. For leather belting, joints are made by scarfing and gluing the ends together, making an endless belt. Some machines are designed to be shifted on their frames to take up slack, but when the limit is reached the belt joint must be made over. ingle leather belts the joint should be as in Fig 13 *a*; for double belts, as in Fig 13 *b*. per belting can be similarly spliced. After the surfaces are covered with a rubber nt and allowed to dry they are stitched together with narrow rawhide lace. LACING ld have uniform tension; it is straight on pulley side but crosses on the back; for ow belts, the holes are in one line; for wider, they may be staggered. In lacing a the 2 ends must meet square and in line; if offset, as shown by dotted lines, Fig he edges will be damaged. Metal fasteners are good and are made in many desi- canvas and woven belts, the maker generally recommends some special type,

adapted to his own belting. In the fastener shown in Fig 15 *b* the wire is inserted by a special machine, the ends being held together by a rawhide pin. It is a hinge joint and can be readily taken apart and replaced.

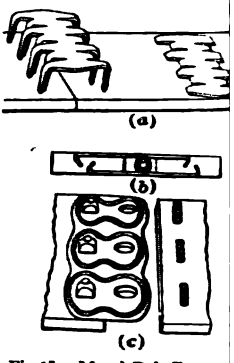


Fig 15. Metal Belt Fastene

3. Pulleys

Pulleys for belt drives are of C I, wood, fiber, or pressed steel. C-I pulleys are solid, or in halves bolted together. A solid pulley must be installed by slipping it over the end of the shaft; split pulleys can be set anywhere on the shaft by bolting them in place.

Principal dimensions for ordering pulleys are bore, diam, width of face, and size of keyway (Fig 16). The face may be crowned or flat (Fig 17). With a CROWNED PULLEY, if the tension is not equal on both sides of belt, or the shafting is slightly out of line, the belt will seek the highest veloc, which is at the middle of the face, and is thus made to run true. Crowning is generally about ⅛ in per ft of width; if excessive, it damages the belt by overstret ing it in the center. Flanges on the pulley face to keep the belt from running off sh be avoided, as they wear the edges of the b If the belt becomes slack it may ride up on t flange and tear. Untrue running of belts is cau by unequal tension or poor alinement of shaft Machinery that must be started and stopped fr quently is sometimes fitted with crowned PU AND LOOSE PULLEYS, the belt being shifted fr one to the other to start or stop the machi The driving pulley has a flat face, and a wi equal to the combined widths of the 2 driven pulleys. BELT SHIFTER should be placed on the side where the belt leads on to the pulley. Very wide C-I pulleys may have double spokes or arms (Fig 18). C-I pulleys are heavier than other types. They vary from 3-in diam and 2-in face to 120-in diam 60-in face. (For details, see catalogues.) Solid C-I pulleys for single-thickness be the cheapest. Split C-I pu cost about 50% more for si sizes, to about 20% more or m dium sizes. C I with the i scale attached withstands s of acid fumes or dampnes be than ste wood. Fa STEEL PU (Fig 1 made in h with d sets of a when nece sary. Sizes vary from 6-in diam by 2-in face to 120-in diam by 36-in face, prices var ing about the same as C-I split pulleys for single belts. Fig 20 shows a set of bush which adapts this type of pulley to several sizes of shaft. WOODEN PULLEYS are made gluing together strips of wood so arranged as to avoid warping. They should not b used in damp places. They are light in wt, and have the same advantages as th

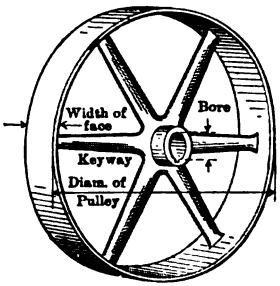


Fig 16. Cast-iron Pulley

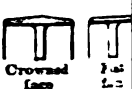


Fig 17. Flat an Crowned Pulle

Fig 18. Double-arm Pulleys

Fig 19. Pressed-steel Pulleys

Fig 20. Pulley Bushings

ɔl-steel pulley. Sizes are from 3-in diam by 3-in face to 120-in diam by 24-in face 21, 22). Prices of wooden pulleys for double belts aver slightly less than solid C-I ys. Catalogues give information as to special types and sizes. If the coeff of friction

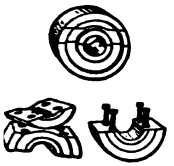


Fig 21. Small Wooden Pulley

Fig 22. Large Wooden Pulley

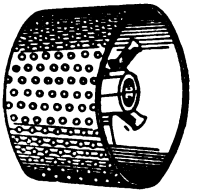


Fig 23. Pulley with Inserts for High Coeff of Friction

ɔe increased, the tension on tight side can also be increased, and more power trans-ɔd; hence, the pulley face is sometimes covered with leather, canvas, or cork inserts 23), increasing the power transmitted for the same drive.

4. Shafting, Hangers, and Bearings

ıafting is of steel, turned and ground, or cold-rolled. The latter has a hard surface, h, if damaged by the cutting of a keyway, may warp the shaft. Shafting is carried ɔck in sizes from 3/16 to 1.5-in diam, in lengths from 1 to 24 ft, and from 1 9/16 to 7 in ., in lengths from 5 to 24 ft. Diam always increases by 1/16 in. Where longer shafts equired, stock lengths are joined by couplings.

ıngers and bearings. Shafting is ed in bearings mounted in adjustable ers attached to the ceiling, walls, floor, ɔsts of a building. Special fittings are ired for steel-frame buildings. Hangers f C I or pressed steel. Fig 24 is a ceil-anger which can be reversed and made or stand. In Fig 25 the bolts *c* pass ıgh slotted holes to permit adjust-t; the bearing can be raised or lowered crews *d*, which support it. The bear-n Fig 26 has a spherical seat at *a*, for ng in any direction about its center. lower cap *b* (Fig 25) can be removed, box taken out, or shafting removed, out interfering with the hanger. Shaft-can be assembled on the floor, hoisted into position in the hangers, and bearings caps readily put on; convenient for heavy work. Boxes should be ring-oiled and dustproof, to minimize attendance. The points for ordering hangers are: style of hanger, drop or distance from center of shaft to the base (Fig 24), and diam of shaft. When in places difficult of access, shafting and bearings are apt to be neglected. When hangers are attached to ceiling beams, the spacing of these determines the distance between bearings; 8 to 10 ft is considered good spacing; if greater, heavier shafting should be used to prevent bending. Length of bearings is given ɛrms of diam of shaft; for diam less than 2 3/16 in bearings are 5 diam long; for heavier ting, 4 diam long. Shafting is subjected to torsion, due to the power transmitted, to bending, due to wt of the pulleys, its own wt, and the wt and tension of the belts. pulleys should be as close to the hangers as possible, and heavy drives placed near the ıt where the power is received. For good service, shafting should be carefully alined, ı a small end play (say 1/8 in), and with collars to prevent any excess end motion. ɛre the power is received, collars should be placed on both sides of one hanger, leavir

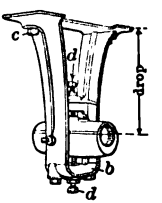


Fig 24. Ceiling Hanger

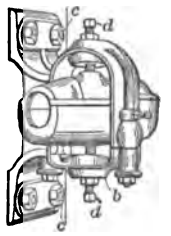
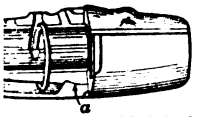


Fig 25. Adjustable Post Hanger



26. Hanger with Spherical Seat

the ends of the shaft free to expand and contract with temp changes. Motors driving shafting in dusty places can be enclosed, and a slight increase of air press above atm maintained on the inside, to prevent entrance of dust. BEARING PRESSURES are determined per sq in of projected area (diam × length of bearing). The aver allowable press, varying with load conditions, speed, lubricant, and method of lubrication, is about 60 lb per sq in of projected area.

5. Rope Drives

Hemp or cotton-rope drives are of two kinds: continuous or American, and multiple or British. AMERICAN SYSTEM. The rope is wound around the pulleys as many times as the power to be transmitted requires; it then passes over a tension carriage (see below), which keeps the rope at constant tension and returns the last wrap to the first groove of driving sheave. This drive has only one splice, which is the weakest part of the rope. Breakage of the rope causes complete shut down, usually avoidable by careful inspection. This system is best for vertical or angular drives, and can be used for heavy powers with short center distance. BRITISH SYSTEM has separate ropes for each loop, and hence has as many splices as ropes. Maintenance of constant tension in all the ropes is difficult but important. If 1 or 2 should fail the others will transmit the power until repairs are made. Low tension lengthens life of the drive, which should average 8 to 10 years.

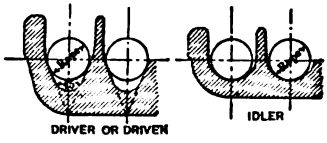


Fig 27. Sheave Grooves, British Rope Drive

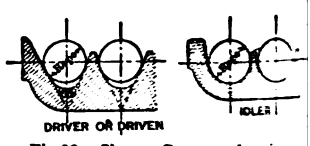


Fig 28. Sheave Grooves, American Rope Drive

Sheaves and rope. Fig 27 shows shape of grooves for main sheaves and idler, of the English system; Fig 28, for the American system. Grooves must be accurately machined with surfaces free from imperfections; rough holes cause rapid wear on ropes. For necessarily heavy, the sheaves act as fly wheels for variable loads, and should be well balanced. Minimum sheave diam is 36 times rope diam; 40 times is better. Rope must not touch bottom of grooves of driver or driven sheaves, otherwise no wedging action occurs to cause the rope to rotate on its axis and so produce uniform wear.

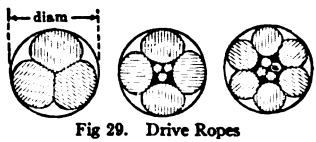


Fig 29. Drive Ropes

Cotton ropes are softer than hemp, cost more, and are more used in Great Britain than in U S. Hemp transmission ropes are laid with internal lubricant to reduce wear. Manila rope has 3, 4, or 6 strands (Fig 29). Diams from 0.75 to 2 in increase by ⅛ in; from 2 to 2.5 in by 0.25 in. (For strengths and wt of hemp rope, see Sec 12, Art 13.) To lie properly in sheave grooves, splices must not increase the rope diam. Strength of a good splice is about 85% that of the rope.

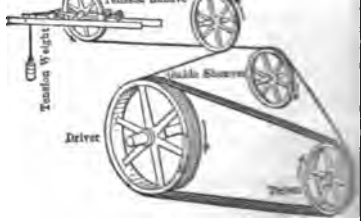
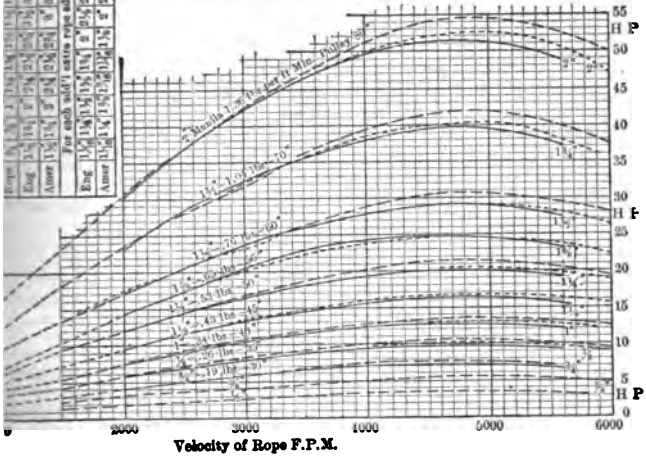


Fig 30. American Rope Drive

In the American drive (Fig 30), constant tension is maintained by a TENSION CARRIAGE, to compensate variation in length, from stretch or shrinkage due to moisture. A tension carriage will take care of 3 000 to 3 500 ft of rope.

ower transmitted by rope drive depends upon the same elements as for belting. friction between the ropes and sides of grooves can be increased by an increase of tension, and the difference between the 2 tensions and centrifugal force is the pull in lb. Rope speed is generally the same as for



$$\text{ıate breaking load, lb} = \begin{cases} 7\,000\,d^2 \\ 720 \times \text{circ'm rope}^2\text{(in)} \end{cases} \qquad \text{Weight of rope, lb per ft} = \begin{cases} 0.34\,d^2 \text{ to } 0.33\,d^2 \\ 0.039 \times \text{circ'm rope}^2\text{(in)} \end{cases}$$

$$\text{er working load} \quad \text{"} = 20 \times \text{circ'm rope}^2\text{(in)} \qquad \text{Diam of a rope} = \text{diam of circumscribing circle}$$

$$\text{(max tension)} \quad = 800d^2 \qquad \text{Circ'm "} \quad \text{"} \quad \text{"} = \text{circ'm "} \quad \text{"}$$

31. English (Multiple) Hemp Rope Drives (C. W. Hunt Co, broken lines; G. V. Cresson Co, dotted lines; Rice & Sargeant, full lines) L. DeG. Moss

ng; but, since rope weighs less than belt, the tension can be increased at high ds to offset effect of centrifugal force; 80 ft per sec is an aver speed.

riving force is $P = T_n - T_0$, where T_n = tension on tight side, and T_0 is the ltant of the tension on slack side (taken as 0.5 P) and centrifugal force C. Hence, ower may be written, $P = T_n - (C + 0.5\,P)$, reducing to $P = \frac{2}{3}(T_n - C)$, and the = $\frac{2}{3}(T_n - C) \times V \div 33\,000$, where V = rope veloc, ft per min. The expression centrifugal force is $C = Wv^2 \div 32.16$, W being the wt of rope per ft and v the veloc, er sec. With fixed rope speed and min diam of sheave it is more difficult to design than belt drives. If the desired rev per min can not be obtained with a certain rope n, the rope diam may have to be reduced, so as to use a smaller sheave. To trans- the same power, this requires more ropes, increasing wear and maintenance. It is l practice to use few ropes of large diam; 1.75-in rope is a good commercial size.

omparison of power drives. Gearing is used for minimum center distances; it gives a tive drive, but is noisy. Chain drives are positive, and special forms can be run at speeds; center distances are greater than for gears, less than for belts. Belts can hifted from one pulley to another, can be run at high speeds or at any angle, require l alinement of shafting, are not noisy, but produce electricity, which is sometimes ob- ionable. Rope drives, used for max center distances, are quiet in operation, produce electricity, do not require accurate alinement of shafting, can be run at any angle, spy less space than belts, and are cheaper than belting for heavy power and long er distances.

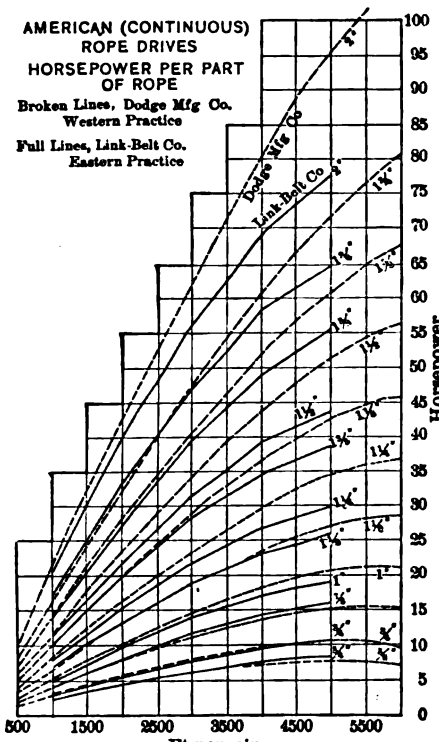


Fig 32. (L. DeG. Moss)

Wire-rope drive (Fig 33), consisting of a single loop of wire rope around a pair of pulleys, can be used for distances of 70 to 400 ft between sheave centers. The sheave grooves are "filled," and the rope should not touch the sides of the grooves, thus requiring good alinement.

Driving power depends upon coeff of friction between rope and sheave filling. (For wire ropes, see Sec 12, Art 11.) 7-wire rope will stand more wear, but requires larger pulleys. Steel rope is stronger and has a longer life, but the size can not be reduced due to the wt required; a small rope must have more tension and damages the pulley lining. Bending stresses (Sec 12, Art 14) generally exceed those due to direct pull. Speed limit is 5 000 ft per min. If the rope lashes at high speed the direct tension may be momentarily doubled, and if the sheaves are small, the bending stresses plus the tension may strain the rope beyond the elastic limit.

Sheaves should be as large as practicable, to give high rope speed and hence permit use of min diam of rope. Fig 34 shows sheave groove and filling. Filling consists of alternate blocks of leather and rubber; rubber has high coef of friction, while the leather has resisting qualities. Hard-wood blocks are sometimes used. 5 000 ft per min rim speed is about the safe limit for C-I sheaves or for the filling.

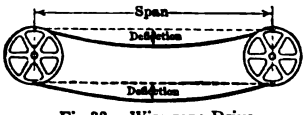


Fig 33. Wire-rope Drive

Fig 34. Filling for Wire-rope Drive Sheaves

Sheaves must be accurately balanced, to prevent lashing of the rope. At less than 70 ft center distance, tension must be excessive, increasing wear on rope and filling. For spans greater than 400 ft, multiple loops may be employed. Where there are sudden variations in load, a flywheel can be attached to the sheave shaft. For changing direction of the drive, bevel or friction gears are used. The heavier the rope, the greater the tension and farther apart may the sheaves be placed, with smaller sag or deflection. Makers advise a sag of 1/30 the span in both ropes when idle; in operation, the sag on tight side is about 0.02 the span and on the slack side 0.04. This makes the tension

ıbout 2 : 1; or difference in tensions may be computed by taking the wt of 1 rope en sheaves and multiplying by 3. The hp can then be found: hp = $T \times V \div$), T being difference in tensions and V veloc, ft per min. No deductions are made ntrifugal force or bending stresses, as they are negligible under usual conditions. sheaves may be used to prevent the rope from striking obstructions. When on llowing side, they are 0.5 the diam of the driver or driven wheel; if in the middle ɔan, the tension and power transmitted will be halved. Tightener sheaves may be l on short spans, but are not recommended. Sheaves are sometimes mounted on ɔle bearings, for taking up slack without resplicing. Catalogues of makers give of hp transmitted, center distances, diam of sheaves, and rev per min for differ- :e ropes.

6. Lubricants

quirements. Lubricant must have sufficient BODY at the probable temp ıich it will be subjected to prevent it from being squeezed out by press be- ı bearing surfaces; it must not decompose nor evaporate at ordinary work- :mp; it must contain no acid which may injure bearing surfaces; its coeff :tion must be low; it must be as fluid as is consistent with the required ; and it must not freeze at any temp to which the bearing may be exposed.

dinary lubricants: animal fats, vegetable and mineral oils, flake graphite, tone, powdered mica.

mal fats or oils are often used for bearings subjected to low temp, but are generally with mineral oils. They decompose at moderate temp and set free acid. Vegetable e used where the bearing press is light and temp low. Mineral oils are obtainable practically all conditions of lubrication; they do not easily decompose and contain d. Greases are mixtures of mineral oil and animal or vegetable soap; useful for ıoving bearings under heavy press.

ıid lubricants, used alone, fill the pores of the bearing metal and form a film They are commonly mixed with oil or grease, and generally improve ation. They must not be used for ball bearings, as they build up a surface : ball race which often splits the bearing.

ting lubricants. The value of ordinary tests are relative rather than absolute, ils, showing approx the same characteristics, will probably serve equally well in me bearing. Oils are tested for the following characteristics. Special machines :d for coeff of friction tests. Other tests are for adulteration and presence of acids. ITY, or degree of fluidity, is a measure of the body of an oil. It is determined :osimeters. Nearly all operate on the principle of finding the time required for a :olume of oil to flow through an orifice. Their design varies greatly, and 2 differ- truments may give widely different results for the same oil. Viscosity is generally :ed as specific or as relative viscosity (Table 3), which denotes the relation between ıes required for the same volume of oil and water to flow through the orifice. Tests de at different temps. FLASH POINT is the temp at which an oil will give off enough :o ignite and flash, when a flame is held over the surface. The heating should be ɔwly. Types of apparatus are the open cup and closed cup. The closed cup has ıing in the cover, closed by a slide. As the oil is heated the slide is periodically d, and a flame inserted until the flash point is reached. During tests a thermom- hung in the body of the oil. BURNING POINT is the temp at which an oil will give ugh vapor to ignite and continue to burn. CHILL POINT is the temp at which an ses. Best determined by placing the oil in a test tube with a thermometer. The ben frozen by an ice and salt mixture. After freezing the test tube is removed ıe ice and the temp observed at which the oil melts. FRICTION TESTS are made by machines, designed to reproduce working conditions, and to measure the temp of and coeff of friction. The speed and press between bearing surfaces may be t will. The simpler machines give only comparative results, but valuable data ten obtained from specially designed apparatus.

ıgtion of adulterants. High-grade oil is sometimes adulterated with low-flash oil, which is detected in flash-test apparatus. Presence of vegetable or animal

oil in mineral oils is determined by mixing a sample with a solution of sodium or pot sium hydroxide and heating to about 200° F. Animal or vegetable oils will saponi and separate from the mineral oil. Presence of mineral oils in animal or vegeta oils is determined by the same process, as the mineral oil will not saponify. Aci detected by litmus paper.

Table 3. Characteristics of Lubricating Oils

Kind of oil	Specific viscosity, 70° F	Specific grav, water=1	Flash point, deg F	Burn point, deg F
Heavy engine and machine	185	0.875	400	445
Ordinary engine and machine	180	0.873	410	460
Light machine	100–150	0.870	400	445
High-grade cylinder	175–200	0.903	600	650
Ordinary cylinder	185	0.899	570	625
Gas engine	300	0.893	320	355
Automobile cylinder	195	0.877	430	485

Most of the above oils have a chill point slightly below freezing point of water may be obtained the chill point of which is considerably lower.

Selection of lubricants. For engine cylinders using high-press steam, h grade cyl oil; for engine cylinders using steam below 100 lb press, second-gr cyl oil; for stationary gas engines, a heavy oil with high flash and burning po (it should preferably have a liquid base); for air compressors and high-s internal combustion engines, a liquid-base oil with a relatively low viscosity a high flash point. Engine and machine oils are sold in many grades to su conditions. For slow-moving bearings under heavy press, use grease. If press is exceptionally heavy, add graphite to the grease.

7. Piping

List sizes of welded W-I or steel pipe indicate the nominal inside di Actual diam varies considerably from the nominal, especially in standard wt of small sizes. Outside diam of all weights is the same, so that the threads be uniform for same nominal size. Added thickness of metal reduces internal diam. Ordinary welded pipe is of mild steel. If iron pipe be desi it should be specified. Tubes are listed according to their actual outside di Pipe is regularly supplied in random lengths, from 18 to 22 ft, threaded at ends, and 1 coupling is furnished with each length. An extra charge is ma pipe in equal lengths. Prices vary with the metal market, and the plan va

Approx discounts from list prices in New York, 1915: standard pipe (slightly higher in small sizes and lower in large); extra-heavy pipe, double extra-heavy, 43%; spiral-riveted pipe (N Y, 1916), 40%; screwed fittings, standard wt, 70%; extra-heavy and double extra-heavy screwed fi 65%; flanged fittings, all wts, 65%. Discounts on valves are from 40 to Prices on pipe coverings and insulating material are practically net.

In the following tables a fairly complete list of pipes and fittings is given, but va and cocks are omitted as they are made in too great variety to be listed here. Th catalogues should be consulted regarding them.

Table 4. Line Pipe, for Oil, Gas, and High-press Mains

inal de n,	Actual outside diam, in	Wt per ft, lb	No of threads per in of screw	Nominal inside diam, in	Actual outside diam, in	Wt per ft, lb	No of threads per in of screw
	2.37	3.61	11½	6	6.62	18.76	8
¼	2.87	5.74	8	7	7.62	23.27	8
	3.50	7.54	8	8	8.62	28.18	8
½	4.00	9.00	8	9	9.68	33.70	8
	4.50	10.66	8	10	10.75	40.06	8
½	5.00	12.34	8	12	12.75	49.00	8
	5.56	14.50	8	.....	.....	.....	.....

Table 5. Root's Spiral-riveted Pipe

t and asphalted pipe is in lengths of 25 ft or less, galvanized in lengths of 20 ft.
pprox for black pipe only; asphalted and galvanized weigh 10 to 20% heavier.
s, $ per linear ft, with plain, crimped, or sleeved ends.
es when furnished are included in linear measurement.

No 24, B W G					No 20, B W G					
Prices			Wt per 100 ft, lb	Approx bursting press per sq in, lb	Diam, in	Prices			Wt per 100 ft, lb	Approx bursting press per sq in, lb
Black	Asphalted	Galvanized				Black	Asphalted	Galvanized		
0.20	0.23	0.30	100	.....	3	0.27	0.30	0.38	150	900
0.25	0.29	0.38	130	.....	4	0.35	0.39	0.48	200	700
0.30	0.35	0.45	160	.....	5	0.40	0.45	0.60	250	550
0.33	0.39	0.50	185	.....	6	0.46	0.52	0.68	300	450
0.37	0.44	0.60	210	.....	7	0.51	0.58	0.75	325	400
0.42	0.50	0.65	240	.....	8	0.58	0.66	0.85	360	350
0.48	0.57	0.75	280	.....	9	0.66	0.75	0.97	410	325
0.54	0.64	0.85	300	.....	10	0.72	0.82	1.05	500	275
0.60	0.71	0.90	330	.....	11	0.78	0.89	1.20	550	250
0.68	0.80	1.05	400	.....	12	0.90	1.02	1.35	600	225
No 18, B W G					No 16, B W G					
0.34	0.37	0.46	185	1 300	4	0.50	0.54	0.70	320	1 250
0.42	0.46	0.58	245	1 000	5	0.60	0.65	0.85	415	1 000
0.50	0.55	0.70	300	800	6	0.70	0.76	1.00	500	800
0.57	0.63	0.85	360	700	7	0.80	0.87	1.10	550	700
0.63	0.70	0.90	400	600	8	0.93	1.01	1.28	650	600
0.73	0.81	1.05	460	500	9	1.08	1.17	1.47	750	550
0.82	0.91	1.18	525	450	10	1.15	1.25	1.55	800	500
0.90	1.00	1.30	575	400	11	1.20	1.31	1.70	850	450
0.95	1.06	1.40	625	360	12	1.45	1.57	2.05	1025	400
No 14, B W G					No 12, B W G					
0.89	0.95	1.15	610	1 100	6	1.25	1.31	1.90	800	1 330
1.02	1.09	1.35	700	950	7	1.40	1.47	2.10	910	1 140
1.15	1.23	1.50	825	825	8	1.55	1.63	2.30	1 040	1 000
1.32	1.41	1.70	925	750	9	1.70	1.79	2.50	1 180	880
1.40	1.50	1.80	1 025	650	10	1.90	2.00	2.75	1 300	800
1.50	1.61	1.95	1 125	600	11	2.25	2.36	3.00	1 425	725
1.80	1.92	2.35	1 325	550	12	2.50	2.62	3.25	1 700	660

— Straight-seam riveted pipe is obtainable in form of punched and formed sheets,
ices ranging from 7 to 8.5¢ per lb black and 10 to 13¢ per lb galvanized, depend
ickness of metal.

Table 6. Wrought-iron Welded Steam, Gas, and Water Pipe

1 in and below, proved to 300 lb per sq in, hydraulic press; 1¼ in and above, proved 500 lb per sq in.

Nominal inside diam, in	Actual inside diam, in	Actual outside diam, in	Thickness, in	Length per sq ft inside surf, ft	Length per sq ft outside surf, ft	Inside sec area, sq in	Outside sec area, sq in	Length containing 1 cu ft, ft	Nominal wt per ft, lb	No of threads per in of screw	Contents of 1 ft length, gal	Price per ft, steam, $
⅛	0.27	0.40	0.07	14.15	9.44	0.06	0.12	2500	0.24	27	0.002	[illegible]
¼	0.36	0.54	0.08	10.50	7.07	0.10	0.22	1385	0.42	18	0.002	[illegible]
⅜	0.49	0.67	0.09	7.67	5.65	0.19	0.35	751.5	0.56	18	0.005	[illegible]
½	0.62	0.84	0.10	6.13	4.50	0.30	0.55	472.4	0.84	14	0.010	[illegible]
¾	0.82	1.05	0.11	4.63	3.63	0.53	0.86	270.0	1.12	14	0.023	[illegible]
1	1.04	1.31	0.13	3.67	2.90	0.86	1.35	166.9	1.67	11½	0.040	[illegible]
1¼	1.38	1.66	0.14	2.76	2.30	1.49	3.16	96.25	2.24	11½	0.063	[illegible]
1½	1.61	1.9	0.14	2.37	2.01	2.03	2.83	70.65	2.68	11½	0.091	[illegible]
2	2.06	2.37	0.15	1.84	1.61	3.35	4.43	42.36	3.61	11½	0.163	[illegible]
2½	2.46	2.87	0.20	1.54	1.32	4.78	6.49	30.11	5.74	8	0.255	[illegible]
3	3.06	3.5	0.21	1.24	1.09	7.38	9.62	19.49	7.54	8	0.367	[illegible]
3½	3.56	4.0	0.22	1.07	0.95	9.83	12.56	14.56	9.00	8	0.500	[illegible]
4	4.02	4.5	0.23	0.94	0.84	12.73	15.90	11.31	10.66	8	0.652	[illegible]
4½	4.50	5.0	0.24	0.84	0.76	15.93	19.63	9.03	12.34	8	0.825	[illegible]
5	5.04	5.56	0.25	0.75	0.62	19.99	24.29	7.20	14.50	8	1.02	[illegible]
6	6.06	6.62	0.28	0.63	0.57	28.88	34.47	4.98	18.76	8	1.46	[illegible]
7	7.02	7.62	0.30	0.54	0.50	38.73	45.66	3.72	23.27	8	2.00	[illegible]
8	7.98	8.62	0.32	0.47	0.44	50.03	58.42	2.88	28.18	8	2.61	[illegible]
9	9.00	9.68	0.34	0.42	0.30	63.63	73.71	2.26	33.70	8	3.30	[illegible]
10	10.01	10.75	0.36	0.38	0.35	78.83	90.79	1.80	40.06	8	4.06	[illegible]
11	11	11.75	0.37	0.34	0.32	95.03	108.43	1.50	45.0	8	4.93	[illegible]
12	12	12.75	0.37	0.32	0.30	113.09	127.67	1.27	49.0	8	5.87	[illegible]
13	13.25	14.0	0.37	0.29	0.27	137.88	153.94	1.04	54.0	8	6.89	
14	14.25	15.0	0.37	0.27	0.25	159.48	176.71	0.90	58.0	8	8.00	

Table 7. Standard Lap-welded Boiler Tubes, of Standard Gage, not Over 20 ft Long

Actual outside diam, in	Price per ft, $	Thickness, in	Thickness, nearest Birm W G	Nom wt per ft, lb	Actual outside diam, in	Price per ft, $	Thickness, in	Thickness, nearest Birm W G
1	0.30	0.095	13	0.90	4½	0.62	0.134	10
1¼	0.28	0.095	13	1.15	5	0.75	0.148	9
1½	0.27	0.095	13	1.40	6	1.00	0.165	8
1¾	0.22	0.095	13	1.66	7	1.20	0.165	8
2	0.20	0.095	13	1.91	8	1.50	0.165	8
2¼	0.24	0.095	13	2.16	9	1.70	0.180	7
2½	0.28	0.109	12	2.75	10	2.10	0.203	6
2¾	0.34	0.109	12	3.04	11	2.50	0.220	5
3	0.35	0.109	12	3.33	12	2.90	0.229	4
3¼	0.40	0.120	11	3.96	13	3.20	0.238	4
3½	0.44	0.120	11	4.28	14	3.65	0.248	3½
3¾	0.50	0.120	11	4.60	15	4.10	0.259	3
4	0.55	0.134	10	5.47	16	4.60	0.270	2½

Tubes may be obtained of other thicknesses than those given above.

Table 8. Extra Strong and Double Extra Strong W-I Welded Pipe

Extra Strong Steam, Gas, and Water Pipe					Double Extra Strong Steam, Gas, and Water Pipe					
Actual outside diam, in	Nominal inside diam, in	Thickness, in	Nominal wt per ft, lb	Price per ft, $	Size, in	Actual outside diam, in	Nominal inside diam, in	Thickness, in	Nominal wt per ft, lb	Price per ft, $
0.405	0.205	0.100	0 29	0.11	½	0.84	0.244	0.298	1.70	0.25
0.540	0.294	0.123	0.54	0.11	¾	1.05	0.422	0 314	2.44	0.30
0 675	0.421	0.127	0.74	0.11	1	1.315	0.587	0.364	3.65	0.37
0.840	0.542	0.149	1.09	0.12	1¼	1.66	0.885	0.388	5.20	0.52
1.05	0.736	0.157	1.39	0.15	1½	1.90	1.088	0.406	6.40	0.65
1.315	0.951	0.182	2.17	0.22	2	2.375	1.491	0.442	9.02	0.95
1.66	1.272	0.194	3.00	0.30	2½	2.875	1.775	0.560	13.68	1.37
1.90	1.494	0.203	3.63	0.36	3	3.50	2.284	0 608	18.56	1.92
2.375	1.933	0.221	5.02	0.50	3½	4.00	2.716	0.642	22.75	2.45
2.875	2.315	0.280	7.67	0.81	4	4.50	3.136	0.682	27.48	2.85
3.500	2.892	0.304	10.25	1.05	4½	5.00	3.564	0.718	32.53	3 30
4.000	3.358	0.321	12.47	1.33	5	5.563	4.063	0.750	38.12	3.80
4.500	3.818	0.341	14.97	1.50	6	6.625	4.875	0.875	53.11	5.30
5.000	4.280	0.360	18.22	1.95	7	7.625	5.875	0.875	62.38	6.25
5.563	4.813	0.375	20.54	2.16	8	8.625	6.875	0.875	71.62	7.20
6.625	5.750	0.437	28.58	2.90	...	.....	.....	.....	....	....
7.625	6.625	0.500	37.67	3.80	...	.....	.....	.....	....	....
8.625	7.625	0.500	43.00	4.30	...	.....	.....	.....	....	....

ra strong and double extra strong will be shipped in random lengths and plain ends otherwise ordered. For pipe fitted with threads and couplings, extra charge will ıde above regular. For cut lengths, extra charge will be made above random. ılvanized or asphalted, extra charge will be made above black.

Table 9. Spiral-riveted Pipe

)rox bursting strength, lb per sq in. (Amer Spiral Pipe Works, Chicago, 1915)

ınal n,	Thickness, U S Standard Gage								
	No 20	No 18	No 16	No 14	No 12	No 10	No 8	No 6	No 3 (¼ in)
	1 125	1 500	1 875	....	.....	.....	.....	.....	.....
	900	1 200	1 500	....	.....	.....	.....	.....	.....
	....	1 000	1 250	1 560	2 170	.....	.....	.....	.....
	....	860	1 070	1 340	1 860	.....	.....	.....	.....
	....	750	935	1 170	1 640	.....	.....	.....	.....
	....	....	835	1 045	1 460	.....	.....	.....	.....
	....	....	750	935	1 310	.....	.....	.....	.....
	....	....	680	850	1 200	.....	.....	.....	.....
	....	....	625	780	1 080	1 410	.....	.....	.....
	....	....	575	720	1 010	1 295	.....	.....	.....
	....	....	535	670	940	1 210	.....	.....	.....
	....	....	....	625	875	1 125	.....	.....	.....
	....	....	....	585	820	1 050	1 290	1 520	1 880
	....	....	....	520	730	940	1 140	1 360	1 660
	....	....	....	470	660	840	1 030	1 220	1 500
	....	....	....	425	595	765	940	1 108	1 364
	....	....	....	390	540	705	820	1 015	1 250
	....	....	....	....	505	650	795	935	1 154
	....	....	....	....	470	605	735	870	1 071
	....	....	....	....	435	560	685	810	1 000
	....	....	....	....	410	525	645	760	940
	....	....	....	....	380	490	600	715	88
	....	....	....	....	365	470	570	680	8

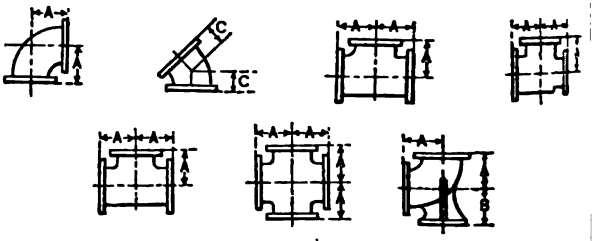
Table 10. C-I Water Pipe, Standard Weight

Size.....in	3	4	6	8	10	12	14	16	18	20	24	30	
Thickness.....in	½	½	½	½	9/16	9/16	¾	¾	⅞	⅞	1	1 1/8	1
Weight per ft.....lb	17	22	33	42	60	75	117	125	167	197	250	350	4

Weights given are standard, but can be changed when pipe is made to order. Coa inside and out, and tested 300 lb per sq in, hydrostatic press. Made in 12-ft lengt 440 lengths per mile.

8. Pipe Fittings and Coverings

Table 11. Standard C-I Flanged Fittings Suitable for 125 lb working pres



Size of run.....	In	2	2½	3	3½	4	4½	5	6	7	8	
Outlets.....	In	All reducing fittings, 2 in to 9 in, inclusive, have same dimensions as straight fittings, center to face										
Face to face.....	AA	9	10	11	12	13	14	15	16	17	18	
Center to face.....	A	4½	5	5½	6	6½	7	7½	8	8½	9	
Center to face of 45° ells.	C	2½	3	3	3½	4	4	4½	5	5½	5½	
Center to base of base ells	B	5	5½	5¾	6¼	6½	6¾	7	7½	8¼	8½	
Diam flanges,.....	....	6	7	7½	8½	9	9½	10	11	12½	13	
Thickness flanges.....	....	5/8	11/16	¾	13/16	15/16	1	1 1/16	1 1/16	1	1 1/16	1

Size of run.....	In	10	12	14	15	16
Outlets.....	In	7 and larger	9 and larger	10 and larger	10 and larger	12 and larger
Face to face.....	AA	22	24	28	29	30
Center to face.....	A	11	12	14	14½	15
Center to face of 45° ells..	C	6½	7½	7½	8	8
Center to base of base ells	B	10	10½	13½	14	14
Diam flanges.....	....	16	19	21	22¼	23½
Thickness flanges.....	....	1 3/16	1¼	1⅜	1⅜	1⅜

Tees having outlets larger than the run have same length center to face of all open as a tee with all openings of same size as outlet.

Table 12. Prices of Standard C-I Fittings

.....in	¼	⅜	½	¾	1	1¼	1½	2	2½	3	3½	4
ws, R H.....each	0.05	0.05	0.06	0.08	0.10½	0.16	0.20	0.28	0.50	0.75	1.05	1.20
R and L.. "	0.06	0.06	0.07	0.09	0.12	0.18	0.23	0.32	0.60	0.85	1.20	1.50
L H..... "	0.06	0.06	0.07	0.09	0.12	0.18	0.23	0.32	0.60	0.85	.....	.....
reducing.. "	....	0.06	0.07	0.09	0.12	0.18	0.23	0.32	0.60	0.85	1.20	1.40
pitched... "	....	....	....	0.10	0.13	0.20	0.25	0.35	0.65	1.00	1.30	1.50
with side outlet.. "	....	....	0.18	0.24	0.30	0.48	0.60	0.84	1.50	2.25	3.15	3.60
bows..... "	0.06	0.06	0.07	0.10	0.12	0.19	0.24	0.34	0.60	0.90	1.25	1.45

.....in	4½	5	6	7	8	9	10	12	14	15
ws, R H.....each	1.75	2.00	2.75	4.70	6.75	9.00	13.50	20.00	57.00	70.00
R and L.. "	2.25	2.40	3.40	.....	.....	.....	.....	.....	.....	.....
reducing.. "	2 00	2.30	3.15	5.40	7.75	10.50	15.50	23.00	63.00	77.00
with side. outlet. "	5.25	6.00	8.25	14.00	20.00	26.00	40.00	60.00	.....	.....
bows..... "	2.20	2.50	3.45	5.90	8.50	11.25	17.00	25.00	55.00	68.00

Table 13. Prices of Extra Heavy C-I Fittings

.....in	¾	1	1¼	1½	2	2½	3	3½
ws .....each	0.30	0.36	0.52	0.60	1.02	1.70	2.60	3.30
reducing..... "	0.36	0.44	0.60	0.70	1.20	2.00	3.00	4.00
45°..... "	0.42	0.50	0.60	0.72	1.20	1.70	3.00	3.80
..... "	0.36	0.50	0.70	0.90	1.20	2.60	3.00	4.80
reducing..... "	0.50	0.60	0.80	1.10	1.40	3.00	3.50	5.70

.....in	4	4½	5	6	7	8	10	12
ws .....each	4.20	5.50	6.80	10.60	14.00	18.50	30.00	56.00
reducing..... "	5.00	6.50	8.00	12.50	16.50	21.50	35.00	65.00
45°..... "	4.60	6.00	7.30	11.20	16.00	21.00	34.00	60.00
..... "	6.00	7.50	9.20	12.60	21.00	30.00	45.00	80.00
reducing..... "	7.00	9.00	10.80	14.60	25.00	35.00	52.00	92.00

.....in	¾	1	1¼	1½	2	2½	3	3½
es ... .each	0.90	0.90	1.40	1.60	1.80	4.00	5.00	6.00
reducing..... "	1.10	1.10	1.60	1.90	2.10	4.80	6.00	7.00
e unions..... "	2.00	2.00	2.10	2.60	2.90	3.30	4.40	5.40
..... "	0.10	0.12	0.16	0.22	0.32	0.56	0.80	1.20

....in	4	4½	5	6	7	8	10	12
es. .....each	7.50	10.00	11.00	18.00	24.00	36.00	60.00	120.00
reducing..... "	9.00	12.00	13.00	21.50	29.00	43.00	72.00	144.00
e unions. ..... "	6.00	7.00	7.50	9.00	13.50	14.00	21.00	31.50
..... "	1.35	2.15	2.75	3.75	6.00	8.60	11.75	15.60

le extra heavy fittings are obtainable at list prices slightly above those for the avy.

Table 14. C-I Fittings. Standard and low-press flanged valves, flanged fittings and flanges, suitable for 125 lb working press

Size, in	Diam of flanges, in	Bolt circle, in	Number bolts	Size bolts, in	Length bolts for standard, in	Size, in	Diam of flanges, in	Bolt circle, in	Number bolts	Size bolts, in	Length bolts for standard, in
1	4	3	4	7/16	1½	10	16	14¼	12	⅞	3
1¼	4½	3⅜	4	7/16	1½	12	19	17	12	⅞	3
1½	5	3⅞	4	½	1¾	14	21	18¾	12	1	4
2	6	4¾	4	⅝	2	15	22½	20	16	1	4
2½	7	5½	4	⅝	2¼	16	23½	21¼	16	1	4
3	7½	6	4	⅝	2½	18	25	22¾	16	1⅛	4½
3½	8½	7	4	⅝	2½	20	27½	25	20	1⅛	5
4	9	7½	4	¾	2¾	22	29½	27¼	20	1¼	5
4½	9¼	7¾	8	¾	3	24	32	29½	20	1¼	5½
5	10	8½	8	¾	3	26	34¼	31¾	24	1¼	5½
6	11	9½	8	¾	3	28	36½	34	28	1¼	6
7	12½	10¾	8	¾	3¼	30	38¾	36	28	1¼	6½
8	13½	11¾	8	¾	3½	36	45¾	42¾	32	1⅜	6½
9	15	13¼	12	¾	3½	....	.....	....	....	....	....

Table 15. Prices of Standard Flanged Fittings

Elbows					Tees				
Size, in	Center to face, in	Diam of flanges, in	With flanges faced	With flanges faced and drilled	Center to face, in	Face to face, in	Diam of flanges, in	With flanges faced	With flanges faced and drilled
2	4½	6	$4.75	$5.75	4½	9	6	$7.00	$8
2½	5	7	5.00	6.25	5	10	7	7 25	9
3	5½	7½	5.75	7.00	5½	11	7½	8.25	11
3½	6	8½	6.50	7.75	6	12	8½	9 50	12
4	6½	9	7.25	9.25	6½	13	9	10 50	13
4½	7	9¼	9.00	11.00	7	14	9¼	13.00	16
5	7½	10	9.75	11.75	7½	15	10	14.25	17
6	8	11	12.00	14.00	8	16	11	17.50	20
7	8½	12½	16.00	19.75	8½	17	12½	23 00	26
8	9	13½	20.00	23.75	9	18	13½	29.00	33
9	10	15	26.00	30.00	10	20	15	38 00	44
10	11	16	32.00	36.00	11	22	16	46 50	52
12	12	19	44.00	50.00	12	24	19	64 00	73

Table 16. Mineral Wool (net price, 1916)

Averages	Lb per cu ft	Lb per sq ft, 1 in thick	Cu ft per ton	Cost per 100 lb (in ton lots) at factory	Cost per cu ft at factory: In ton lots
Ordinary slag wool	12	1	166	$2.00	$0.24
Selected " "	9	¾	223	3.43	0.30
Extra " "	6	½	333	8 00	0 48
Ordinary rock "	12	1	166	4 00	0.48
Selected " "	8	¾	250	8 00	0 64
Extra " "	6	½	333	14 00	0 84

In less than ton lots add 25¢ per 100 lb to factory prices

Table 17. Pipe Covering

de m pe,	Price per linear foot, canvas jacketed	Size of fittings, in	Elbows	Tees	Crosses	Globe valves
½	$0.22	½	$0.30	$0.36	$0.48	$0.54
¾	0.24	¾	0.30	0.36	0.48	0.54
	0.27	1	0.30	0.36	0 48	0 54
1	0.30	1¼	0.30	0.36	0.48	0.54
1½	0.33	1½	0.30	0.36	0.48	0.54
	0.36	2	0.36	0.42	0.54	0.60
2	0.40	2½	0.42	0.48	0.60	0.78
	0.45	3	0.48	0.54	0.70	0.96
2½	0.50	3½	0.54	0.60	0.80	1.20
	0.60	4	0.60	0.75	0.95	1.50
3	0.65	4½	0.72	0.90	1.10	1.85
	0.70	5	0.90	1.20	1.50	2.25
	0.80	6	1.30	1.60	2.00	2.80
	1.00	7	1.80	2.20	2.80	3.60
	1.10	8	2.40	3.00	3.60	4.40
	1.20	9	3.00	3.80	4.40	5.30
	1.30	10	3.60	4.60	5.20	6.20
	1.85	.....	.....	.....	.....	.....
	2.10	.....	.....	.....	.....	.....

ck covering for boilers, drums, and tanks, made with wire g, smooth cement finish: 1.5 in thick, 25¢ per sq ft; 2 in 35¢ per sq ft.

r felt. Per sq ft: 0.5 in, 4.75¢; 0.75 in, 5.75¢; 1 in, 6.5¢; 10¢; 2 in, 13¢.

Table 18. Block Covering

Thickness, in	Price, sq ft
½	$0.27
¾	0.27
⅞	0.30
1	0.30
1⅛	0.34
1¼	0 38
1⅜	0.42
1½	0.45
1⅝	0 49
1¾	0.53
1⅞	0.57
2	0.60
2⅛	0.64
2¼	0.68
2⅜	0.72
2½	0.75
2⅝	0.79
2¾	0.83
2⅞	0.87
3	0.90
3¼	0.98
3½	1.05
3¾	1.13
4	1.20

Any thickness made to order.

e 19. C-I Flanged Fittings, Faced and Drilled, for Spiral-riveted Pipe

),	Elbows, 45° or 90°	Tees	Crosses or Y bends	Taper reducers	Flanges	Disks	Bolts and gaskets, per set
Black							
	$1.25	$1.75	$3.00	.....	$0.24	$0.28	$0.19
	1.50	2.20	3.70	$3.00	0.32	0.40	0.30
	2.00	3.10	4.80	3.50	0.40	0.48	0.32
	2.75	3.90	5.70	4.50	0.48	0.72	0.40
	3.50	5.00	7.70	6.00	0.64	0.96	0.42
	4.50	6.60	9.80	7.00	0.72	1.12	0.47
	6.00	8.75	12.00	9.00	0.96	1.44	0.55
	6.80	10.50	14.50	10.00	1.12	1.62	0.64
	8.75	13.00	18.00	12.00	1.20	2.00	0.81
	10.00	14.50	21.00	13.50	1.28	2.40	0.86
Galvanized							
	$1.60	$2.35	$4.15	.....	$0.39	$0.45	$0.25
	2.10	3.25	5.30	$4.00	0.52	0.65	0.42
	2.85	4.40	6.70	5.00	0.65	0.78	0.44
	3.75	5.50	8.00	6.00	0.78	1.17	0.52
	5.10	7.00	11.00	8.00	1.04	1.56	0.54
	6.50	9.30	13.75	9.50	1.17	1.82	0.59
	8.75	12.50	16.80	12.00	1.56	2.34	0.67
	10.00	15.20	21.00	13.50	1.82	2.47	0.76
	12.50	18.20	25.00	15.00	1.95	3.25	0.9
	14.00	21.00	30.00	17.00	2.08	3.90	1.0

9. Wire Gages, Bolts, Rivets, Spikes

Table 20. Wire Gage Standards Used in United States

No of gage	American or Brown & Sharpe	Birmingham or Stubs'	Washburn & Moen Mfg Co	Imperial Wire gage	Stubs' steel wire gage	U S stand for sheets and plate	Approx th'knes, fractions of in	W: sq ft
0000000	.....	.....	.....	.....	.....	0.50	½	20 00
000000	.....	.....	.....	0.464	.....	0.46875	15/32	18 75
00000	.....	.....	.....	0.432	.....	0.4375	7/16	17 50
0000	0.46	0.454	0.3938	0.400	.....	0.40625	13/32	16 25
000	0.40964	0.425	0.3625	0.372	.....	0.375	3/8	15 00
00	0.3648	0.38	0.3310	0.348	.....	0.34375	11/32	13 75
0	0.32486	0.34	0.3065	0.324	.....	0.3125	5/16	12 50
1	0.2893	0.3	0.2830	0.300	0.227	0.28125	9/32	11 25
2	0.25763	0.284	0.2625	0.276	0.219	0.265625	17/64	10 625
3	0.22942	0.259	0.2437	0.252	0.212	0.25	¼	10 00
4	0.20431	0.238	0.2253	0.232	0.207	0.234375	15/64	9 375
5	0.18194	0.22	0.2070	0.212	0.204	0.21875	7/32	8 75
6	0.16202	0.203	0.1920	0.192	0.201	0.203125	13/64	8 125
7	0.14428	0.18	0.1770	0.176	0.199	0.1875	3/16	7 5
8	0.12849	0.165	0.1620	0.160	0.197	0.171875	11/64	6 875
9	0.11443	0.148	0.1483	0.144	0.194	0.15625	5/32	6 25
10	0.10189	0.134	0.1350	0.128	0.191	0.140625	9/64	5 625
11	0.090742	0.12	0.1205	0.116	0.188	0.125	⅛	5 00
12	0.080808	0.109	0.1055	0.104	0.185	0.109375	7/64	4 375
13	0.071961	0.095	0.0915	0.092	0.182	0.09375	3/32	3 75
14	0.064084	0.083	0.0800	0.080	0.180	0.078125	5/64	3 125
15	0.057068	0.072	0.0720	0.072	0.178	0.0703125	9/128	2 8125
16	0.05082	0.065	0.0625	0.064	0.175	0.0625	1/16	2 5
17	0.045257	0.058	0.0540	0.056	0.172	0.05625	9/160	2 25
18	0.040303	0.049	0.0475	0.048	0.168	0.05	1/20	2
19	0.03589	0.042	0.0410	0.040	0.164	0.04375	7/160	1 75
20	0.031961	0.035	0.0348	0.036	0.161	0.0375	3/80	1 5
21	0.028462	0.032	0.03175	0.032	0.157	0.034375	11/320	1 375
22	0.025347	0.028	0.0286	0.028	0.155	0.03125	1/32	1 25
23	0.022571	0.025	0.0258	0.024	0.153	0.028125	9/320	1 125
24	0.0201	0.022	0.0230	0.022	0.151	0.025	1/40	1
25	0.0179	0.02	0.0204	0.020	0.148	0.021875	7/320	0 875
26	0.01594	0.018	0.0181	0.018	0.146	0.01875	3/160	0 75
27	0.014195	0.016	0.0173	0.0164	0.143	0.0171875	11/640	0 6875
28	0.012641	0.014	0.0162	0.0149	0.139	0.015625	1/64	0 625
29	0.011257	0.013	0.0150	0.0136	0.134	0.0140625	9/640	0 5625
30	0.010025	0.012	0.0140	0.0124	0.127	0.0125	1/80	0 5
31	0.008928	0.01	0.0132	0.0116	0.120	0.0109375	7/640	0 4375
32	0.00795	0.009	0.0128	0.0108	0.115	0.0101562 5	13/1280	0 4062
33	0.00708	0.008	0.0118	0.0100	0.112	0.009375	3/320	0 375
34	0.006304	0.007	0.0104	0.0092	0.110	0.00859375	11/1280	0 34
35	0.005614	0.005	0.0095	0.0084	0.108	0.0078125	5/640	0 31
36	0.005	0.004	0.0090	0.0076	0.106	0.0070312 5	9/1280	0 28
37	0.004453	.....	.....	0.0068	0.103	0.0066406 25	17/2560	0 265
38	0.003965	.....	.....	0.0060	0.101	0.00625	1/160	0 25

Table 21. Corrugated Iron or Steel Plates

Wt, lb per 100 sq ft (Amer Sheet and Tin Plate Co)

rugations	5/8 in		1¼ × 3/8 in		2 × ½ in		2½ × ½ in		3 × ¾ in		5 × ⅞ in	
standard et metal gage	Painted	Galvanized	Painted	Galvanized	Painted	Galvanized	Painted	Galvanized	Painted	Galvanized	Painted	Galvanized
28	72	87	72	87	68	85	68	85	68	85	68	85
27	79	94	79	94	76	91	76	91	76	91	76	91
26	86	101	86	101	83	98	83	98	83	98	83	98
25	100	115	100	115	96	111	96	111	96	111	96	111
24	114	129	114	129	110	125	110	124	110	124	110	124
23	.....	.....	128	143	123	138	123	138	123	138	123	138
22	.....	.....	142	157	136	151	136	151	136	151	136	151
21	.....	.....	156	171	150	165	150	165	150	165	150	165
20	.....	.....	170	185	163	178	163	178	163	178	163	178
18	.....	.....	.....	.....	217	232	217	232	217	232	217	232
16	.....	.....	.....	.....	271	286	271	286	271	286	271	286

Table 22. Square-head Bolts, Wt in Lb per 100 (Hoopes & Townsend)

m.	3/8	7/16	½	9/16	5/8	¾	⅞	1	1⅛	1¼	1⅜	1½	1¾	2
gth, in														
"	9.7	14.7	20.4	26.0	37.0	58.0	.....	.....	.....	.....	.....	.....	.....	.....
"	11.3	16.5	22.4	29.0	39.9	63.2	97.7	145	.....	.....	.....	.....	.....	.....
"	12.9	18.5	25.0	32.2	44.1	69.0	105.6	153	.....	.....	.....	.....	.....	.....
"	14.5	20.5	27.8	35.4	48.3	75.2	113.8	163	240	309	350	480	.....	.....
"	16.1	22.6	30.6	38.7	52.5	81.4	122.0	174	253	325	370	500	.....	.....
"	17.7	24.7	33.4	42.0	56.7	87.6	130.2	185	267	342	390	520	800	.....
"	19.2	26.8	36.2	45.3	60.9	93.8	138.4	196	281	359	410	545	833	.....
"	20.7	28.9	39.0	48.6	65.1	100.0	146.6	207	295	376	430	570	866	1 370
"	22.2	31.0	41.8	51.9	69.2	106.1	154.9	218	309	394	450	595	900	1 414
"	23.7	33.1	44.6	55.2	73.4	112.2	163.2	229	323	412	470	620	934	1 458
"	25.2	35.2	47.4	58.5	77.6	118.3	171.5	240	337	430	490	645	968	1 502
"	26.7	37.3	50.2	61.8	81.8	124.4	179.8	251	351	448	510	670	1 002	1 546
"	28.2	39.4	53.1	65.1	86.0	130.5	187.1	262	365	466	530	695	1 036	1 590
"	29.7	41.5	56.0	68.5	90.0	136.6	195.4	273	379	484	550	725	1 070	1 634
"	33.1	45.7	61.5	75.2	98.0	148.8	212.0	295	407	518	590	775	1 138	1 722
"	36.5	49.9	67.0	81.9	106.3	161.0	229.0	317	435	552	630	825	1 206	1 810
"	40.0	54.1	72.5	88.7	114.6	173.2	246.0	339	463	586	670	875	1 274	1 898
"	43.5	58.3	78.0	95.5	122.9	184.4	263.0	361	491	620	710	925	1 342	1 986
"	.....	.....	83.5	102.3	131.2	196.6	280.0	383	519	655	751	975	1 410	2 074
"	.....	.....	89.0	109.1	139.5	208.8	297.0	405	547	690	793	1 025	1 478	2 162
"	.....	.....	94.5	116.0	148.0	221.0	314.0	427	575	725	835	1 075	1 548	2 250
"	.....	.....	100.0	123.0	156.5	233.2	331.0	449	603	760	877	1 125	1 616	2 338
"	.....	.....	105.5	130.0	165.0	245.4	348.0	471	631	795	919	1 175	1 684	2 426
"	.....	.....	111.0	137.0	173.5	257.6	365.0	493	659	830	961	1 225	1 752	2 514
"	.....	.....	116.5	144.0	182.0	269.8	382.0	515	687	865	1 003	1 275	1 820	2 602
"	.....	.....	122.0	151.0	190.5	282.0	399.0	537	715	900	1 045	1 325	1 888	2 690

[redacted]

[redacted]

[redacted]

[redacted]

10. Springs

mulas relating to springs are based on torsion formulas, modified according to of spring and cross-section of the wire or rod. The following brief notes relate ially to springs of hoisting cages.

lical **spring**, of round wire, wn in Fig 35; Fig 36 illus- its action. A load, P lb, is ed by a cord to the rim of a , of a diam D = mean diam spring coils, and produces n in the rod. A line AB, ally straight, becomes a AC, length of rod being l. between AB and AC is the of shear ϕ, and angle BOC s the angle of twist. For rium the external and in- moments must be equal,

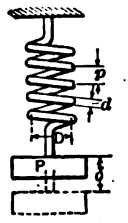


Fig 35. Helical Spring

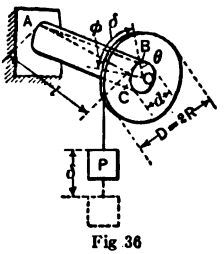


Fig 36

(Fig 36) $P\frac{D}{2} = f_s Z_p = f_s \frac{\pi d^3}{16}$, where Z_p equals section modulus (Sec 43, Art 4) d from the polar moment of inertia, and d equals diam of wire or rod. From the on, the load P which the spring will carry can be determined for any given fiber f_s by transposition, $P = \frac{2\pi d^3}{16 D} f_s = \frac{\pi d^3}{16 R} f_s$, all dimensions being in inches. The of a coil of the spring is generally taken as πD; accurately, it is $\sqrt{(\pi D)^2 + p^2}$. On ng the load, the rod is twisted through an angle θ; or through a length of arc red on the circumference of $D = \delta$, which is the moment of the weight P. If the ce from B to C measured on the rod circumference = x, then

$$\frac{x}{l} = \frac{\text{Shear stress}}{\text{Modulus of rigidity}} = \frac{f_s}{G}.$$

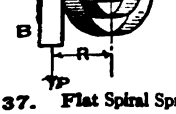
$\phi = x + l$ and angle $\theta = x + r$ or $2x + d$; hence, $\theta = 2 f_s + Gd$, and $\delta = R\theta l = Gd$. Since $f_s = 16 PR + d^3$, $\delta = 32 PR^2 l + Gd^4$, which gives the deflection of ing, where $l = 2\pi Rn$, if n = the number of complete coils. The resilience = 0.5 oviding the spring is not too open.

rectangular wire or rod, the max stress occurs at centers of the longer sides. If b = ss and h = width of section of wire,

$$I_p = \frac{b^3 h^3}{3(b^2 + h^2)}, \text{ and } Z_p = \frac{b^3 h^3}{3\sqrt{h^2 + b^2}}.$$

ing the same analysis, $P = \frac{f_s}{3R} \frac{b^3 h^2}{\sqrt{b^2 + h^2}}$, and $\delta = \frac{3 PR^2 l}{G} \frac{b^2 + h^2}{b^3 h^3}$.

e of square section, $b = h$, $I_p = b^4 \div 6$, and $Z_p = b^3 \div 3\sqrt{2}$.



37. **Flat Spiral Spring**

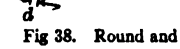


Fig 38. Round and Flat Helical Springs

spiral spring (Fig 37). In this the center is fastened to an arbor, the other end tached to the framework. The formulas are correct if the spring winds uniformly **center** remains the center of gravity. If the load is applied at B, the gr **stress** may occur at a point diametrically opposite A, or at the spindle if t

ature is great. For the first condition the external moment $= P \times 2R$ and $f = 2Pr \div$ $= 12\,PR \div bh^2$; in the second case, $f = 6\,PR \div bh^2$. The load is $P = fbh^2 \div 12\,R$ (first case), or $fbh^2 \div 6\,R$ (second case).

From the theory of the elastic curve, the angle of deflection $\beta = PRl \div EI$, and $\delta = \beta R = PR^2l \div EI = 12\,PR^3l \div Ebh^3$, and resilience $= 0.5\,PR^2l \div EI$, and $I = bh$ $\div 12$.

Round and flat helical springs (Fig 38). For a spring subjected to a force P, tending to twist the spring at a radius R, the load $PR = fZ = fnd^3 \div 32$ for the circular section, and $PR = fbh^2 \div 6$ for the square section; also $\delta = 64\,PR^3l \div Ed^4$ for circular section, and $\delta = 12\,PR^3l \div Ebh^3$ for square section; the resilience being determined as above.

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SECTION 42

ELECTRICAL ENGINEERING

BY

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PROFESSOR OF ELECTRICAL ENGINEERING, COLUMBIA UNIVERSITY

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1. Definitions, Units, and Standards

Potential (E or V), or DIFFERENCE IN POTENTIAL between two points, is oretically the work required to move a unit quantity of electricity from one nt to the other. It is analogous to difference in pressure, or head, of a current water The practical unit is the (International) VOLT, or electrical pressure ich, when steadily applied to a conductor the resistance of which is 1 ohm, l produce a current of 1 ampere. Difference of potential is measured in prac- by a VOLTMETER (Art 4). For comparison and calibration the Weston rmal Cell is used, the electromotive force (e m f) of which is 1.0183 volt at C. These cells are calibrated by Bureau of Standards, Washington. A livolt is 0.001 volt.

Current (I) expresses the quantity of electricity flowing past a given point 1 sec; it is analogous to gal per sec of a current of water, or to the miner's h. The practical unit is the (International) AMPERE, or the unvarying tric current which, when passed through a solution of $AgNO_3$ in water, in ordance with standardized specifications, deposits silver at the rate of 0.001118 per sec. Current is measured by an AMMETER (Art 4). A milliampere is 01 ampere. DIRECT CURRENT (d c) is officially one which always has the ne direction, though it may pulsate in value; but the term is po

applied to a unidirectional current of constant value, officially called a CONTINUOUS CURRENT (c c). ALTERNATING CURRENT (a c) is one which changes its direction and amount in regularly recurring periods.

Resistance (R) in an electric circuit is similar to friction in mechanics. It is the property by which a circuit limits the amount of current caused by a steady voltage. The (International) OHM is the practical unit of resistance and is the resistance offered to an unvarying electric current by a column of mercury, at temp of melting ice, which is 14.4521 gm in mass, of constant cross sec and a length of 106.3 cm. Resistance is usually determined by computing the ratio between volts and amperes in a circuit according to Ohm's law (Art 2).

Power unit is the watt or kilowatt (kw). A watt equals 10 million units of power in the centimeter-gram-second system, and in practice equals the product of 1 volt by 1 ampere, or is the rate of expenditure of energy by 1 ampere in a circuit of 1 ohm. A KILOWATT is 1 000 watts; a horsepower (U S) = 746 watts.

Energy. The theoretical unit is the JOULE, which is 10 million ergs; in practice, equal to 1 watt times 1 sec. Commercial unit is the kilowatt-hour (kw hr) equal to 3 600 000 joules. A circuit taking 10 amperes at 100 volts receives 1 kw of power, or 1 kw-hr of energy per hr; hence 1 kw-hr = 2 655 000 ft lb.

Coulombe is the unit of quantity of electricity (corresponding to a gallon of water) or the quantity transferred in 1 sec by a current of 1 ampere.

Capacity is the ability of an electric circuit to store energy in the space between conductors, measured in coulombes or quantity of electricity. A condenser is a device to provide capacity. FARAD (C) is the unit of capacity; or the capacity of a condenser which will be charged to a potential of 1 volt by 1 coulombe of electricity. A microfarad is one-millionth of a farad.

Maxwell (ϕ) is the unit quantity of magnetic flux, and is equal to 1 line of force.

Gauss (B) is the unit of magnetic flux density, and is equal to 1 line per sq cm, or 1 maxwell per sq cm.

Gilbert (F) is the unit of magnetomotive force (m m f), and is that m m f which will produce 1 line or 1 maxwell in a path in air having a cross-sec of 1 sq cm and a length 1 cm. One ampere turn = 1.257 gilberts.

Oersted (S) is the unit of magnetic reluctance, and is the reluctance of that path which requires 1 gilbert to set up a flux of 1 maxwell. Reluctance in a magnetic circuit is similar to resistance in an electric circuit.

Inductance is a characteristic of an electric circuit by which the circuit opposes any change in value of the current. It is similar to mass and inertia in mechanics. A variation in current produces an ELECTROMOTIVE FORCE (e m f) which opposes the change in current (Art 2).

Henry (L), the unit of inductance, is the inductance of a circuit in which a variation of current of 1 ampere in 1 sec will induce an e m f of 1 volt.

Hysteresis. When iron or steel is magnetized by an electric current, the value of magnetic flux does not vary directly with the current. An increasing current does not produce the same flux for a given current as a decreasing current. This lagging of the flux behind the magnetizing force is called hysteresis, and may be considered as the effect of friction between the molecules of iron which, in aiming themselves with the flux like small magnets, rub against each other. Hysteresis causes loss of energy and heating of the iron when the direction or magnitude of flux is changed. If the magnetizing force varies through complete cycles, positive and negative, the power lost in hysteresis is:

$$\text{Watts} = k_1 f V (B_m \div 1\,000)^{1.6} \times 10^{-6}$$

where $k_1 = 6$ to 18, depending upon quality of iron; f = frequency in cycles per sec, V = volume, cu in; B_m = maximum magnetic density in lines per sq in.

Eddy currents are local currents induced in any conducting material located within the influence of any varying magnetic field. The term is especially applied to currents

ıp in the iron core of a machine, or in a conductor carrying an a c. These currents e heating and loss of power and energy.

ffective resistance of a circuit carrying an a c is found by dividing the total loss in s due to the combination of true resistance, eddy currents and hysteresis, by the ıre of the effective value of the current.

eries. Two circuits are in series when the same current passes through ı successively.

hunt. A circuit is in shunt, in MULTIPLE or in PARALLEL, with another n they form a divided circuit, so that a part of the total current passes ugh each, while the potential across them is common.

onstant potential circuit is one across which the potential is kept as nearly ossible constant, irrespective of the current flowing. It infers a supply ı a circuit of low resistance. Most house lighting (incandescent), motor ly, and transmission systems are of constant potential type.

onstant current. Certain systems of lighting by arc lamps require con- t value of current in all the lamps, which are therefore connected in series the voltage across the whole circuit is varied to suit the number of lamps peration. A special type of generator or transformer is required. In pe the Thury system of long-distance transmission employs a constant d c, voltage generated being made to vary with the load.

nductance (G) of a circuit is that quantity which, if multiplied by the voltage, will total current if voltage is unvarying, and which will give the component of current ase with voltage ($I \cos \phi$) if voltage is alternating. For d c, conductance is the recip- of resistance. For a-c conductance, see Art 7. Conductance is used in analyzing mul- circuits. 1 volt applied to a circuit of 1 MHO conductance will cause 1 ampere to flow.

.ting of an electrical machine is the load in kw which it will give for a fied period of time without injury to itself. There are several limitations e output, but the one most frequently met and inferred is injury to insula- by the heat developed. The limit is the temp of the hottest part of the ine, and the allowable temp depends upon character of the INSULATING RIALS. These are divided into 3 classes: (*a*) cotton, silk, and other fibrous ials, not so treated as to increase their heat-resisting qualities; (*b*) simi- aterials, but treated and impregnated, including enameled wire; (*c*) mica, tos, and other materials capable of resisting high temp. Temp of machines ially determined by thermometer or by resistance measurements. Since r method gives the temp of the hottest spot, it has been agreed to assume mp of the hottest part as 15° C higher than indicated by a thermometer e surface, and 10° C hotter than indicated by resistance measurement. of surrounding air (ambient temp) influences directly the temp of the ine. Provision must be made that, when the air is at its highest possible (assumed at 40° C), the machine will not become excessively hot. This c by specifying that the rise in temp above surrounding air shall be such if operating in a room of 40° C, its hottest spot shall not exceed certain

ble 1. Temperature Conventions (Standard Rules of Am Inst Elec Engs)

ss of ation	Highest permissible temp	Limiting temp by thermometer	Limiting temp rise	Limiting temp by resistance	Limiting rise by resistance
a	95° C	80° C	40° C	85° C	45° C
b	105	90	50	95	55
c	125	110	70	115	75

established values. CONTINUOUS RATING is the load in kw which a machine will carry continuously without exceeding the above temp rise. INTERMITTENT RATING, or short time rating, is the load in kw a machine will carry for a specified period (up to 2 hr) without exceeding these limitations. Unless otherwise specified continuous rating is understood (Art 6).

2. Principles

Electric circuit. OHM'S LAW: $E = IR$, where I = current, amperes; E = e m f or difference of potential, volts; R = resistance, ohms. Used for solution of all simple d-c circuits and for a-c circuits having no reactance. For an a-c circuit having reactance, $E = IZ$, where I and E are effective values and Z is impedance in ohms (Art 7).

Kirchoff's laws are for solving any combination of series, multiple, or series-multiple circuits. FIRST LAW, for series d-c circuits: $E_0 = E_1 + E_2 + E_3$, where E_0 = generated e m f, E_1 = counter e m f (motor), and $E_2 = IR_2$, etc, are the drops in voltage in different resistances (Fig 1). Special case $R_0 = R_1 + R_2 + R$. SECOND LAW (for multiple d-c circuits): the algebraic sum of all currents meeting at a point = 0 (Fig 2). $I_0 = I_1 + I_2 + I_3$, where I_1 may be found by $I_1 = E \div R_1$, whence

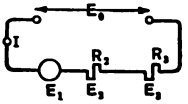


Fig 1. Series Circuit

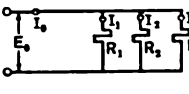


Fig 2. Multiple Circuit

$$I_0 = \frac{E}{R_1} + \frac{E}{R_2} + \frac{E}{R_3}; \quad \text{or in general} \quad \frac{1}{R_0} = \frac{1}{R_1} + \frac{1}{R_2} + \frac{1}{R_3}$$

Magnetic circuit is calculated similarly to electric circuit: $\phi = \frac{\text{m m f}}{\text{reluctance}}$ where ϕ = flux in maxwells, m m f is in gilberts and reluctance S is in oersteds $= l \div \mu A$, where l = length of path, cm; A = cross-sec of path, sq cm; μ = permeability of the material, varying from 1 for air to 3 000 in some steel

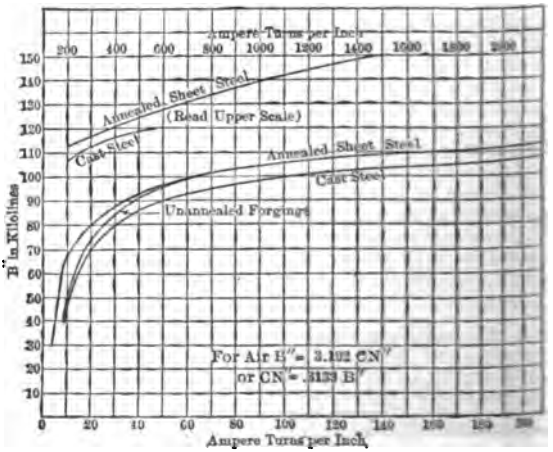


Fig 3. Typical Magnetization Curves

Permeability. Since μ varies greatly with different materials, and with different densities in same material, it is more practical to calculate a magnetic circuit by use

gnetization curves, which show relation between m m f per unit length of path and ensity in maxwells or lines per sq cm. Such a curve must be made for each different ial used. In U S it is customary to plot the magnetization curve between the re turns per in length of path and maxwells or lines per sq in (Fig 3). To determine irrent to produce a given flux, divide number of lines of flux by cross-section of t in sq in, and find required magnetic density in lines per sq in. Referring to magtion curve (Fig 3) of the particular material to be used, the number of ampere per in of path is obtained for this particular density. Multiplying this number by ı of path (in) gives total ampere turns required, which may be divided into any er of amperes or any number of turns, the product of which will give the required er. For a path in air, or in any non-magnetic material, the ampere turns per 2.313 times the density in lines per sq in.

ce on a conductor. Experiment shows that if a conductor carrying a current having a length l is placed in the magnetic field of density B gausses, at right angles direction of lines of force, a mechanical force will be exerted on the conductor, at angles to the conductor and to the flux, the value of which will be $f = Bli$, where mes, if the other quantities are expressed in c-g-s units. Practical form of this on is $F = 0.89\ Bli\ 10^{-8}$, where F = lb, B = lines per sq in, l = in, I = amperes. elation is basis for calculation of torque of any motor. In this case l must be the of conductor in the active field only. The pull of any magnet is given by equation,

$$P = B^2A \div 72\,134\,000$$

P = lb, B = lines per sq in, A = cross-sec of core, sq in. This is the case when a tic elements are in contact; when separated, the average gap density must be used.

neration of an electromotive force (e m f). When a conductor is moved nagnetic field so that it cuts lines of the field, an e m f is generated, nually proportional to rate of cutting the flux. When conductor cuts 10^8 lines ec, one volt is generated. If Z conductors are connected in series, as is in practice, $E = Z\phi \div 10^8\ T$, where E = average volts generated and ϕ = ut in T sec. In the armature of any machine the relations are:

Average E of a d-c machine = $(4\ PNS\phi) \div (120 \times 10^8)$

Effective E in an a-c machine = $(4.44\ PNS\phi) \div (120 \times 10^8)$

P = number of poles, N = rev per min, S = turns in series between term- ϕ = flux per pole, maxwells.

3. Conductors

pper is generally used, iron and aluminum being next in importance. gh all metals are conductors, only these 3 are of commercial value, since cost is reasonable and loss of energy is low. SILVER is the best conductor, too expensive for practical use. COPPER is nearly as good as silver, and t economically. ALUMI- next; in the form of solid is as cheap as copper, but uniformity in mechanical th. It is better when in of stranded cables, but n more expensive than r of equal conductivity. and STEEL approach cop- economic value, and for conductors are preferable se of their mechanical strength. An INSULATOR is a material used to prelow of current, but division between insulators and conductors is not sharp; ulators conduct some current. Common insulators are air, glass, porcelain r, cotton, paper, and dry wood.

Table 2. Relative Characteristics of Conductors of Equal Resistance and Length

(Costs are for Dec, 1915)	Cross-sec for equal res	Wt for equal res	Approx cost per lb, cts
Copper.....	1	1	20
Aluminum....	1.6	0.48	58
Iron, steel.....	8	7.1	2

Table 3. Resistances of Conductors

Material	Ohms resistance per mil-foot at 20° C, ρ	Temp coef. per cent increase pe deg C, from 2
Silver, annealed	9.5	0.377
Copper, annealed (100%)	10.3	0.388
" , hard drawn	10.5	.....
" , cast	12.0 to 97.0	.....
Gold, annealed	13.5	0.365
Aluminum, commercial	16.8	0.388
Zinc	38 0	0 379
Platinum	57.0	0 24
Iron, pure	61.5	0.56
" , cast	34.0 to 68.0	.....
Steel, soft *	104.0	0 388
" rails *	121.0	.....
" special *	84.0	.....
Nickel	84.0	0.56
Tin, pure	86.0	0.365
Lead	130.5	0 387
Mercury	570.0	0.072

* Steel being an alloy has varying spec resist. Steel used for track rails has appr ately the value given in table, while for the "third" or conductor rail a special co osition is used with lower spec resist.

Table 4. Brown & Sharpe Wire Table

B & S or A W G size	Bare diam, in	Double cotton covered diam, in	Area, circ mils	Wt, lb per 1 000 ft	Resistance		Curr capa ral be insul
					Ohms per 1 000 ft, 75° F	Ohms per mile, 75° F	
.....	1.152	.....	1 000 000	3 050	0.01051	.....	6
.....	1.035	.....	800 000	2 440	0.01313	.....	5
.....	0.891	.....	600 000	1 830	0.01751	.....	4
.....	0.819	.....	500 000	1 525	0.02101	.....	3
.....	0.728	.....	400 000	1 220	0.02627	.....	3
.....	0.590	.....	250 000	762	0.04203	.....	2
0000	0.460	.....	211 600	640	0.04906	0.2589	2
000	0.4096	.....	167 805	507	0.06186	0.3265	1
00	0.3648	.....	133 079	402	0.07801	0.4117	1
0	0.3248	.....	105 538	318.8	0.09838	0.5189	1
1	0.2893	0.303	83 694	252.8	0.12404	0.6546	1
2	0.2576	0.272	66 373	200.5	0.15640	0.8254	9
3	0.2294	0.243	52 634	159	0.19723	1.0409	7
4	0.2043	0.216	41 742	126.1	0.24869	1.3125	6
5	0.1819	0.194	33 102	100	0.31361	1 6550	5
6	0.1620	0.174	26 250	79.3	0.39546	2.0871	4
7	0.1442	0.156	20 816	62.9	0.49871	2.6318	3
8	0.1284	0.140	16 509	49.88	0.62881	3.3184	3
9	0.1144	0.126	13 094	39.56	0 79281	4.184	2
10	0.1018	0.112	10 381	31.37	1	5 277	2
11	0.0907	0.101	8 234	24.88	1.2607	6.654	2
12	0.0808	0.091	6 529	19.73	1.5898	8.390	1
13	0.0719	0.082	5 178	15.65	2.0047	10.580	1
14	0.0640	0.075	4 106	12.44	2.5278	13.341	1
15	0.0570	0.067	3 256	9.84	3.1150	16.822	
16	0.0508	0.059	2 582	7.81	4.0190	21.213	
17	0.0452	0.053	2 048	6.19	5 0683	26.749	
18	0.0403	0.048	1 624	4.19	6.3911	33 739	

esistance of any conductor is given by: $R = \rho L + A$, where R = resistance, s; L = length of conductor, ft; A = cross-sec of conductor, circular mils; pecific resistance, or resistance of a conductor of unit cross-sec and length. igh the cm and sq cm are the more scientific units, it is convenient to use ircular mil for unit of cross-sec and the foot for unit of length, as adopted for nercial wires. A circular mil is the area of a circle having a diam of one mil 1 in). A circle 1 in diam has an area of 1 000 000 circ mils, and 1 sq in = 000 circ mils. Hence, in above equation, if length be 1 ft and A = cross- circ mils, then ρ has values given in Table 3.

ecific resistance ρ varies directly with the temp. Table 3 gives a constant letermining the percentage of increased resistance per deg C in excess of . General law of increase in resistance of copper with rise in temp is $R_0 (1 + 0.0042\, t)$, where R_t = resistance at any temp, $t°$ C; R_0 = resistance C; t = temp above 0° C.

ire gages. Standard classification of wires in the U S is the Brown and pe or American wire gage (Table 4).

& S gage sizes have definite relations, readily remembered: A wire 3 sizes larger another has nearly half the resistance, twice the area, and twice the wt per unit h. Thus, No 7 has half the resistance of No 10. A wire 10 sizes larger has 0.1 the ance and 10 times the area and wt. No 10 is approx 0.10 in diam, has area of 10 000 nils, resistance of 1 ohm per 1 000 ft at 20° C, and weighs 32 lb per 1 000 ft.

sistance and wt of standard sizes of wire of other materials are found by following ratios: ALUMINUM, multiply resistance given for copper by 1.61 the wt by 0.301. IRON telegraph wire, resistance is 5.96 and wt 0.877 that pper. STEEL wire, resistance is 8.82 and wt 0.883 that of copper.

rmingham wire gage (B W G), also known as Stubbs, is largely used in t Britain and somewhat in this country. Metric system is used in France Germany. Roebling table is used in the U S for steel wires and cables.

Table 5. High-resistance Wires

	Composition	Resistance relative to copper	Temp coef, % increase per deg C	Max allowable temp, deg C
rman silver, 18%.....	Ni-Cu	19.0	0.0310	.....
" " 30%.....	"	28.0	0.0310	.....
a (Boker).....	"	27.3	0.0005	300
vance.....	"	28.6	0.0018	480
nganin *.....	Cu-Fe-Mn-Ni	24.0 to 43.0	0.0011	.....
erior.....	Ni-Steel	49.0	0.0720	.....
max	"	51.00	0.0540	540
hrome.....	.....	56.0	0.0430	1 200

* Manganin is used in electric meters because of its low temp coef.

4. Measuring Instruments

lenoid. Simplest form of measuring instrument is the PLUNGER TYPE of oid. It consists of a coil of thick wire carrying current to be measured, and t steel plunger which is pulled into axis of coil by effect of the current acting st weight of plunger, or against a spring. Operates with both d c and a c, is not accurate with either. MAGNETIC VANE is a special form of solenoid ument. Inside the coil carrying the current is a thin steel vane, pivoted nter of coil, and a fixed vane of triangular cross-section. When c

flows in the coil, both vanes are magnetized alike and repel each other; hence pivoted vane moves away from fixed vane, moving force being proportional to square of the current. Used both for d c and a c, but its accuracy varies with frequency and wave shape. INCLINED COIL solenoid instrument is very generally used to measure a c. A cylindrical coil of wire carrying current to be measured is set inclined to a spindle or axis carrying a steel vane. When no current flows, the vane is held in plane of coil by a spring. When current flows, the magnetic field is perpendicular to plane of coil and the pivoted vane in moving to set its longer axis in the line of flux, rotates the spindle to which pointer is attached. Force is proportional to square of the current. Used for d c and a c, but is affected by wave shape and frequency. D'ARSONVAL PRINCIPLE, used in almost all instruments for d c, employs a coil of fine wire placed in the field of a permanent magnet suspended so that it can rotate. Current to be measured, or a definite part of it, passes through the movable coil, causing a torque which is resisted by a spring. The force is directly proportional to the current. Only operative with d c, and is affected by stray magnetic fields.

Hot-wire instruments may be used to measure both d c and a c. The current is passed through a wire, which it heats and lengthens. Increase in length is taken up by a spring or weight, the movement of which is proportional to square of current.

Dynamometer consists of 2 coils, fixed and movable, so arranged that their magnetic fields act upon each other. The current, passed through the two in series, develops a torque (as in a series motor) proportional to square of the current. Movement of the coil is resisted by a spring and indicated by a pointer. As no iron is present, this instrument is accurate for d c and a c of all frequencies. The principle is used in the Siemens dynamometer, in some a-c voltmeters and ammeters and in all wattmeters. In wattmeters one coil is in series with the line and the other coil across the load, receiving current through a high resistance proportional to the potential.

Induction-type instruments utilize the fact that, if a coil carrying an a c has within it a piece of copper or aluminum, eddy currents will be induced in the metal, and these will react on the main current giving a force or torque. The torque is proportional to square of current. Instrument operates only on a c, and is affected by varying frequency.

Ammeter operates on any one of above-mentioned principles. Its electric circuit must have very low resistance, and usually the measuring coil carries only a small though definite part of the current. In parallel with working circuit, there is usually a metallic "shunt," which has such resistance that the maximum current to be measured causes a drop of about 50 millivolts (Art 1, par 1) in the shunt. Some meters have interchangeable external shunts, while others have shunts placed inside the case and permanently connected in the circuit.

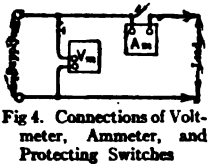


Fig 4. Connections of Voltmeter, Ammeter, and Protecting Switches

When the instrument is used with an external shunt it is sometimes called a MILLIVOLTMETER, its scale being marked in millivolts. An ammeter must never be left connected in the circuit without its shunt. Most ammeters for measuring d c operate on the d'Arsonval principle, as it gives the most accurate results. For a c the inclined coil is in most general use. Most a-c meters will indicate as correctly on a c as on a c, but not as accurately as an instrument of the d'Arsonval type (Fig 4).

Voltmeter resembles an ammeter and in reality measures current, but its electrical circuit has a high resistance connected in series with it, to limit the current to a reasonable value. Since, in accordance with Ohm's law, with a

stant resistance the current is proportional to the potential across the cir-, the scale deflection will indicate volts. The resistance must be non-in-tive, and must not vary with temperature. When the resistance coil is rate from voltmeter the coil is called an multiplier.

e dynamometer type voltmeter is to be preferred for a c, and a potential transformer be used instead of a multiplier if required to measure a voltage higher than the range e voltmeter proper. ELECTROSTATIC VOLTMETER, a special instrument for measur-ery high voltages, utilizes the force exerted between two charged bodies. A moving is connected to one side of the circuit, a fixed vane to the other. As the high poten-charges these vanes, a force is exerted between them causing the movable vane to e. This instrument is used for potentials greater than 10 000 volts, either a c or d c.

Wattmeter is used on both a-c and d-c circuits for measuring the true or active er in watts; *EI* in d-c circuits, *EI cos* ϕ in a-c circuits. In d-c circuits a meter and an ammeter may be used to measure power, the product of the s and amperes giving true power, but in an a-c circuit this is not true. The tmeter has a potential coil connected in parallel with the load (like a volt-er), and a current coil connected in series with the load (like an ammeter). e needle shows negative deflection, the terminals of one coil or other must be rsed. Range of the wattmeter must be selected with reference to the volt current capacity of its coils. A "multiplier" may be used in the potential and a current transformer in the circuit of the current coil.

Watt-hour meter measures energy by recording watt-seconds. Coils are arranged to a motor, which causes a disk to revolve at a velocity proportional to the power. A ter registers number of revolutions of the disk, and this number represents energy. t-hour meters for a-c circuits are constructed differently from those for d-c. (Art 11.)

Power-factor meter determines the phase angle (ϕ) between voltage and ent of an a-c circuit. Since in practice the power factor is the more impor-t, the scale is graduated in *cos* ϕ. Instrument is constructed like a wattmeter ing 2 coils, one for potential and one for current.

Frequency meter indicates the frequency in electrical cycles per second of the circuit age. The two common types are the vibrating reed and the induction. They are nected to the line-like voltmeters.

Recording meters are often used in power stations to provide a continuous record of age, current or power. A pen, attached to the indicating element, rests on a strip isk of paper, and draws a line as the paper moves by clock work.

5. Direct-current Generators

Application. Direct-current (recently officially named continuous current) aratus is used for short transmission distances. For long distances the a-c em (Art 7) is preferable, because the transformer (Art 13) permits trans-sion at very high voltages. However, since the d-c motor is more convenient more easily controlled than the a-c motor, the d-c system is used whenever sible. It is often found advisable to generate and transmit by means of a c, ch is then converted to d c at the place where the motors are located. Thus, it generating stations produce a c, though many of the motors supplied by e stations use d c provided by synchronous converters.

Fundamental principle of a dynamo is the production of e m f (Art 2) in one or more luctors by the motion of these conductors in a magnetic field. In most machines the f is alternating; first positive, as conductor passes a north pole, then negative, as it es a south pole. For use as a d-c machine a commutator must be provided to rectify age and current. The commutator is the distinguishing feature between d-c and machines. Small machines are usually of the bipolar type, with a more or less en-ed frame of cylindrical shape. Large machines are multipolar; that is, hav-e number of inwardly projecting radial pole pieces. Commutating pole or INTF

machines have small auxiliary poles placed alternately with respect to the main poles and excited by a few turns in series with the load. Effect of these poles is to improve commutation of the machine. Machines are divided generally into belt-driven and direct-connected, the former having the higher speed.

Shunt machine, either generator or motor, is one in which the entire excitation is derived from a circuit of many turns and high resistance, connected in shunt or multiple with the armature circuit (Art 6, Fig 8). The characteristic of a shunt generator is poor regulation; that is, voltage decreases as load increases, and hand regulation of a rheostat in the field is necessary to provide steady voltage.

Compound-wound machine has on each field pole, in addition to its shunt winding, a few turns of thick wire which carry the load current and are known as series winding (Fig 5, 6). This causes excitation to increase as load increases, and tends to keep the terminal voltage constant or even to increase it. If the field windings are proportioned to cause a higher voltage at full load than at no load the machine is over-compounded. A flat-compounded machine has the same voltage at full load as at no load, and a compounded machine has a lower voltage at full than at load. A compound-wound machine may be connected either in short shunt (Fig 5), which is usual practice, or in long shunt (Fig 6), in which the field winding is connected from outside of the series field to outside of the armature.

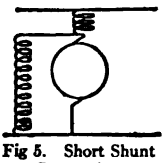


Fig 5. Short Shunt Connections

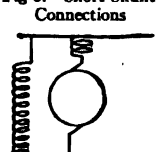


Fig 6. Long Shunt Connections

Constant potential vs constant current. In constant-potential type an endeavor is made to maintain a constant potential across the load. For particular purposes machines sometimes have the field connected in series with the armature, and with a special regulating device for giving constant current irrespective of load. These machines were formerly much used in arc lighting, but are not now built.

Installation. In installing and erecting a d-c machine, certain features must receive careful attention, so that the machine shall operate properly and not deteriorate too rapidly. The most important considerations are: (*a*) For large machines the base is bolted to the foundation in accordance with the drawings. (*b*) Bearings are lined up, cleaned, and oiled. (*c*) Field coils are tested for open circuit and wrong connections. Test is made with a compass for correct polarity. For a self-excited generator there is one particular connection of the field to the armature for each direction of rotation. (*d*) Armature must be properly centered so that the air gap is correct at all points. Gap is measured by taper wedges. (*e*) Magnet frame is bolted to base. (*f*) Commutator is smooth and polished. Use sand paper, never emery paper, to polish commutator. Brushes are properly and accurately spaced around the commutator, sandpapered and fitted to its curvature, and their pressure adjusted to the correct value, usually 1.5 to 2 lb per sq in of contact surface. (*h*) Field connections are adjusted for correct direction of rotation and substantial connections made. (*i*) Machine must be protected from moisture during shipment; if it be damp it must be dried out by heat before being subjected to a voltage.

Operation. In starting a single generator it may be necessary to CREATE THE FIELD by exciting the shunt field separately for a moment to set up residual magnetism. To cause the machine to PICK UP, or generate voltage by self-excitation, the rheostat connected in series with the shunt field must be cut out or short-circuited. If total resistance of shunt-field circuit exceeds a certain critical value the machine will not "pick up," however much time is allowed

ırallel operation. For economical operation of a power station there must number of machines, whose aggregate capacity equals the maximum de- d on the station. As the demand varies the number of machines in oper- ı is adjusted so that those running are operating at a load near their rating, therefore at good efficiency. To operate SHUNT GENERATORS in parallel t is, feeding the same bus-bars), it is only necessary to adjust all to the : polarity and voltage, connect them to the bus-bars, and adjust division ad by strengthening the field of the underloaded machine if voltage of the bars is low, or weakening the field of the overloaded machine if voltage is . To operate COMPOUND GENERATORS in parallel there must be an equalizer ıection, making a common connection on all the machines at the point 'een the armature and the series field (Fig 7). The equalizer divides the current in proper proportion between the series fields of the different gen- ›rs, preventing the machines from acting as series generators or differential ›rs, which would cause short-circuits. For successful operation of compound hines in multiple all those connected to set of bus-bars must have same amount ompounding, and same voltage at no . The compounding curves of machines t be investigated before they are oper- in parallel. With unlike curves, one hine may be overloaded while another ıderloaded, unless the field current of of them is adjusted.

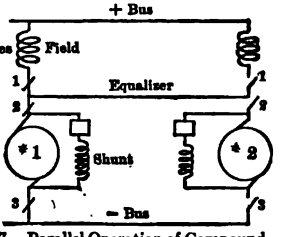


Fig 7. Parallel Operation of Compound-wound Generators

a machine is to be connected in parallel ı those already in operation it must have ıer polarity and voltage, the equalizer ıit must be made, and the switches are :d in the order 1, 2, 3, as in Fig 7. With any other order the effect is the : as having no equalizer. In shutting down one machine, switch No 3 must pened first; then No 2 and No 1.

[anagement. See Art 6.

esting. D-c machines are tested for regulation, commutation, and heating, ıubjecting them to a load, but their efficiency is best determined by the rate loss method, for which the machine need not be run under load. Cus- ary commercial tests are:

) **Resistance measurements** are made on armature winding, shunt field, and series , at known temp, before the machine is run and again after the heat run (*d*). Re- nce of a lap or multiple winding is found by measuring between two diametrically site points on commutator. If this resistance be R', then the actual armature re- nce is $R_a = 4\,R' \div p^2$, where p = number of poles on machine. Resistance of a series- ıd armature is found by measuring between two commutator segments separated distance equal to periphery of commutator divided by number of poles. Resistance ıe brush contact can not be accurately measured by any simple means, as its value is ted by value of the current, speed of commutator and pressure of brushes. An pted method is to assume a total loss of 2 volts in all brushes.

) **Core loss and friction (stray power).** Machine is run as motor without load, current being held at proper value and the voltage applied to armature adjusted to desired speed. Input to armature is measured; subtracting from this the calculated of armature circuit, the net value of input = core loss + friction.

) **Load runs.** Machine is run and the excitation adjusted to give proper voltage at ıad. Load is then added and terminal voltage noted. If E_0 = voltage at no load E = voltage at full load, the regulation is $(E_0 - E) \div E$. Brushes must be set in ıer position for commutation, which is judged by observing action of brushes at no l, full load, and 150% load.

(*d*) **Heat runs** are made by operating at full load until the temps of the various p that can be noted during operation become constant. The greater the machine's capac the longer is the testing time required. Heat runs are sometimes made at 1.25 or times rated load for 2 hr. Heat runs may be made either by the DEAD-LOAD METH which a resistance load such as a water rheostat is connected to the terminals and rated load power is required to drive the machine; or by the Hopkinson or PUMPING METHOD, in which 2 machines of similar characteristics are run in multiple, one acting a generator to supply electrical power to the other as motor, which in turn drives the by a belt or equivalent mechanical connection. For this test a LOSS SUPPLY is requi which may consist of a source of either electrical or mechanical power. The power quired is from 10 to 20% of the rating of one machine. During a heat run, thermo readings are taken at stated periods, showing temp of frame, field-coils, bearings, and surrounding air. After the run thermometers are placed on various parts of the ma the bulbs being protected from radiation by small cotton pads, and the temps are noted armature core surface, ventilating ducts and winding, commutator surface, pole-t field-coils, bearings, frame, and room. Resistance of the electrical circuits sho measured, and average temp of the copper calculated from: $t_1 = (R_1 \div R_0)(239+t_0) - 2$ where R_1 and R_0 are the hot and cold resistances, and t_1 and t_0 are the hot and cold temp

(*e*) **Compounding test.** To adjust the current in series field so that a compound wound generator will give specified voltages at no load and full load, the machine is operated at no load and current in shunt field adjusted to give desired no-load voltage. Load is then put on, and it will usually be found that the terminal voltage is too great. Strips of German silver or other resistance metal are then connected across the series field terminals, until by shunting current from the series field the machine voltage is reduced to desired value. This "series-field shunt" is then insulated, and made up permanent form. Before making the no-load adjustment, it is advisable to overexcite the shunt field for a moment to overcome hysteresis.

(*f*) **Insulation tests.** After the heat run the insulation is tested, first by measuring its resistance and then by applying a high-potential test. INSULATION RESISTANCE is found by connecting one terminal of a 500-volt d-c circuit to the windings through a 500 voltmeter, and connecting the other terminal of the 500-volt circuit to the machine frame. If voltmeter resistance is R_v and the deflection during above connection is x, then insulation resistance in ohms is: $R = R_v(500 - x) \div x$. The value of R should be approximately one megohm (1 000 000 ohms) for each 1 000 volts of rated potential of the machine. HIGH-POTENTIAL test is made by applying an alternating high potential between each winding and the frame of the machine for 1 min, in accordance with specifications adopted by Am Inst Elec Engrs. This test is usually made only at the factory and should be under supervision of an experienced person.

6. Direct-current Motors and Motor-generator Sets

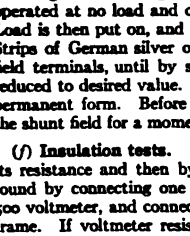


Fig 8. Shunt-Motor Connection

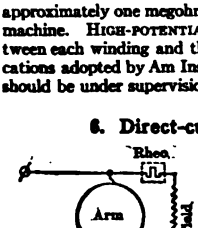


Fig 9. Series-Motor Connection

Shunt motor (Fig 8) has only one exciting winding which is connected across armature terminals, and in parallel or shunt with the armature. Field winding consists of a large number of turns of fine wire on each pole, and all poles are usually connected in series in the circuit. Field current depends upon line voltage and resistance of field winding. Resistance of field winding is made high, so that the field current will be between 1% (large motors) and 5% (small motors) of the full-load current of motor. The characteristic of the shunt motor is a fairly constant speed for all reasonable load values.

Series motor (Fig 9) has only one exciting winding which is connected in series with the armature, and so the current flows through both field and armature. Field winding consists of a few turns of thick wire on each pole, the windings on all poles being connected in series. Field current depends upon the load; larger

heavy load and small with light. Resistance of field winding is made low, iat loss of voltage and power in that circuit will be small. Characteristics series motor are: a speed varying with every change in load, high speed at load, and low speed at heavy load. Efficiency is high throughout a wide e of speed. Speed will be dangerously high at no load; thus a series motor t always be connected rigidly to its load. Since torque is high at low speeds, motor is especially adapted to work requiring frequent starting.

ompound motor. Usual form is the cumulative motor, each pole of h has both series and shunt winding, wound and connected for mutual tance in producing the magnetic field. It is a combination of shunt and s motor, designed to give the good starting qualities of the series motor and void danger of excessive speed at light loads. DIFFERENTIAL MOTOR has osed shunt and series winding; not much used and not recommended.

iclosed vs open type referring to the mechanical housing of the motor. N TYPE has all parts freely exposed to the air and is therefore well ventilated. nded to be used indoors or in protected places. INCLOSED TYPE is for use xposed locations, where there is liability of dampness and dirt. Special ns must be provided to circulate air inside the machine, but even then an osed motor is larger and more expensive than an open motor of same capacity. itive capacities in output of open, semi-inclosed and totally inclosed motors, shown by Table 6. Inclosed motor generally weighs about 15% more than pen motor of same capacity, notwithstanding that it is allowed by comcial convention to operate at 15° C higher temp.

Table 6. Data of Typical Commercial Motors

Type	Output, h p	Temp rise, deg C	Weight, lb	Speed, r p m
en	10.0	40	970	700
mi-inclosed	8.0	40	970	700
tally inclosed	5.75	55	970	775

ating. Motors are rated on basis of their continuous or their intermittent icity. CONTINUOUS RATING is the output in h p (or kw) which a motor will continuously with a maximum temp rise (measured by thermometer above ounding air) not exceeding 40, 50, or 70° C in field or armature, depending n character of insulating material (Art 1). INTERMITTENT RATING is the ut which the motor will give for 1 hr (starting at room temp), with a maxm rise in temp above surrounding air not exceeding the above values (Standzation Rules of Am Inst Elec Engs, 1914, Art 1).

oltage and current. Usual values of voltage for d-c motors are: 110–125 small units on lighting circuits; 220–250 for motors in factories, shops, and es, on power mains or on outside mains of a 3-wire system; 500–600 for ral railway work; 1 200–2 400 for heavy railway work. Current in amperes ired for any motor is: I = (output in h p × 746) ÷ (effic × voltage). Usual iencies of motors of different sizes are given in Table 7.

pplications of motors. Chief applications of d-c motors: SHUNT MOTOR, ing shafting, machine tools, blowers, reciprocating pumps, motor-generators; IES MOTOR, R R, and all transportation work, hoists, cranes; COMPOUND OR, elevators, hoists, and machinery that is stopped and started frequently.

peed characteristics. Shunt motor is always used when constant speed is

may have same speed at full load as at no load, but in this case the mot quires special attention in starting and is subject to bad commutation at loads. At full load the speed of any well-designed shunt motor may be incre to equal the no-load speed, by putting resistance into the field circuit. Vari speed motors are of the series type; they decrease in speed as load incre every change in load causing change in speed. Multispeed motors are of type, and usually have commutating poles so that they may operate with dif strengths of field without suffering from bad commutation. They may be trolled by varying the field resistance, or the voltage impressed upon the arma By increasing resistance in series with the field of a shunt motor speed is incr due to weakening of the field. If the motor has commutating poles, to assure commutation, speed may be varied in ratio of 1 to 2, or even 1 to 3 in small Motor speed is stable with this mode of control and efficiency is good.

Potential control. By means of several generators and several wires various dif voltages are available. Thus, with one main generator giving 240 volts and 3 sm ones giving 40, 80, and 120 volts, combinations may be made giving 40, 80, 120, 160, and 240 volts. By connecting the armature terminals to these voltages, the motor run at speeds proportional to the voltages. The shunt field is left continuously connec to one circuit, usually the maximum voltage circuit. A shunt motor with normal ex tion will be stable at each speed; that is, will operate at fairly constant speed irrespec of load. Efficiency will be good at all speeds. A number of wires are needed to make different voltages available at the places where the motors are located.

Adjustable-speed motors. By connecting a rheostat in series with a shunt armature, the voltage impressed upon the armature will be reduced by an amount portional to the current, and speed will thereby be reduced. The speed is unst changing with every change of load, and efficiency is poor.

Starting box (rheostat) is always employed in starting d-c motors, to red the voltage impressed on the armature when not running at a speed high en to generate the proper counter e m f. Fig shows usual connections of a starting bo the line and motor.

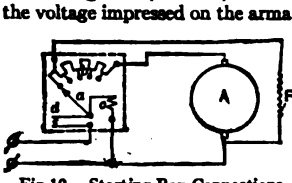


Fig 10. Starting Box Connections for Shunt Motor

Starting box contains: (*a*) means of opera closing the circuit supplying current to the m including the field current; (*b*) set of resis steps in series with motor armature, and a of short-circuiting this resistance step by (*c*) a magnet coil connected across the mot minals to open the circuit if impressed fails (low-voltage release); (*d*) a magnet coil carrying main current to actuate a and open main circuit, if current exceeds a specified value (overload release).

Specifications. In calling for bids for d-c motors it is customary to spe kind of service and load, method of drive, voltage, horse-power, speed whether open, semi-inclosed or inclosed, series, shunt, or compound-wound shunt-wound, whether shunt field rheostat is to be supplied; if compound-wo whether cumulative or differential; dimensions of pulley and shaft extens whether rails or base are required; whether starting box is to be supplied if so, the style; temp rise for continuous full load; temp rise for 25% over efficiency at 25, 50, 75, 100, 125, and 150% load; starting torque requir necessity for withstanding moisture; regulation; percentage of variation speed between no load and full load, with field resistance constant; variation speed at full load obtainable by variation of field rheostat.

Installation. (See Art 5.)

Operation. All motors should be frequently inspected and following point noted: (*a*) Bearings filled with proper amount of oil; (*b*) brushes securely held

roper position; (*c*) brushes fit properly; (*d*) commutator smooth: look high mica," or projection of the insulation above the bars; (*e*) air gap (*f*) commutator not worn in grooves.

oubles. Following troubles that may occur in operating d-c motors, and causes, are stated by Crocker and Wheeler: (*a*) Sparking at commu-. Causes: armature carries too much load, brushes improperly spaced ot in proper position, rough commutator, poor brush contact, internal short pen circuit, field too weak, unequal strength of poles, vibration. (*b*) Heat- of commutator and brushes. Causes: sparking, bearing trouble, bad con- ions, brush friction too great. (*c*) Heating of armature. Causes: overload, rnal short circuit, moisture or ground, reversed coil. (*d*) Heating of Cause: internal short circuit. (*e*) Heating of bearings. Causes: ings dry or dirty, shaft out of true, bearings out of line, thrust due to belt, lanced magnetic pull. (*f*) Noise. Causes: armature not balanced, hes dry or not set at proper angle, armature strikes. (*g*) Speed too low. ses: wrong voltage, overload, armature strikes, bearing too tight. (*h*) Speed high. Causes: wrong voltage, field too weak. (*i*) Motor stops, or fails art. Causes: overload, open circuit, wrong connection.

sts. A motor is usually tested for resistance of insulation and of windings, heat run, power, regulation and commutation. Regulation test consists in measuring speed at ad and at full load. Other tests are the same as made on d-c generators (Art 5).

nstants. Efficiency of any particular size of electrical machine may vary widely rding to its design. For weight and cost, see Art 19.

Table 7. Average or Usual Constants for D-C Machines

w rating	1	5	10	20	50	100	200	500
ficiency at full load	0.80	0.84	0.86	0.88	0.90	0.91	0.92	0.93
v per min	2 000	1 100	850	680	650	600	450	350

Motor-generator set is a combination of motor and generator having sepa- fields and armatures, but mounted on same shaft with common base and rings. The two machines may both be for d c, both for a c, or one for each, ccordance with the use to which the plant is to be put.

rincipal applications are:

alancers. Two similar d-c machines mounted on the shaft, and connected in series ss the mains of a 250-volt circuit. From the common connection a third wire is ght out, thus giving a 3-wire system for 125-volt lamps on each leg, and for 250-volt ps on the outside main. With compound machines the regulation on the 125-volt uits is good.

oosters. To raise the voltage on a particular feeder a series booster is used, consist- of a shunt motor driving a series generator, the latter being connected in series with ply and feeder. The set runs at constant speed, and the voltage increases with the ent to the load. Voltage at full load is from 10 to 20% of the rated voltage of the uit. The combination may be designed to neutralize loss in voltage due to drop in feeder.

Multiple-voltage systems. Motor-generator sets are used to give the fractional ages required for these systems. Each set consists of 2 or 3 machines operating as a ancer; that is, each may be a motor at one moment and a generator at the next, depend- upon which circuit is supplying the load.

Ward-Leonard system. For regulating the speed of a motor a special gen- tor may be provided, which, being driven by its own motor, forms a motor- erator set. By varying the field strength of the generator any voltage may generated, from 0 to full rated voltage. If this generator supplies only the

armature of the motor the speed of which is to be regulated, its speed may controlled from o to full speed by merely varying the small current flowing the field of the special generator. Main supply circuit furnishes current the motor of the set, field of the generator, and field of the controlled motor.

Ilgner sets. Where a motor must start and accelerate a heavy load frequently, as in mine hoists and rolling mills, the motor is advantageously supplied from a special set, consisting of a motor (shunt or induction) driving a special generator, with a heavy flywheel forming part of the set. The set motor may have poor speed regulation. As the accelerating motor starts, it draws a heavy current from the generator, causing the speed of the set to drop and the flywheel to give up a considerable amount of stored energy. This energy supplies the peak demands, and when the peak load period has passed the set speeds up gradually and stores more energy in the flywheel. This combination is expensive, but is used where conditions limit the maximum demand which may be made on the electrical supply system. (See Sec 16, Art 9.)

7. Alternating-current Circuits

Sine wave. Every generator for d c as well as for a c has induced in its armature windings an alternating voltage. The commutator of the d-c machine rectifies this, giving direct or continuous voltage. Since an a-c machine has no commutator, this voltage is brought to the load in its alternating form by sliprings.

In designing machines every endeavor is made to obtain an e m f which follows a sine law: $e = E_m \sin \theta$, in which e is the instantaneous value, E_m the maximum value and θ gives the phase time, or scissa at which the instantaneous value is specified. All calculations of a-c circuits are based on assumption that voltage and current follow a sine law.

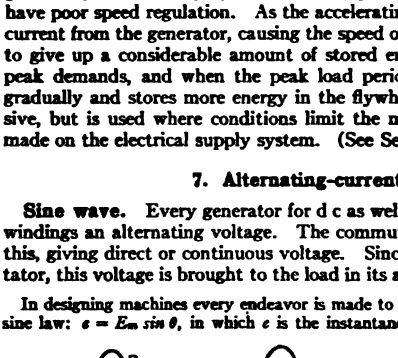


Fig 11. Alternating-current Voltage and Power, Rectangular Coordinates

Frequency. The series of values from a to c (Fig 11) constitutes a cycle or period. Time in seconds required for one cycle is the periodic time T, and $1 \div T$ (number of cycles per sec) is the frequency. In commercial practice frequencies of 25, 50, and 60 cycles per sec are used, the first for general power, the latter for lighting purposes. If θ is expressed in radians then $\theta = 2\pi ft$, and $\theta = 2\pi f \times t$. The term phase θ is a conventional means of expressing a given instant of time t. A cycle may be divided into 360°, and θ may be the abscissa expressed in degrees.

Effective, virtual, or root mean square value of an a c is equal to that value of a d c which would produce the same heating effect or average power loss in a resistance RI^2. If current follows a sine law, the effective value is equal to maximum value divided by $\sqrt{2}$. Effective value of e m f is that which, when multiplied by the effective value of current in a simple resistance, will give the true power in watts. It is also equal to maximum value divided by $\sqrt{2}$. Effective values of current and voltage are the values shown by all instruments, and are the only ones used in practical engineering. (See Sec 16, Art 9.)

Inductive reactance, inductive circuit. Any circuit in which the presence of a current produces a magnetic flux is an inductive circuit, the quantity of inductance depending upon the number of turns and the quantity of flux produced by one ampere. When an

ws in an inductive circuit the current lags behind the impressed e m f, and the relation en voltage and current in such a circuit is $E = 2\pi fLI$, where E = effective value of f = frequency in cycles per sec, L = inductance in henries, I = effective value of nt. $X_L = 2\pi fL$ is the inductive reactance in ohms. $E = X_L I$ in a circuit having in- nce only, and the current will lag behind the voltage by one-quarter of a period.

pacity reactance, anti-inductive circuit. If an alternating e m f is impressed upon uit having capacity only, the current which flows precedes or leads the e m f by 90°, ts value is $I = 2\pi fCE$, where C is the capacity in farads and the other quantities s in preceding paragraph. $X_c = 1 \div 2\pi fC$, called capacity reactance, is expressed ms and $E = IX_c$.

›wer and power factor. In an a-c circuit containing inductance or capac- as well as resistance, the product of effective volts and effective amperes does not give true power, because the current will be out of phase with the ge; that is, the maximum value of current will occur either later than or e the maximum value of voltage. Therefore EI is the volt-amperes or arent power." Ratio of true power P to apparent power is the power r. It may be proved that the power factor is equal to the cosine of the of phase difference between the voltage and current as shown by equation: r factor = $\cos\phi = P \div EI$, where ϕ is the time differ- in phase between voltage and current, expressed in lar measure.

11 shows an alternating voltage E and an a c I differing ase by an angle ϕ, which means that their maximum or their alues occur at a difference in time of $(\phi \div 360)$ T sec, where the periodic time of one cycle. Fig 12 is a vector repre- tion of these quantities, and conventionally I is said to lag d E by an angle ϕ. $E\cos\phi$ is the component of the voltage ase with the current, and is called the "active component." useful or heat power in watts is $P = EI\cos\phi$. $E\sin\phi$ is omponent of the voltage out of phase with the current, and led the "reactive component." $Q = EI\sin\phi$ is the reactive power, as it reacts on the ssed forces without producing useful work or heat. Total volt-amperes or apparent r is $EI = \sqrt{P^2 + Q^2}$.

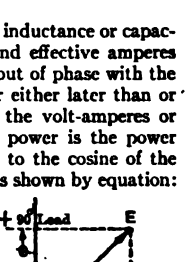
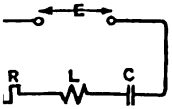


Fig 12. Vector Diagram of Alternating-current Voltage

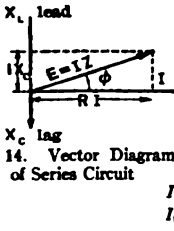
lculations. A corollary of the law of conservation of energy is that the sum of all rue (heat) power consumed in a circuit is equal to the input of true power, $P_0 = P_1 + P_2 + P_3$. The sum of all the reactive power consumed in a circuit is equal to the input of reactive power, $Q_0 = Q_1 + Q_2 +$ etc, and $E_0 I_0 = \sqrt{P_0^2 + Q_0^2}$.



3. Series A-c Circuit, h Resistance Induc- ce and Capacity

Kirchoff's laws for a-c circuits. FIRST LAW (for series circuits) (Fig 13 and 14): The sum of all active components of voltage ($E\cos\phi = IR$) in a closed circuit is equal to zero, and the sum of all reactive components ($E\sin\phi = IX$) is equal to zero. Hence,

$$E_0\cos\phi_0 = E_1\cos\phi_1 + E_2\cos\phi_2 + \text{etc}$$
$$E_0\sin\phi_0 = E_1\sin\phi_1 \pm E_2\sin\phi_2 + \text{etc}$$
$$E_0^2 = (E_0\cos\phi_0)^2 + (E_0\sin\phi_0)^2$$



14. Vector Diagram of Series Circuit

Reactive component of voltage in an inductance is positive; that in a capacity is negative. SECOND LAW (for multiple circuits) (Fig 15 and 16): The sum of all active components of current ($I\cos\phi = ER \div Z^2$) meeting at a point is equal to zero, and the sum of all the reactive components ($I\sin\phi = EX \div Z^2$) is equal to zero.

$$I_0\cos\phi_0 = I_1\cos\phi_1 + I_2\cos\phi_2$$
$$I_0\sin\phi_0 = I_1\sin\phi_1 + I_2\sin\phi_2 - I_3\sin\phi_3$$
$$I_0^2 = (I_0\cos\phi_0)^2 + (I_0\sin\phi_0)^2$$

eactive component of current in an inductance is negative; in a capacity, p

Ohm's law for series a-c circuits (Fig 13). In a circuit having resistance R, inductive reactance X_L, and capacity reactance X_c in series, the impedance in ohms is $Z = \sqrt{R^2 + (X_L - X_c)^2}$, the current $I = E \div Z$, and the power factor $\cos\phi = R \div Z$. Voltage across the resistance RI (Fig 14) is in phase with the current. Voltage across the reactance IX_L is 90° ahead of the current, and the voltage across the capacity IX_c is 90° behind the current. These combine geometrically to give $E = IZ = \sqrt{(RI)^2 + (X_L I - X_c I)^2}$, which differs in phase from I by the angle ϕ, the $\cos$ of which $= R \div Z$, and $\tan = (X_L - X_c) \div R$.

Multiple circuits (Fig 15, 16). In multiple circuits having resistance and reactance in various branches:

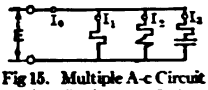


Fig 15. Multiple A-c Circuit with Resistance Inductance and Capacity

Conductance	$G_0 = R_1 \div Z_1^2 + R_2 \div Z_2^2 + R_3 \div Z_3^2 + \cdots$
Susceptance	$B_0 = X_L \div Z_2^2 + X_c \div Z_3^2$
Admittance	$Y_0 = 1 \div Z_0 = \sqrt{G_0^2 + B_0^2}$
Current	$I_0 = E_0 Y_0, \quad \cos\phi_0 = G_0 \div Y_0$

The negative sign is used to denote a lagging current in inductance in multiple circuits.

Resonance. If an inductance and capacity are connected in series, the voltages across them are opposite in sign; if equal in magnitude their resultant or difference is zero. This is resonance in series circuits. If an inductance and capacity are connected in parallel the two currents will oppose each other; if the currents are equal a large current may flow in each, but the resultant current in the supply circuit is zero or very small. This is resonance in multiple circuits. The phenomenon is utilized to regulate long transmission lines. The line has inductance, while near the load a synchronous motor is installed, which can act like a large condenser. By adjusting the excitation of the latter its capacity effect is varied until it neutralizes the line inductance, and it is possible to obtain a higher voltage at the load than is given by the generating station.

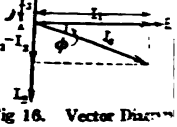


Fig 16. Vector Diagram of Multiple Circuit

Polyphase circuits. The single-phase a-c system, while simpler to calculate and best for lighting purposes, is not satisfactory for general transmission of power purposes. This is because: (*a*) a polyphase machine, whether generator or motor, weighs and costs less than a single-phase machine; (*b*) single-phase motors will not start from rest without complicated auxiliary devices; (*c*) in transmission work the 3-phase system requires only 75% as much copper for given power and voltage as the single or 2-phase systems. Therefore most commercial installations use 2-phase, while the great majority use 3-phase.

Two-phase system (sometimes called quarter-phase) consists of 2 circuits, which are electrically independent and come from independent circuits in the generator. Voltage of each circuit is the same, and it is endeavored to maintain same current in each. This is called balanced conditions. Generally 4 wires are required, though sometimes 3 wires are used; the latter arrangement is undesirable as it causes poor regulation. The wiring connections of 2-phase circuit are simple, as each phase is treated as an independent single-phase circuit.

Three-phase system saves 25% of the copper required for transmission line, as compared with any other commercial system, and is in general use. Theoretically, it consists of 3 independent circuits in which the 3 e m f's differ in phase by a period, or 120 electrical degrees. Since the currents in the 3 circuits combine to neutralize each other the 3 return wires are not necessary and the out-going current may use a wire returns by the other two without interference with regulation is, therefore, common in practice machinery to connect the 3 windings inside the machine so that only 3 conductors are brought out from the machine. There are two methods of making this connection. Fig 17 shows a Y-connected generator lea...

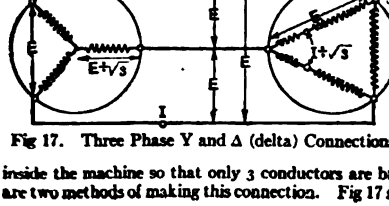


Fig 17. Three Phase Y and Δ (delta) Connections

base line, at the other end of which is a delta (Δ) connected motor. The different ges in this system have definite ratios as follows: Let E be the voltage between any es of the transmission line, and I the current in each line: Generator voltage per $= E \div \sqrt{3}$. Line voltage (neutral) $= E \div \sqrt{3}$. Motor voltage per phase $= E$. ator current per phase $= I$. Motor current per phase $= I \div \sqrt{3}$.
oice of Y or delta-connections is a matter of convenience, except for some exceptional of apparatus. To the operator it makes no difference which connection is used, and t it is impossible to tell without having the design drawings.

yphase transformations. There may be systems of 5, 6, or any number of phases, he 1, 2, and 3-phase systems are the only ones used for transmission. Some ma- have 6-phase windings, but these are always supplied by 3-phase transmission lines ethods of converting from one polyphase system to another see Art 13.

8. Alternating-current Generators

nchronous generators. Field structure consists of a number of poles, ed by coils carrying a d c. The relative motion of armature conductors ield structure generates the voltage. Frequency of the voltage depends tly upon number of field poles and rev per min of revolving part. Revolv- eld generators make more effective use of material than those with re- ng armatures, and are more easily insulated for high voltages. Most gener- of greater capacity than 300 kw are of the revolving-field type. Advantages: regulation, high efficiency, and satisfactory operation in parallel. Most ators built at present are separately excited, because voltage regulators ow available which are simpler and more reliable than the self-excited ines formerly in use. INDUCTOR GENERATORS are a special form of syn- ous machine having a revolving-field structure consisting of a spider with projecting poles, and one coil of d c which excites all poles.

duction generators are constructed like induction motors. The magnetic is of the rotating polyphase type, produced by a c in same windings as the current. An induction generator must always be connected to a synchro- generator, from which it must receive its excitation. Only a minority of nits of a station may be of the induction type. Advantage: less violent s in case of short circuits.

nnections (single, two, and three-phase). Single-phase generator is ly about 30% heavier and more costly than a polyphase of same rating. and three-phase generators of same capacity and voltage are practically me dimensions, weight, and cost. By changing the internal armature ctions any polyphase machine may be reconnected to a 3, 2 or single-phase ine. As transmission by 3-phase currents is more economical of copper by 2-phase or single-phase currents, all transmission lines are 3-phase; , the 3-phase generator is preferable.

ltage. Alternators are now built for voltages up to 13 000 and 15 000 en lines, either single or polyphase. For higher voltage, owing to extra f insulation and danger of damage, it is cheaper to install transformers a lower voltage alternator.

equency depends upon speed of rotation and number of poles. If the ve speed of revolving part is given in rev per min, the frequency is $f = (\text{r p m}$ les) ÷ 120. Formerly it was found more economical to run alternators gh speeds. Hence frequencies as high as 133 and 125 cycles per sec were mary; but, as systems increased in size and complexity, high frequencies d electrical difficulties. Usual frequencies are now 60, 50, 40, and 25 cycles c, of which 25 and 60 are standard in U S. In Europe 50 cycle id of 60. Recently there has been an attempt to have 15 or

standardized for railway work. Frequency of 25 cycles is preferable for re long transmission lines, because of the lower inductive voltage drop; and where there is much synchronous machinery (synchronous motors and rot converters), as these are more stable and better adapted to parallel operatio low frequencies. Higher frequency (60 cycles in U S, 50 in Europe) is pref able for electric lighting, as the light is steadier; it is also preferable where tr transformers are used, as these are cheaper and more efficient at high freque

Single phase. In a single-phase generator the voltage per phase is the sa as voltage between lines, and the current per phase is the same as current p line. The product of voltage and current gives the volt-ampere rating.

Two-phase or quarter-phase. Each phase supplies half the rating; th the voltage and current per phase in the machine are the same as the volt and current per phase of the line. The current is

$$I = \text{Power output in watts} \div 2\,E\phi$$

Three-phase machines may be either Y or delta-connected. In both cas the relation between output, current in each line, and voltage between lines the same:

$$I = \text{Power output in watts} \div \sqrt{3}\,E\phi$$

where I = amperes in each line; E = volts between lines; ϕ = load power-fact

Tests. Principal commercial tests on a-c generators are: resistance, cold and h friction; saturation and coreloss; synchronous impedance and load loss; heat run a insulation. Friction, coreloss, and saturation tests are made during one run; synchron impedance and load loss during another. For testing, the generator is driven by a m motor of about 1/10 the capacity of the generator. Synchronous impedance is use calculate regulation of the generator; results of the other tests give the efficiency. Lo are friction, coreloss, excitation, armature I^2R or copper loss, and load loss. Efficienc the output divided by the output plus all losses at that output. Armature I^2R and l loss vary as the square of the load; other losses are approximately constant. Heat r may be made by operating the generator under normal load conditions, or by a co promise method in which the generator supplies current to a synchronous motor w runs free and is under-excited, the current being wattless or reactive. Temp of wind after the heat run is determined by comparing their resistance at that time with th resistance when machine was at a known (room) temp.

Usual efficiencies at full load for different sizes of generator are given Table 8. These values are merely indications and vary with frequency, volta speed, power-factor, etc. A 60-cycle low-voltage machine will probably ha better efficiency than a machine of same rati for 25 cycles or high voltage.

Table 8. Usual Efficiencies

Rating, *kw*	Efficiency, %
100	91
500	94
1 000	95
2 000	96
3 000	96.5
5 000	97
10 000	97.2

Regulation of a generator is defined as the ra of the difference in terminal voltage at no load a at full load to the voltage at full load, the field e tation being kept constant at full load value the speed being constant. Expressed as a per ce age, the regulation is $100\,(V_0 - V) \div V$, where terminal voltage at full load and V_0 is term voltage at no load, with excitation for full loa Usual value, 5 to 15%.

Operation. In the operation of an a-c generator several factors are sidered: EXCITATION. A-c generators of the synchronous type (that in m general use), require d c to excite their field windings; usually obtained fr an exciter, which is driven by a separate engine, or by an electric motor, or f the generator itself by belt or other mechanical connection. Usual potent for field circuit is 125 volts, though some machines are designed for excitatio

volts. Since most machines have revolving fields, the d c is led into the coils by collector rings on the main shaft. In series with exciter and field uit a rheostat is connected, to vary the current in field coils, and thereby excitation and voltage of the generator. Usual power for excitation is from :o 2.5% of rating. Before STARTING A GENERATOR its bearings are inspected, ned, and oiled. Machine is then brought up to speed, and bearings again ected to see that oil rings are running properly. Exciter or excitation cir- is then put in readiness, and the rheostat in alternator-field circuit adjusted naximum resistance. Before exciting the field the armature insulation must horoughly dry; otherwise, the armature is short-circuited through an am- :r, and run for several hours at a partial excitation to give about rated current ie short-circuited armature. When insulation is thoroughly dry the short uit is removed and the excitation adjusted to give rated voltage in armature iinals at correct speed. To stop the machine, load is first removed by open- :he circuit breaker; then the field rheostat is turned to maximum resistance, also the rheostat in the exciter field if there is an individual exciter. Field uit is then opened.

ıralleling of generators. Before connecting a generator to the bus bars (to which or more other generators are connected), following conditions must be satisfied: requency of generator must be the same as that of bus bars; (*b*) frequency of gen- ır, and therefore its speed, be constant for an appre- e interval of time; (*c*) volt- of generator must be the : as that of bus bars; (*d*) rator and bus-bar voltage : be in phase. two machines have not the : frequency, or if frequency t constant, a condition will occur intermittently in which the voltages are 180° apart, e machines are in series on a short circuit, and a dangerous current will flow. If volt- are not equal, a large wattless current may flow, and if they are not in phase, a large r current will flow, causing a mechanical shock.

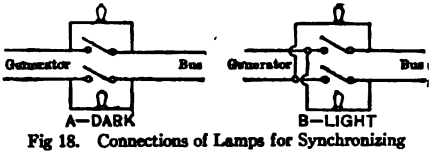
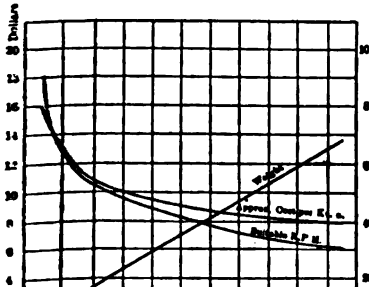


Fig 18. Connections of Lamps for Synchronizing

Two methods are used for determining when these conditions are favorable for connecting the machines together without disturbance; that is, for indicating when the machines are in SYNCHRONISM. For synchronizing with lamps, the connections as in Fig 18a are such that the lamps remain dark when above conditions are satisfied; with connections as in Fig 18b, they will remain bright. If the frequencies are wrong, the lamps will flicker (the slower the flicker the nearer the frequencies). If voltages are wrong, lamps will glow slightly but steadily. SYNCHRONOSCOPE is an instrument which indicates the relative frequencies and phases of a alternators; usually installed on switch board of a station, if a number of generators are used.



19. Cost, Weight, and Suitable Speed of Water-wheel Driven A-c Generators

ost and weight. Generators vary widely in their specific weights and ---- is, weight and cost per *kva* rating; but they may be divided into cla:

which these characteristics are fairly definite. Conditions affecting specific weight are: method of rating, speed, frequency, voltage, and size. Method of rating is fundamental. For comparison, rating is taken as the output in kw which each machine will give continuously with a temp rise not exceeding 45° C. On this basis, specific weight decreases as the speed, frequency or capacity increases, and it increases with increase of voltage. A-c generators are divided into 3 classes according to their speeds and purposes: high speed, as turbine-driven generators; medium speed, as belt-driven and water-wheel driven; slow speed, or engine-driven. Fig 19 shows approximate weights, costs per kva and suitable speeds, for medium-speed, 60-cycle, 2 400-volt polyphase machines. Weight and cost of 25-cycle generators are from 10 to 20% greater than for 60-cycle. These data are suitable only for preliminary estimates. Costs particularly are subject to large variations due to changes in commercial conditions from year to year. For final estimates exact dimensions and quotations must be obtained from manufacturers (see Art 19).

9. Synchronous Motors

Operation. Any alternator will operate as a motor. If two synchronous alternators are connected in parallel to bus bars supplying a load, and the driving power is removed from one prime mover, the alternator connected to this prime mover will continue to run at same speed, taking power from the other alternator and driving its own prime mover or other apparatus coupled to it; this alternator thus acts as a motor. Speed of such a motor depends solely upon the speed of the generator or generators supplying electric energy to it; it is therefore said to run in "synchronism" with source of supply and is called a synchronous motor. The speed of a synchronous motor having p poles and supplied with current of a frequency of f cycles per sec, is: r p m = $120f \div p$. If load on the motor increases, speed will not decrease, unless load reaches a value so excessive that the maximum output or "pull-out torque" is reached; then the motor will drop out of step and come to rest, while the current takes will increase to short-circuit value and torque will decrease to a negligible value. Difference in construction between an alternator and a synchronous motor is that the latter has, in the face of the field poles, a squirrel-cage winding, intended to give good starting torque and prevent "hunting" while running. HUNTING defines the occasional undesirable action of synchronous machines, of varying in speed, current, and voltage, at a frequency observable in meters. If excessive a short circuit results (see Specifications below). A standard 2 300-volt generator will operate satisfactorily as a motor at 2 080 volts, and as these are the natural values of the generated and delivered voltages, this characteristic of the synchronous motor accords well with customary distribution practice. A standard generator may have a squirrel-cage winding added to its poles and become a good synchronous motor.

Number of phases. Synchronous motors may be single, 2, or 3-phase. The single-phase motor is not self-starting, and has a considerably lower efficiency than the polyphase. It is also more liable to hunt and be unstable, and is therefore far less desirable than a polyphase motor. The 2 and 3-phase motors are very similar in all their characteristics.

Terminal voltage. Since synchronous motors are usually built with a revolving field and a stationary armature, the armature winding can be insulated for voltages as high as 13 000, thus often obviating need of transformers.

Relations of voltage and current. Relations between line voltage and phase voltage are the same as in a-c generators (Art 8). Current in each line of a 3-phase motor $I = \frac{746\,P}{\sqrt{3}\,\epsilon E \cos\phi}$, where P = output, h p; E = voltage between lines; ϵ = efficiency;

ed load; $\cos\phi$ = power factor (may be unity). Usual values for efficiency are the same as for a-c generators.

vantages of synchronous as contrasted with induction motors: higher effi-y, higher power factor, controllability of power factor with constant speed, voltage, lower cost. DISADVANTAGES: need of an exciter, will not start load, possibility of hunting.

plications. To transform from a c to d c, or from one kind of a c to ier differing in frequency, potential, or phase relation, motor-generator sets, sting of a synchronous motor direct connected to one or more generators, ften employed (Art 6). The potential of the secondary or distribution t is thus made independent of the variation in potential of the primary it supplying power to the motor. In certain cases it is desired to take power a 25-cycle circuit and supply power at 60 cycles for lighting purposes. a synchronous motor-generator set would be used; often called a "fre-y changer." In some applications of electric drive by induction motor ynchronous motor is installed to make it take leading current, in order to alize the lagging current taken by the induction motors. This effect is ıced by over-exciting the fields of the synchronous motor. The motor be used to drive any machinery not requiring much starting torque. Such tor is a "rotary phase modifier" or "rotary condenser " (see below).

its of synchronous motors. The first four tests of an a-c generator (Art 8) apply synchronous motors.

ıse characteristics, or V-curves at no load, full load, and any other specified load. ine is operated as motor with specified load kept constant throughout the run. ge and frequency impressed upon the motor are also kept constant. Current in the s varied from minimum at which motor will operate to the maximum (from 0.25 to rmal), and the variation in current input to armature noted. Readings are taken d, volts armature, amperes armature, and amperes field. A curve is plotted with es armature as ordinates, and amperes field or ampere turns per pole in field as sas. This gives the characteristic V-curves. At point of minimum current input ch load the power factor is unity. At lesser values of field current the armature it lags and power factor is poorer; at greater values the armature current leads.

rting tests. A low voltage is impressed on the armature and gradually increased the motor starts. Field circuit is open in two or more places, and a high-potential ltmeter is connected across one section to determine the voltage induced in field by the rotating magnetic flux. Test is repeated for several different initial positions revolving part, and a record made of time required for machine to reach synchro-Synchronous speed is attained when the induced voltage in the field becomes zero. ngs are taken of volts armature, amperes armature, initial position, maximum volts time to reach synchronism.

ıtallation and operation. Precautions to be taken in installation are the as for a-c generators (Art 8). Direct current must be provided for excita-If a synchronous motor is operated on a polyphase system having unbalanced ges, it will take unequal currents in the different lines and tend to balance the ges. But these unequal currents increase the heating for a given load.

ırting. Provision must be made for reduced voltage when starting the r, either by means of taps on the transformers or by a starting compensator 13). Large motors require 2 steps in starting, ½ and ¾ of normal voltage. motors will start with 1 step at ½ voltage. Field circuit is opened by eld "break-up" switch, and a voltage applied to the armature. When rmature current has decreased from its large starting value to a reasonable the voltage is increased, step by step. When motor reaches maximum the field is excited and the motor pulls into exact synchronism. Field nt is then adjusted until the condition of minimum armature cu

found, or until the power-factor indicator records a unity power factor. Lo may then be applied.

If a synchronous motor is to be started several times a day, a special starting m is desirable. This brings the synchronous motor up to a speed a little above synchron which is then effected as in a-c generators. Such a starting motor would require o about 30% of the full-load current of the large motor, and therefore have little effect the regulation of the system and not cause disturbance to lights and other motors on system. It usually requires less than one min to bring a motor up to speed.

Specifications. Synchronous motors are rated in same manner as synch nous generators, and same heating limits and specifications apply (Art 8). It customary to specify the value of current taken by the motor in starting with load other than the friction of its own bearings; or its own friction plus that the machine to which it is connected, in case it is part of a motor-generator s It is sometimes stated in specifications that the motor will not hunt, provid the total resistance drop between generator and motor is less than some speci value (10 or 15%).

Dimensions, weight, and costs. These are approximately the same as f synchronous generators of the same *kva* rating (Art 8, 19).

10. Induction Motors

Principles. Induction motor is a machine having distributed windings. E those of the armature of a d-c machine, on both stationary and revolving mem bers. One winding, the primary, receives polyphase currents from the supp circuit and thereby sets up a rotating magnetic field. This field cuts the co ductors of the secondary winding, induces currents in it, and thereby drags along with the field. The action is reversible; either member may be the pri mary, provided it is connected to the line. That member which remains stati ary, whether it be primary or secondary, is called the stator; the other, th rotor. In U S the stator is usually primary.

Synchronous speed. Speed of rotation of the magnetic flux is called th synchronous speed. At light loads the rotor speed is nearly equal to the syn chronous speed. If f is the frequency of the current, and p the number of pa of motor windings, the synchronous speed is: rev per min $= 120 f \div p$.

Slip. At any appreciable load the rotor speed is less than the synchron speed. Difference between the actual rotor speed N_1, and synchronous speed is "slip." It may be expressed as a percentage or a fraction: $S = (N - N_1) \div N$

Methods of rating. Amer Inst of Elec Engs recommends that the rat of an induction motor shall be the h p which it will deliver continuously at t shaft with a maximum rise in temp not injurious to its insulation (Art 1).

Starting and break-down torque. In addition to the ability to carry rated load without excessive heating and with reasonable values for efficien power factor, and slip, the motor should be able to start such loads as must brought up to speed, because good starting ability in an induction motor involv certain complications and expense. It is important also that the motor be able to carry momentary overloads without "breaking down," that is, gradua decreasing in speed to a standstill. To prevent this the maximum motor out put must be known and should be at least 50% greater than rated output.

Frequency and speed. Induction motors can be built for any frequency The higher frequencies are satisfactory where the load never exceeds the normal. Frequencies as low as 25 are preferable where overloads are common, or hig starting torques are required. Rotor speed of an induction motor at norm loads approaches within 5 to 10% of the synchronous speed, which is fixed b

requency of the system and the number of poles (see above). Hence, for ven frequency of supply circuit, only certain speeds are available. Thus, 5 cycles, the rev per min are: 1 500 for 2 poles; 750 for 4 poles; 500 for les; 375 for 8 poles, etc.

ase connections. Two-phase or quarter-phase motors are usually wound with inde- ent phase windings. Three-phase motors are connected in Y or delta, depending upon nience of designing engineer. In single-phase and 2-phase motors the voltage and nt per phase are the same as voltage between lines and current in line; in a Y-connected se motor, the current per phase is equal to line current, and voltage per phase is equal e voltage divided by $\sqrt{3}$; in a delta-connected 3-phase motor the current per phase is to line current divided by $\sqrt{3}$, and voltage per phase is equal to line voltage.

rrents taken by motors. Let: P_0 = output, h p; I = current in each line; e = ncy as a decimal fraction; $p = cos\,\phi$ = power factor as a decimal fraction; E = ge between lines (between one outside wire and the middle wire for 3-wire, 2-phase Then, for 2-phase: $I = 373\,P_0 \div peE$; for 3-phase: $I = 431\,P_0 \div peE$.

9. Power Factors and Efficiencies at Full Rated Load for Polyphase Motors

H p	25 cycles		60 cycles	
	Effic	Power factor	Effic	Power factor
1	0.79	0.78	0.78	0.78
5	0.85	0.88	0.82	0.88
20	0.88	0.91	0.84	0.91
50	0.90	0.92	0.87	0.92
100	0.905	0.925	0.89	0.92
200	0.91	0.925	0.905	0.92

sting. Induction motors may be given an "input-output" test (Art 5) at working from which efficiency, power-factor, slip, and maximum output may be determined; no-load excitation and a id short-circuit test, from i all characteristics may be lated. The former is sim- a stray-power test (Art 5) d-c motor, the latter to a nce measurement. No- tests require very little r, and are preferable in the of large motors, where it l be expensive to supply r and inconvenient to dis- the energy. They are ly accurate.

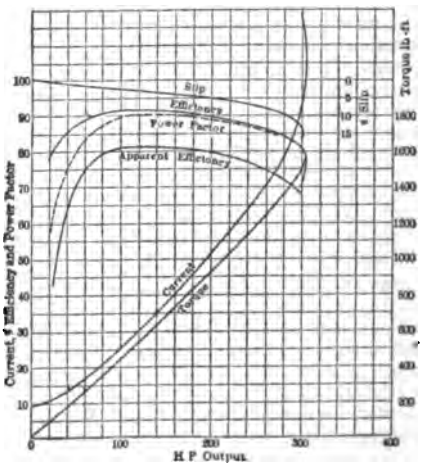


Fig 20. Characteristic Curves of a 3-Phase Induction Motor

aracteristic curves. Fig 20 examples of the usual char- stic curves of an induction . Typical break down of curves at maximum output wn at 300 h p. Power-fac- rve shows relation between rue power input and the ent power, or volt-amperes. power factor involves no er registration of the watt- meter, or cost of energy to te the motor, but does in- poor voltage regulation of the system as a whole and a larger capacity of w formers, etc. Apparent efficiency is the product of the power factor and tru

ency and is equal to output in watts divided by input in volt-amperes. Its value d
mines the actual capacity of the lines and transformer supplying the motor.

Starting. To start an induction motor without injurious heating, a resi
must be connected in the secondary circuit, or the voltage impressed up
primary must be reduced. These methods are known as RHEOSTATIC CON
and POTENTIAL CONTROL. They are also available to control the motor spe

Potential control consists in reducing and regulating the voltage impressed
primary usually by a starting compensator or auto-transformer, which provides
ratio of voltages. For this the secondary must be of higher resistance than
for the most of starting, but no changes of connection of the secondary are nec
Hence a squirrel-cage rotor winding is used, made of one bar per slot, all bars
connected at both ends to solid copper rings. To start the motor the primary
switch is thrown on the compensator, which gives a voltage of from ⅓ to ⅗ the rated v
a small motor and ½ to ½ if it is large. A small motor may be brought
speed with this voltage, but a larger one may require an intermediate step. When
speed indicated by the low, steady value of the current) it is connected
full voltage.

Rheostatic control. Better apparent torque efficiency, that is, more torque
given current, is obtained by inserting in the secondary circuit a much greater re
than can be permanently used. To do this a special starting resistance is connec
series with each of the armature windings, and a switch provided for short-circuiting
resistance either step by step or as a whole, as the motor speeds up. There are two
general methods. The first is for use only when the torque required at starting is
small, in which case the starting resistance may be small and placed within the arm
frame. The switch lever is so arranged that the resistance can be short-circuite
steps while the armature is revolving, thus obviating the need of collector rings and
connections. The second method, for large starting torque, consists of bringing the
ends of the secondary winding to collector rings. From brushes on these rings conduc
through external resistances, with steps or taps, so that the resistance can be short-circui
gradually by a controller.

Speed control. Speed of an induction motor may be controlled in 5 w
(*a*) By varying the potential applied to the primary of a motor having su
permanent resistance in the secondary. (*b*) By varying the resistance in exte
circuit of the secondary. (*c*) By so changing the internal connections of
primary winding as to change the number of poles. (*d*) By connecting two m
in "concatenation" (similar to series connection). That is, the stator of
motor is connected to the supply, rotor of first motor is connected direct
rotor of the second, and stator of second motor is treated as its secondary.
(*e*) By changing frequency of the applied voltage. Method (*b*) (rheostat
the most general and practical where more than 2 speeds are required; any s
may be obtained. Method (*c*) is best where only 1 motor is available and
2 speeds are required. Method (*d*) requires 2 motors and gives 2 sp
not as desirable as (*c*). Method (*e*) is of theoretical interest only.

Installation. Induction motors, even in large sizes, are usually self-conta
the bearings being a part of the end frame of the motor. They are either dir
connected or belted. Belt drive is the commoner, since an induction m
can be built only for a definite speed, corresponding to a certain number of
Small motors need no foundation, and are often attached to the wall or ce
the bearings and end shield being so made that they may be turned through
or 180°, for proper operation of the oil rings. Most motors must have ra
free ventilation, free from dust and dirt. For severe service, as in cement
or mines, motors are built totally inclosed ("iron-clad") and may then be ev
submerged in water. This increases both size and cost.

Operation. Small motors are designed to start merely by closing the
switch. With larger motors, if starting switch is in proper position, note

be applied to the motor, and the starting resistance gradually cut out by switch. Induction motors are very sensitive to variations in the impressed age. A decrease from 100 to 80 volts will cause the maximum output and ing torque to decrease from 100 to 64, with a roughly proportional increase eating for a given load. UNEQUAL VOLTAGES in the different phases cause ease in maximum output, and increase in heating for a given output. An alancing of 25% in voltage doubles the heating effect at full load; giving same heating as 50% overload. CARE IN STARTING. Before starting for irst time, see that the starting device is in operating condition and in proper tion, to avoid injurious heating. Wiring must be proportioned to carry the ing current without an excessive drop in voltage (see above).

ults. Some of the common faults in induction motors, with their indications and dies, are: SECONDARY OPEN-CIRCUITED. Motor will not start, and will not take a nt greater than exciting current; probably due to the starting resistance not being ected in. ONE PHASE OF SECONDARY OPEN-CIRCUITED. Motor tends to remain at synchronous speed, although the current is apparently normal. If the armature is ted, the currents in primary will be unbalanced. ONE PHASE OF PRIMARY OPEN. or will not start, and current will be unbalanced. ONE PHASE OF PRIMARY REVERSED. ents in primary will be very much unbalanced when motor is running, and starting ue will be very slight. SHORT-CIRCUITED COIL IN PRIMARY. There will be hum- when potential is applied to the motor and excessive local heating around the short- ited coil. VIBRATION due to mechanical unbalancing is chiefly noticeable at high ds, particularly in high-speed machines. If vibration is due to magnetic unbalancing, probably caused by inequality in the air gap at different portions of circumference, at different positions of armature. It may be detected by measuring the air gap taper wedges at different points around the circumference, with the armature in ral positions.

pecifications. Following memoranda (from Amer Handbook for Elec s) will assist in framing specifications: PRINCIPAL CHARACTERISTICS AND DITIONS OF SERVICE. Use to which motor is to be put; kind of load and hod of drive; voltage and number of phases; rating, horse-power; frequency speed. STYLE AND DESCRIPTION; CONSTRUCTION DETAILS. Whether to pen, semi-inclosed or inclosed; requirements regarding pulley and length of ft; whether rails are required; method of starting; compensator, external stance or internal resistance; whether motor is to be run at speeds other than speed; whether starting devices are to be supplied. PERFORMANCE AND TS. Final temp rise at full-rated load; temp rise in 2 hr at 25% overload applied ediately after full-load run; efficiency at 25, 50, 75, 100, 125, and 150% load; ting torque with full-load rent, ft-lb; high-potential s of insulation; require- nts as to effect of moisture n insulation.

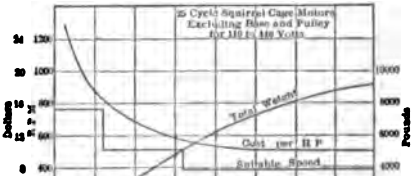


Fig 21. Weight, Cost, and Suitable Speed of a Line of 25-Cycle Induction Motors

Weight and cost of induc- n motors vary with type of nature winding and character nechanical frame and hous- ; also with speed, frequency, l voltage. Curves in Fig 21 l 22 indicate weights and ts of a line of 25-cycle and cycle motors respectively, ring squirrel-cage rotors (the mplest and least expensive type), and with simplest form of mechanical fr cy therefore represent minimum values (see also Art 19).

Single-phase motors may be of either induction or commutator type. I
DUCTION TYPE is the commoner, as its constant-speed characteristic is more ge
erally appropriate than the variable speed of the usual commutator a-c m
A 2 or 3-phase induction motor may be operated as a single-phase machine
it is once up to speed; but it has lower efficiency and power factor, and small
maximum output than it would have as a polyphase motor. Slip, for a giv
output, is less in a single than in a polyphase motor. Because of the poo
operating characteristics, and especially the smaller maximum output or ma
imum torque, the motor m
be rated at lower capacity w
run single phase. A sing
phase motor having same fr
and wt of iron and copper a
polyphase motor will rate
from 60 to 70% of the capa
Single-phase induction mo
usually consist of standard
phase motors with their ra
changed; thus a 10-h p, 90
3-phase motor would ma
typical 7.5-h p, 120-volt sing
phase motor. STARTING.
single-phase induction mo

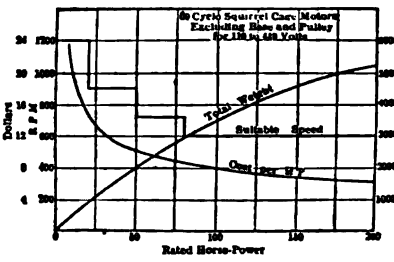


Fig 22. Weight, Cost, and Suitable Speed of a Line of 60-Cycle Induction Motors

has no torque at standstill, and must be started by some such means as a ph
splitting device. It may be started in either direction, and immediately dev
a slight torque, which increases with speed. When the torque reaches a valu
overcome friction and inertia the starting device may be disconnected, and th
motor will continue to accelerate to its proper speed. A small motor may
started without auxiliary aid by giving the armature a spin by hand. It will th
accelerate if there is no load. Some small motors are designed with a loose pull
which is clutched at a certain speed by a centrifugal governor. These motors ma
be provided with "shading coils," or with a small phase-splitting device, to gi
them enough starting torque to overcome their own friction. Large motor
are sometimes started by a phase-splitting device consisting of a resistance a
reactance connected to 2 windings of the motor. This gives an unbalan
3-phase voltage on the motor which produces a starting torque. Or, a commu
tor on the rotor may be used, making it possible to operate as a repulsion m
(see below) having good starting qualities. After reaching considerable spe
the brushes are removed and the secondary windings are short-circuited

Commutator type. The several types of a-c commutator motors designed to oper
on single-phase circuits differ chiefly in their electrical connections. They may be div
into series motors and repulsion motors. Both have an armature wound like that
d-c motor and a commutator. SERIES A-C MOTOR is connected like a d-c series mo
and has the same general characteristics. Principal difference is that the a-c mot
an extra winding placed in the face of the poles and connected in series with armat
and field. This winding improves the power factor. Limiting feature of most a-c c
mutator motors is sparking at the brushes, which is much worse than in d-c mot
REPULSION motor has a stationary structure or field (primary) with a completely dist
uted winding like that of an induction motor. The winding may be for any voltag
The armature or secondary is like that of a d-c commutator motor. The commutat
brushes are short-circuited upon themselves, and are placed at a small angle away fr
the neutral. This motor acts like a combination of transformer and series motor, ha
the variable speed, high torque, and runaway characteristics of the series motor. It
started by applying a reduced potential to the primary, and is reversed by moving the
brushes or by reversing the current in one particular portion of the primary winding known
as the exciting turns. REPULSION-INDUCTION TYPE of motor is made in small sizes as

: machinery requiring good starting torque. It is a repulsion motor with extra hes on the commutator, the brushes being connected across a portion of the primary. gives a constant speed characteristic like a shunt motor, and a good power factor.

11. Synchronous Converters

nce it is more economical to transmit electrical energy in the form of a c more convenient to utilize it in the form of d c, some means of converting trical energy from one form to the other is desirable. For this purpose hronous converters and motor-generator sets (Art 6) are available.

onstruction. A synchronous converter, also called "rotary converter," milar to a d-c generator, in which certain commutator segments, or the con- ors leading from them, are connected to 2, 3, 4, or 6 collector rings. When movable member rotates, the voltage between any 2 rings is alternating. 1 a machine may be operated as an a-c generator, or as a "double-current" rator giving a c from its collector rings and d c from its commutator. If rings are connected to an a-c source, the machine will run as a synchronous or, and d c may be obtained from the commutator brushes; that is, the ma- e with but one set of windings acts simultaneously as an a-c motor and a generator. It therefore has the friction, core-loss, and excitation loss of one hine instead of two; and since the motor and generator currents flow in e winding, and in opposite directions, they more or less balance each other, the armature RI^2 loss is much less than in either a motor or generator alone.

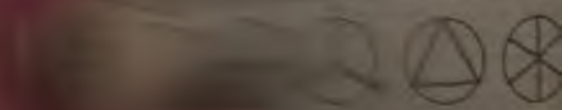
onverter *versus* motor generator. A converter is much more efficient, hs and costs less than a motor-generator set of same capacity, and occupies space. But, since only one winding is used, there is a definite relation veen the voltages of the a-c and d-c terminals. Maximum value of the mating wave bears a definite relation to the direct e m f (see below). The rerter must therefore be supplied with a voltage of the same order as the ct voltage, which requires transformers if a high-voltage transmission line ed to supply the converter. EFFICIENCY of a converter approximates 93%, of the transformers, 97%; the efficiency of the combination is therefore it 90%. Efficiency of a synchronous motor is about 93% and of a d-c rator, 92%; hence the combination motor-generator set has an efficiency 5.5%. If the supply voltage is greater than 13 000 volts, transformers will be needed for the motor-generator set, the net efficiency being then 83%.

ctifier differs from a converter, in that the latter gives a direct e m f of constant and rm value, and the former a pulsating unidirectional voltage and current. In the erter the energy is stored for an instant in the form of magnetism; in the rectifier : is no magnetic field and no storage of energy. A rectifier will not work on an in- ive d-c circuit, but a converter will.

pplication of converters. The commonest application of synchronous rerters is in electric railway work. Nearly all motors for electric traction of d-c series type, operating at 500 to 600 volts. For this service the energy ansmitted over long distances, requiring a high-voltage a-c transmission and converters to link the d-c distribution with the a-c transmission.

ases and rings. SINGLE-PHASE CONVERTER has 2 collector rings, each connected to windings by as many equally spaced taps as there are pairs of poles. The taps for rings alternate at equal spaces. A single-phase converter is therefore a 2-ring erter. THREE-PHASE CONVERTER has 3 rings and 3 equally spaced taps (one for each for every pair of poles. A 4-phase or quarter-phase converter has 4 rings and 4 and a SIX-PHASE CONVERTER has 6 rings and 6 taps per pair of poles.

unt and compound-wound converters. A converter may be shunt or compound d, depending upon the service. Series winding is intended to make the com



nsformers to supply converters. The transformer secondaries must give a tage which is a definite fraction of the direct voltage desired (Table 10).

Starting. A converter may be started by either of the methods described synchronous motors, or it may be speeded up as a shunt motor, and then chronized with the a-c supply circuit like an alternator (Art 8). All conters are supplied with a "field break-up switch," which opens the field circuit several places to avoid the strain of the high potential induced in field windgs during starting, and to reverse direction of field current after the machine ıp to speed, in order to reverse the polarity if necessary. This is usually a ible-throw switch with several poles. For normal operation it is thrown rn to "running" position. To change the polarity it is thrown up to "rese" position and left closed only for a moment; then it is returned to nning" position. SPEED-LIMITING DEVICE. In case a converter should be connected from the main a-c supply circuit, and still remain connected to the circuit, it would tend to operate as a d-c motor, and its speed might become ngerously high. To avoid this a centrifugal governor is placed on the shaft l arranged to open the main d-c switches of the converter.

12. Electric Power Plants (See also Sec 16)

Location of an electrical power plant is governed by: Form of power; disnution of load; cost of ground; possibility of good foundations; room for ension; convenience of coal, and of water supply for evaporation and consing; limitation of voltage used.

Form of power, whether steam or water, is usually definitely determined by al conditions. Choice between steam, gas, or oil depends chiefly on relative t of fuel and of maintenance of an oil or gas engine. If steam is to be used, choice between turbines and reciprocating engines is governed largely by of units; in large sizes the steam turbine costs less per h p and is cheaper operate, while in small sizes the reciprocating engine is more economical. designing a power plant and selecting the number and size of units, it is portant that the total capacity be sufficient to carry the maximum load with rgin, and that it be divided into units of a size to carry average load economily. Each unit is usually capable of carrying 25 or 50% overload for 2 hr. ere should be at least 4 units in a large plant, and nothing is gained by having re than 8 units. A sufficient number of exciters are necessary to carry the ık load, with one exciter to spare. Transformers are commonly arranged in ups, one group for each generator. In hydro-electric plants it is often visable to install 1 or 2 steam-driven units and boilers, to serve if water supply uld fail (see also Sec 16).

Alternating vs direct current. Choice depends upon length of transmission and value of energy. For utilizing all the energy within a short distance the generating station d c is preferable, because d-c motors (particularly in all sizes) are more convenient, easier to control, and are available in greater riety of form and characteristics than a-c motors. But the voltage obtaine from a d-c generator is limited by the commutator to about 600; and, for nsmission to a considerable distance by d c, the quantity of copper becomes hibitive; hence a c is used, and with the a-c transformer the transmission potential is independent of the voltage at either generators or motors. The iding line between the best fields for d-c and for a-c plant is not clear cut, l can be determined only by careful calculation and comparison of costs. ne installations use a-c generators and a c for transmission, but these are ıverted into d c in sub-stations near the point of utilization, and at a di m the generating station. When the sub-station cost is less than th

ence between the cost of the copper requisite for d c and for a c, the a-c system with sub-station is preferable. If the a-c system be chosen, 3-phase generators are advisable, as 3-phase transmission requires 25% less copper than 2 phase or single phase, and there is no advantage in the 2 phase over the 3 phase. If the power plant is for a varied service, including lights as well as motors, the frequency should be 60 cycles, because incandescent lights operated at 60 cycles are satisfactory, while at 25 cycles the flickering is harmful to the eyes. Transformers for 60 cycles are considerably cheaper than those for 25 cycles. But if the system is primarily for transmission, and at the place of utilization comparatively large synchronous motors or converters, then 25-cycle frequency is better, as this type of machine sometimes gives trouble when operated at 60 cycles.

Voltage. The most convenient and widely used voltage for a local power plant is the 220–240 volt 3-wire system, in which two 110-volt generators are connected in series, or a 220-volt generator and a balancer set provide the two voltages. In this case the electric lights and small motors are connected to the 110 volts from the neutral to either of the outside wires, while all the larger motors receive 220 volts from the 2 outside wires, thereby saving in copper. If 110 volts were used, either an excessive amount of copper would be required, or the voltage on the lamps would vary enough to cause dissatisfaction. For a number of large motors, 500 volts is generally used, but the lighting problem then becomes difficult, as the lamps must be connected in groups of 4 in series, and if any one of the 4 burns out, all in that group go dark. 500 volts is also undesirable in damp places, as in a mine, because there is danger of an unpleasant shock, particularly to horses or mules. The 220-volt, 3-wire system is therefore best, if d c be used. For the 3-phase a-c system the distribution circuits are best run at 220 volts between lines, giving a good voltage for motors, while the lights may be distributed on the 3 phases by transformers giving 110–125 volts on the low potential side. Lowest standard voltage for transmission is 2 200, which will take care of reasonable loads within a radius of 1 or 2 miles from the power station, stepdown transformers being used to supply the different sections of the area.

Arrangement. It is common practice to divide the power station, if steam or gas driven, into two parts, separated by a fireproof, dustproof wall. On one side are the boilers or gas producers; on the other, the prime movers, generators, and all electrical apparatus. In the simplest form the boilers are set in a single row along one side of this wall, and the generating units along the other side, the piping being so arranged that any engine may be supplied from any boiler. If transformers are used, they are generally placed in the room with the generators, but on opposite side and on the lowest level, the switchboard being in a gallery above. Cables from generators to switchboard are run underneath the floor and running up the side of the wall.

Switchboards are of 2 kinds: DIRECT CONTROL, in which case the current is carried by bus bars immediately behind the switchboard panel, the opening and closing of switches in this circuit being effected directly by the operator; and REMOTE CONTROL, in which the current is carried by conductors placed in brick compartments in the basement, and the connections are made by electrically operated switches controlled by a small relay circuit coming from control switches on a benchboard above. If potential is 2 200 or less, and power is less than 5 000 kw, direct control is satisfactory and feasible, but for greater voltage or power the remote control system is preferable.

Switchboard is divided into a number of units, each known as a panel. There are generator panels, exciter panels, and feeder panels, and at one end of the switchboard is a voltmeter; also, in an a-c station, a synchroscope is placed on a swinging bracket which may be distinctly seen from any part of the switchboard. A circuit-breaker is a switch which opens the circuit when current is excessive; operated by hand or automatically by

current itself. While the make-up and arrangement of the different panels may vary tly according to individual ideas, the essentials are common to all, as follows. D-C ERATOR PANEL, 2-wire system: 1 circuit-breaker at top of board, 1 ammeter, 1 hand-el for rheostat, 1 field rheostat back of board, 1 single-pole field switch, 1 triple-pole a switch (or 1 double and 1 single-pole) for the equalizer, 1 4-point voltmeter receptacle. FEEDER PANEL 2-wire system: 1 single-pole circuit breaker, 1 ammeter, 1 double or ugle-pole main swtiches, 1 4-point voltmeter receptacle, 1 watt-hr meter. A-C GEN-TOR PANEL, 3 phase: 3 ammeters, 1 3-phase wattmeter, 1 field ammeter, 1 double-pole switch, 1 handwheel for rheostat, 1 synchronizing receptacle, 1 potential receptacle, phase oil switch, 3 disconnecting switches, 2 current transformers, 1 potential trans-er, 1 voltmeter for group, 1 synchronizer for group. A-C FEEDER OR OUTGOING LINE EL, 3-phase: 3 ammeters, 1 3-phase oil switch, 3 disconnecting switches, 1 polyphase -hr meter, 3 current transformers. A VOLTAGE REGULATOR may be used, requiring adependent panel.

osts. See Art 19.

13. Electric Transmission

ypes. Transmission may be in form of single-phase, 2-phase or 3-phase or as d c. Single-phase and d-c systems require 2 wires; 3-phase, 3 wires; ase, 4 wires. For given voltage between wires the relative weights of copper Single-phase and 2-phase, 100; 3-phase, 75; d c, 50. Because of the ted voltage at which d c is obtainable, it is not applicable to long distances. le-phase system is used where the object is lighting only, on account of its le connections. For important transmission problems the 3-phase system ld be chosen. Respecting the transmission line only, 25 cycles is preferable o cycles, due to the lower inductive and capacity effect; but, for ordinary smission problems, the higher frequency is preferable, because of the lower of 60-cycle transformers.

ine drop is the difference between voltage at generating station and at the l. It is usually expressed as a percentage of the delivered voltage, and t transmission lines are designed for 10% drop when transmitting full or d load power. In general, the weight of copper required varies inversely he line drop allowed. This is not strictly true in a-c systems, but is exact in systems, and the relation may be written: $W = 125\,PD^2 + KE^2$, where wt of copper, lb; P = power delivered, watts; D = distance one way, in sands of ft; K = ratio of power lost in line to power delivered; E = voltage vered.

an a-c system the inductive reactance must be considered, as well as the resistance. stance for a certain length of given size of wire is obtained from a wire table, and is ated for the given distance. Inductance in henries per mile for one wire of a trans-ion system is given by: $L = 74 \times 10^{-6} \times \log_{10}(2D \div d)$, where D = distance between s, and d = diam of wire, both in inches. Reactance in ohms is $x = 2\pi fL$, where f e frequency. In a single-phase transmission the total resistance and reactance are e the above values. In a 2-phase circuit, the resistance and reactance per phase are e the above values. In a 3-phase circuit, the resistance and reactance per line or per ase are used, and these are the values given by the above formulas.

ine regulation depends upon: line resistance and reactance, the current, and power or of the load. It may be calculated by: $E_0^2 = (pE_1 + IR)^2 + (IX \pm qE_1)^2$, where voltage at generating end; I = load current per line; $p = cos\,\phi$ and $q = sin\,\phi$ at load; qE_1 is negative for anti-inductive load; E_1 = voltage at receiving end; X = re-nce of line; R = resistance of line. For single-phase, E_1 is the voltage at the load, and nd X are taken as the sum of outgoing and return circuits. For 2-phase, E_1 is the age per phase at the load, and I is the current per phase in a 2-phase system (Art 7); nd X being taken for the complete circuit of one phase, outgoing and return. For ase, E_1 is the voltage to neutral at the load = 0.58 × voltage between lines; I is the ent per line, or Y current; R and X are taken for one line, one way. This gives E_0, voltage to neutral at generating station. Voltage between lines is then 1.73 E_0.

Size of wire for an a-c transmission. Select first a wire which gives an allowable resistance drop (IR = 5 to 10%), and then find loss in voltage [E_0 (generator) $- E_1$ (load)] due to combined effect of R and X. If loss be too great, select a larger wire and try again. To obtain a solution in one calculation is too complicated a problem except for experts. Remember that the impedance drop in the line (IZ) is usually much greater than the difference between generator and load voltage due to vector relations. Usual voltages for transmission wires are: 2 200, 6 600, 13 000, 19 000, and 33 000 volts between lines, whether single, 2 or 3-phase. A rough rule for choice of voltage is to allow 1 000 volts for each mile of distance between generating station and load. Usual sizes of wire are from No 4 to No 0000 B & S gage in solid conductors and larger sizes in stranded conductors (Table 4). Smaller wire than No 4 has insufficient mechanical strength. Wires larger than No 0000, if solid, would be too stiff, and would break from the repeated bending to which they are subjected.

Stranded conductor. Gage number or rating of a stranded conductor is that of a solid conductor having same cross-section of metal. Resistance and weight of a stranded conductor are greater for a given length than of a equivalent solid conductor, due to the twisted path (and hence greater length) of the individual wires. Actual length of path of the current is increased by about 7%. In spacing the 2 wires of a transmission line it is usual to allow about 1 ft for every 10 000 volts.

Transformer, commonly called static transformer, is used to transform a system of a given voltage to a higher or lower voltage. Single-phase transformer consists of 2 electrical circuits, usually of a large number of turns, interlinked with a common magnetic circuit of iron. Since the power is approx the same in both windings, the currents are inversely proportional to the voltages. Polyphase transformer is essentially two or more single-phase transformers, made into a single piece of apparatus, and so designed that at least part of the magnetic circuit is common to all the phases. A 3-phase transformer has 3 high-tension and 3 low-tension windings, arranged on an iron core (Fig 24, 25). The winding by which energy enters the transformer is the PRIMARY; that by which it leaves, the SECONDARY. Since either winding may be connected to the source of energy, these terms are not definite unless the manner of connection is also stated. HIGH-TENSION WINDING and LOW-TENSION WINDING are used to distinguish the two windings, the high-tension being the one with the greater number of turns. When the high-tension winding is connected to the source of supply, it is a STEP-DOWN TRANSFORMER; when the low-tension winding is so connected, it is a STEP-UP TRANSFORMER.

Classification of transformers. (A) according to operating characteristics. CONSTANT-POTENTIAL transformers are intended to give an approx constant potential on the secondary side; CONSTANT CURRENT, to give an approx constant current on the secondary for arc lamps. Both are designed to operate on a constant-potential supply circuit. SERIES TRANSFORMERS are connected in series with the main circuit and receive the line current in the primary. The secondary circuit includes only a meter and the secondary current will be inversely proportional to the transformer ratio. They usually supply low-reading ammeters and wattmeters from circuits carrying very heavy currents. AUTO-TRANSFORMERS OR COMPENSATORS, sometimes called single-circuit transformers, consist of one electric circuit interlinked with a magnetic circuit and a tap brought out from some part of the winding. As the voltage between this tap and either terminal of the circuit is a fraction of the total, a fractional voltage may be secured. Windings on each side

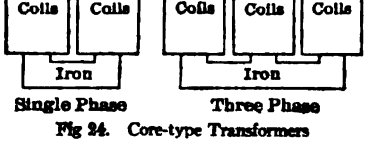
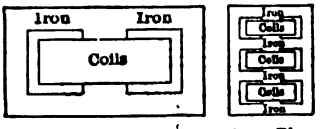


Fig 24. Core-type Transformers

tap are usually proportioned to the current to be carried. Auto-transformers are ble where the ratio of voltages approximates unity, as they then require much less than regular transformers. POTENTIAL REGULATORS are transformers in which the e of one member may be varied from zero to a fixed maximum, either by changing the on of the magnetic flux or by changing the phase of the e m f of the secondary with t to that of the primary.

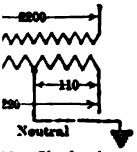
according to construction. There are two methods of arranging the electric and tic circuits of transformers, the corresponding construction being called "core-type" shell-type." CORE-TYPE transformer (Fig 24) has a single magnetic circuit inter- with 2 electric circuits, each electric circuit containing primary and secondary coils. est adapted to small sizes or high volt- nd to oil cooling, and is the commoner. -phase core-type is a combination of 3 -phase transformers into one, to save ial and space. SHELL-TYPE (Fig 25) magnetic circuits in parallel, interlinked electric circuit containing primary and dary. Best for air cooling and large its. Three-phase shell-type is a con- tion of 3 transformers into one.



Single Phase Three Phase

Fig 25. Shell-type Transformers

according to method of cooling. As a transformer is a very compact piece of appa- the problem of carrying away the heat is important. NATURALLY-COOLED type has ecial means of cooling, but relies upon ordinary circulation of air. Only used in small sizes, as for meters. In the OIL-COOLED, the core and windings are submerged letely in a tank of oil, and are subdivided by ducts so that oil may circulate and act heat from internal parts. The heat is carried to surface of tank containing the former, and thence dissipates into the surrounding air. Tank must be specially ed, with deep corrugations or projecting vanes or tubes, to provide large air-cooling ces. In the AIR-BLAST type there are passages through which air is forced by a r. WATER-COOLED transformer resembles the oil-cooled type, but a coil of pipe ing running water is submerged in the oil. Transformers artificially cooled by oil are used when the size is too great for the self-cooling oil type and no water is available. Oil circulates through external coils or tanks, to afford greater cooling surface.



16. Single-phase, Three-wire

Transformer connections commonly used in lighting and power service are: SINGLE-PHASE SYSTEM with 3-wire secondary (Fig 26) is standard practice in residence lighting with a-c system, the neutral wire being grounded on the low- tension side, the primary side not being grounded. Lamps or motors operating at 110 volts are connected between the ral and either side. Maximum potential between any secondary and ground o volts, but if either outside wire becomes grounded a short-circuit occurs hat half of the transformer. TWO-PHASE, or quarter-phase 4-wire system. standard is essentially 2 independent single- e systems, which are usually electrically pendent throughout. TWO-PHASE 3-wire em is occasionally used for distribution of er in small systems. Return wire is common oth circuits. There may be a slight saving pper, but the chances of unbalanced voltage bad regulation, especially with an inductive , render it objectionable. THREE-PHASE Y DELTA CONNECTIONS (Fig 27, 28). Trans- ation in a 3-phase system, with 3 single- e transformers or one 3-phase transformer, ng 3 primary coils and 3 secondary coils, is accomplished by connecting the ary either in Y or in delta, and the secondary either in Y or in delta. owing relations exist between voltage per transformer winding and v

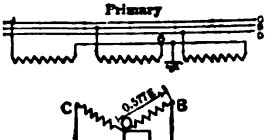


Fig 27. Three-phase Y

between lines, current in transformer windings, and current in lines. Pow the 3 transformers in any case is $3 \times 0.58 \times EI = 1.73\ EI$. THREE-PHASE

Connection	Volts between lines	Volts per winding	Current per line	Current pe winding
Y	E	$0.58E$	I	I
Delta (Δ)	E	E	I	$0.58I$

DELTA, or V-connection, saves expense by omitting one transformer from t delta connection. Is recommended only for low voltages, as 2 300. Regulati

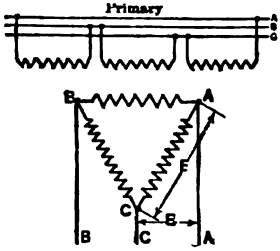


Fig 28. Three-phase Delta

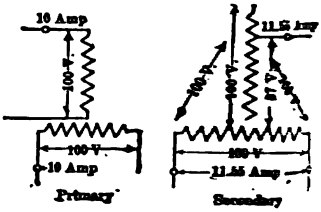


Fig 29. Two-phase to Three-phase

and efficiency are poor, as one phase of the load receives power from 2 tra formers in series. Aggregate capacity of the transformers should be 15% grea than the load. TWO-PHASE TO 3-PHASE (Fig 29). Scott, or T-connection, co sists of 2 transformers which, on the 2-phase side, are connected in norm 2-phase manner. On the 3-phase side one transformer has a tap at the midd point, the other a tap giving 87% of full-transformer voltage. Fig 29 sho method of connecting, and the currents in primary and secondary with balan loads. Total transformer capacity must be 15% greater than the load.

Efficiency is equal to the power output divided by output plus coreless p total copper loss, and is very high. "All-day efficiency" is equal to the en output for a day divided by the input; with short periods of load it is low, bec coreless goes on when there is no load. If is the load in watts for h hr per day, A the coreless, and B the copper loss, the all-da efficiency is $100\ kP \div (kP + 24\ A + kB$ percentage.

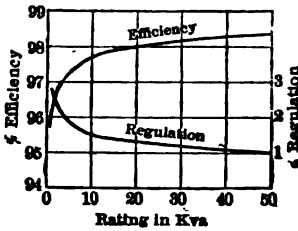


Fig 30. Efficiency and Regulation of a Line of Transformers

Regulation is $(E_0 - E) \div E$, where E_0 secondary voltage at no load and E = ondary voltage at full load. It should from 1 to 4% for load of 100% power fac and is poorer for inductive loads. Fig shows efficiency and regulation at full lo of a line of 60-cycle lighting transformer for 2 300 volts.

Installation. Transformers require no special foundations, but merely pro vision to carry their dead weight. They must be in a dry place, unless of the totally enclosed "subway" type. They are frequently placed on poles. They must be thoroughly cooled.

Cost and weight. (See Art 19.)

station is a building containing an equipment for transforming or converting the of transmission line into a convenient form for use in small units. Since the may be used as a c or d c, there are 2 general types of substations: a c to a c and d c. In either case the energy must be distributed to the various consuming de- t a much lower voltage than that of transmission, sometimes at a different fre- , and sometimes in an entirely different form. The substation is therefore the g point between the transmission and distribution systems. A-C SUBSTATION in it form comprises step-down transformers for lowering the voltage, and incidental ing and protective devices. There may be a group of 3 single-phase transformers, 3-phase. Single-phase transformers are preferable for moderate sizes, as they re- smaller reserve. Since the transmission system is always 3-phase, these groups e low-voltage 3-phase currents for different groups of motors, or the individual may be wired separately for lighting purposes. If transmission system is 25 cycles be necessary to install motor-generator sets (Art 6) (frequency changers) to give le current for lighting. These usually consist of a synchronous motor driving an tor. If the distribution system voltage must be kept constant, some form of e regulator is installed, which will keep the voltage constant on each individual , or on the 3-phase circuit as a whole. A switchboard with the necessary switches easuring instruments is requisite. D-C SUBSTATION consists of stepdown trans- s, converters, or motor-generator sets, with incidental regulating and protective s. The converter (Art 11) is cheaper and more efficient than the motor-generator d is advisable in absence of any local reason for using motor-generators. Con- may be arranged to give constant voltage irrespective of load, or may be over- unded. The substation then resembles a d-c generating station.

htning arresters are to protect the apparatus of a station from natural ing discharges, and from "internal lightning" caused by sudden changes rrent or voltage conditions in the system itself. A lightning arrester its a current to flow from conductor to ground, whenever the potential ds the normal, and stops this current when the excess voltage ceases to This is usually done by letting the excess voltage break down a spark by means of an arc which stops as soon as the voltage drops below a critical depending upon the width of gap.

es of arresters. HORN-GAP ARRESTER is simplest and is commonly used for high es, but is not sensitive enough for moderate and low-voltage systems. It consists pper rods, bent like horns, separated by a fraction of an inch at the nearest point iverging from there. The arc starts at the small end of gap, lengthens as it rises, nally breaks. MULTIGAP ARRESTER comprises a number of small cylinders of spe- n-arcing metal, through which the circuit to ground is made. When voltage rises essive value it causes the current to jump all the small gaps in series between these ers and establishes a circuit to ground. When the potential is relieved the arcs die In more elaborate form, several groups of cylinders are arranged in parallel, and in with different resistances, to take care of potentials of different values and charac- cs. ALUMINUM-CELL arrester, the latest, best, and most expensive form, consists of k of aluminum pans, separated by an alkali electrolyte forming electrolytic cells. ch plate a film is formed, which, for reasonable potentials (200-300 volts per cell) nts any current from flowing, but with excessive voltage the film breaks down and s a large current to flow. For high voltage a large number of plates and a short gap are usually connected in series. The gap prevents a capacity-charging current flowing, but must be closed once a day to keep the film in condition.

14. Electric Distribution

inciples. Three fundamental considerations determine the cross-section weight of copper conductors for a particular purpose (see also Art 13): Voltage lost in the line, or IR drop. This depends upon the line current the resistance of outgoing and returning lines. It is important in affecting aleability of the energy and desirability of the apparatus served. It causes derable variation in the light given by lamps, and a variation in speed of ors. (*b*) Energy lost in the line due to I^2R, which represents an

money loss. (*c*) Current density in conductors. This determines their te rise, which, if excessive, causes danger of fire.

Usual loss in voltage allowed in distribution systems is less than in tr mission; it varies from 2 to 5% of the delivered voltage. In a d-c syste proper cross-section of conductor in circular mils (Table 4) is given by: (21.6 DI) + e; where D = distance one way, ft; I = current, amperes; specified loss, volts. In an a-c system the drop due to resistance of wire is ally the determining factor; reactance drop is probably negligible, as the are strung close to each other. Choice between a c and d c for distribution depends upon character of the load apparatus and the available current. motors are generally preferable, and therefore d-c distribution systems are unless the cost of the converting substation is too great.

Examples

Series-arc system (Fig 31). Problem: 80 arc lights, requiring 10 amperes at each in a 5-mile series circuit. Allow 3% loss. Voltage consumed = 50 × 80 =

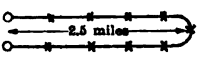


Fig 31. Series-arc Distribution

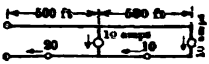


Fig 32. Multiple Distribution

Let R = resistance per 1 000 ft of wire. Length of wire in thousands of feet = 5 = 26.4. Voltage loss = 4 000 × 0.03 = 120 = 10 × 26.4 × R; hence R = 0.455. Table 4, the nearest size of wire is No 7 B & S.

Multiple system (Fig 32). Problem: two loads at 500 and 1 000 feet respectively distribution center. Each load, 10 amperes. Allowable maximum drop, 5 volts size of wire. Let R = resistance per 1 000 ft of wire. Then, 5 = 20 × 2 × 0.5 R + 10 × 0.5 R; hence R = 0.166. Choose No 2 B & S wire, of which R = 0.156.

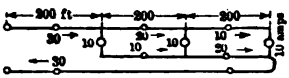


Fig 33. Anti-parallel Distribution

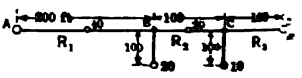


Fig 34. Tapered-conductor Distribution

Anti-parallel system. Problem: find size of wire to give a maximum drop (on m group) of 5 volts on loads shown in Fig 33. Let R = resistance per 1 000 ft of Then, 5 = 30 × 0.2 R + 20 × 0.2 R + 20 × 0.2 R + 30 × 0.6 R; hence R = Choose No 2 B & S wire.

Tapered conductor. Problem: given 3 loads as in Fig 34, and maximum dro 5 volts, to find sizes of wires. Assume drop proportional to distance: AB = 1.25, CD = 1.25. Then,

2.5 = 40 × 2 × 0.2 R_1; hence R_1 = 0.156 per 1 000 ft; choose No 2 wire.
1.25 = 20 × 2 × 0.1 R_2; " R_2 = 0.312 " " " " " 5 "
1.25 = 10 × 2 × 0.1 R_3; " R_3 = 0.625 " " " " " 8 "

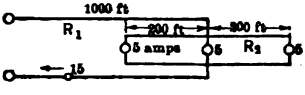


Fig 35. Feeder and Main Distribution

Fig 36. Three-wire Distribution

Feeder and main. Problem: given loads as in Fig 35, and max drop of 10 Find size of wire. Assume drop to junction point = 8 volts (0.75 of total).

Feeder, 8 = 15 × 2 R_1; hence R_1 = 0.266; choose No 4 wire.
Main, 2 = 5 × 2 × 0.2 R_2; " R_2 = 1.0 " " 10 "

ree-wire system. Principle: at any given distance from generator the sum of out- currents equals sum of returning currents (Fig 36). Problem: Given 3 loads, ad 400 ft from generator, as shown. Neutral or middle wire to be same size as out- ires. Max drop, 5 volts. Let R be resistance per 1 000 ft. $5 = 10 \times 0.4\,R + 0.2\,R + 15 \times 0.2\,R$; hence $R = 0.555$. Choose No 7 wire.

nstruction. Distribution systems may be overhead or underground. rground system costs about 5 times as much to install as the overhead, but o maintain. Steel poles are used for high voltage or regular transmission ms, but wooden poles are satisfactory for ordinary distribution. Cedar, nut, pine, and cypress are good, costing $5 to $10 per pole, according to ion. They are usually 35 to 40 ft long, have a taper of 1 in in 5 to 8 ft. to 8 in diam at top and set 5 to 8 ft in ground. Spans, from 75 to 125 ft, s are guyed to anchors on curves, at ends, and at every 20th pole, to take of breaking of wires. Glass insulators on cross arms are used for moder- voltages, porcelain insulators for higher. In stringing wires, sag must be sted with reference to the temp; a large sag in summer, small in winter.

cities most of the distribution is laid underground in elaborate conduits, which are nsive and unnecessary for mining work. Good results are obtained by laying a lead- ed double conductor cable in a shallow trench and covering it with earth.

terior wiring of buildings. Definite methods, practices, and regulations covering llation of distributing circuits are based on following considerations: (*a*) avoidance nger from fire or risk of shock; (*b*) limitation of loss of power and energy; (*c*) main- ice of reasonably constant voltage at the load; (*d*) assurance of reasonable mechani- trength; (*e*) compliance with local specific regulations. STANDARD VOLTAGES: for ing and incidental power, 100 to 125 volts; for power and lighting on 3-wire system, o 250 volts; for power alone, 550 volts. Usual allowance for drop in building wir- 2%. LOCAL REGULATIONS to be observed: National Electric Code, for obtaining ance on buildings; Municipal Inspection in cities; regulations of local Light and er Co, if energy is purchased. These regulations are so elaborate that any one ex- ing to be responsible for an installation of wiring must be acquainted with the National tric Code, on which most of the other regulations are based. ALLOWABLE CURRENT- YING CAPACITY OF WIRE. Values given in Art 3 are for rubber-covered wire installed ors. For other insulation, and for exterior work, somewhat greater current is allowed. ying out a distribution system, a CUT-OUT OR FUSE must be inserted at every junction e size of wire changes, and there must not be more than 660 watts in lamps on any fuse. Larger motor loads may be taken care of by a CIRCUIT-BREAKER.

ethods of indoor wiring. OPEN WIRING ON CLEATS. Wire must be insulated with erial resisting fire, and dampness and corrosive vapors, if present. Wires must be rated from each other and from any nearby surface by specified distances and the ts may be spaced not more than 4.5 ft apart (local regulations). No 14 is the smallest rable wire. MOLDING, either wood or metal, may be used to hold and protect wires, ifference in potential does not exceed 300 volts and the power transmitted is within ified limits. KNOBS AND TUBES. If not prohibited by special regulations, wires may ttached to knobs and run through walls in porcelain tubes; a cheap form of wiring. ONDUIT. Best but most expensive method is to run the wires in conduits, built into walls, floors, or framework of building. Conduit may be lined or not, which has a ring on amount of insulation required on the wires. Conduit may be rigid or flexible; former is better and more expensive, and is often required in large cities. ARMORED LE may be used in buildings not provided with conduits, and where very reliable in- lation is required. Lead-armored cable is used where dampness is expected.

Metering. Charges for electric energy are based upon the kw-hr, which is total integrated amount of energy delivered in 1 hr to a circuit in which the rage power is 1 kw. Cost per kw-hr varies from 1¢ in large quantities, to and 15¢ for small quantities. The ampere-hr is not an accurate measure of rgy, unless the circuit voltage remains absolutely constant.

Instrument used for measuring electric energy is the watt-hr meter, sometimes called egrating wattmeter. Electric connections are similar to those for wattmeters

measuring power. MEASUREMENT OF TOTAL POWER OR ENERGY: (*a*) in d-c or s
phase a-c circuit, by one meter having its current coil in series with the load and its
tential coil in parallel with the load; (*b*) in a 3-wire d-c or in a 2-phase circuit, balanc
unbalanced, by 2 meters connected as if in 2 independent single-phase circuits
power being the sum of the 2 readings); (*c*) in a balanced 3-phase system, by 1 m
connected in 1 of the 3 receiving circuits (its reading being
tiplied by 3) or by 1 meter connected with its current coil in
the line wires and its potential coil connected from 1 line
the neutral (total power being obtained by multiplying the
ing by 3); (*d*) in a 3-phase system, balanced or unbalanc
2 meters, connected as in Fig 37. Total power is the
the 2 readings, if the power factor be greater than 0.5, in which case
difference of the readings if it be less than 0.5, in which case
potential leads of 1 meter must be reversed, to bring the
on the scale.

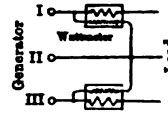


Fig 37. Power and Energy in Three-wire Distribution

Watt-hour meter comprises 3 parts: (*a*) a small electric motor, of commutator
cury and disk, or induction type, which runs at a speed proportional to the power;
brake or generator system, usually consisting of an aluminum disk revolving in the
permanent magnets to give a drag by eddy currents proportional to the speed;
of gears driving pointers for recording number of revolutions. While the type of
varies with the make and purpose of the meter, the brake and recording devices
mon to all. COMMUTATOR TYPE consists of a small d-c motor, the field being
the load current in current coil and the armature connected in potential coil
current proportional to potential. It is operative on both d-c and a-c circuits, but
only on d-c, the induction type being cheaper and better for a-c. MERCURY-MOTOR
hr meter contains a motor consisting of a disk of copper or aluminum floating in a
liquid mercury. Main current is lead to the mercury, but passes through the
cause of its lower resistance. This forms an elementary armature. Potential coils
an iron core; they form the motor field and produce a torque in the armature
volves it. Operative on both a-c and d-c circuits, but the induction type is prefer
a c. INDUCTION TYPE watt-hr meter is similar to a 2-phase induction motor, the
coils forming 1 phase, the potential coils the other. This gives a rotating magnet
which drives a metal disk rotor. Operates only on a c. Cheaper than either of the
types, and more reliable and accurate. POLYPHASE WATT-HR meter contains
coils and 2 potential coils, connected as 2 wattmeters would be connected; but
ments act on same shaft and drive same set of gears, thus mechanically taking
any positive and negative readings in either element, and giving correct total
one reading. Sometimes a single-phase watt-hr meter is used on a balanced 3-phase
cuit (such as one containing motors only). Its current coil is connected in one line
its potential coil is connected between one line and a neutral obtained by connecting
equal high resistances in Y. Total energy is 3 times the reading.

Meter capacity. In dwellings it should be 50% of the total capacity
connected load; in offices, 75%; for elevators, hoists, etc, 150% of the load.

Installation. Watt-hr meters must be installed on a rigid support free
vibration, and must be level. They should be at least 15 in between
and must not be near iron girders nor steam pipes, nor conductors carrying
currents. They must be protected from mechanical shock, weather, heat,
vermin, and dampness.

Sources of error. Adjustments are provided to take care of normal
loss. Variation in friction of brushes or bearings causes error. Mecha
shock, dirt, or dampness may alter the friction. External magnetic fields
variation in adjustment. In induction meters variations in frequency, volta
or load power factor may cause a slight error. Meters should be inspected
calibrated at least once a year. To CALIBRATE a watt-hr meter, connect it to
constant known load and count revolutions of the disk per min. Note the co
stant K painted on the disk, and substitute in formula: watts = $60 r p m \times K$.

15. Electric Lighting

ght distribution follows the law that the intensity is inversely as the square
e distance, providing the greatest dimension of the light source is less than
f the distance between source and object.

efinitions. INTENSITY of light is the relative amount of luminous energy
n by any source and is measured in candles or candle power. CANDLE POWER
is a measure of light intensity, determined by comparison with a well-
ned incandescent electric lamp, that has been accurately checked with a
dard lamp at the National Bureau of Standards, Washington. ILLUMINA-
is measured in ft-candles. One ft-candle is intensity of illumination on a
ace 1 ft distant from a source of one c p. At 2 ft distance the illumination
ld be 0.25 ft-candle. Light is the means; illumination, the end. INTRINSIC
LIANCY of a source is measured in c p per sq in. When excessive it is harm-
o the eye. Intrinsic brilliancy of tungsten lamp is 1 000 c p per sq in. A
source generally gives different intensities in different directions. Hence,
means nothing unless direction is specified. MEAN HORIZONTAL CANDLE
ER (m h c p) is the average c p of a lamp in all directions in a horizontal
e passing through center of the source, and is usually obtained by rotating
lamp about a vertical axis. MEAN SPHERICAL CANDLE POWER (m s c p) is the
age c p of a lamp in all directions, or the c p of a uniform source giving
same total flux of light. It is directly proportional to the total light given
he lamp, and is measured by taking intensity readings in all directions, or
lacing the lamp in a hollow dull white sphere and measuring average inten-
. REDUCTION FACTOR is an empirical factor for each type of lamp, by which
m h c p is multiplied to obtain the m s c p. Mean lower hemispherical c p
h c p) is the average c p in all directions below a horizontal plane through
source. LUMINOUS EFFICIENCY is expressed in watts consumed per c p
n, and the c p may be the m h c p (as in incandescent lamps) or more cor-
ly the m s c p. For arc lamps it is sometimes expressed in watts per m l h c p.
of light is measured in LUMENS, and is the quantity emanating from a source
nit intensity (one c p), and contained in unit solid angle. There are 4π or
6 lumens emanating from a source of unit intensity and 12.56 N lumens from
urce of N spherical c p.

n applying the laws of photometry to the calculation of a problem in illumina-
certain qualifications are necessary: Source of light must be small compared
he distance of the object illuminated; luminous surface must be at right angles
he line of direction from the object to be illuminated; reflection and refrac-
must be negligible. The most useful and general application of the law is
ressed by: $I_h = (c\,p + H^2) \cos^3 \alpha$; where I_h = intensity of illumination in ft-
dles at a given point p (Fig 38) on any horizontal plane;
= intensity of light source in given direction, expressed in
dles; H = height in ft of lamp above horizontal plane
taining point p; α = angle of incidence.

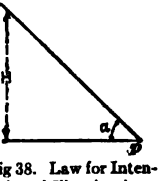


Fig 38. Law for Intensity of Illumination

eneral illumination of reasonable intensity (0.5 to 1.0 ft-
dles) facilitates a general view and perspective of the surround-
, while a greater intensity (1 to 5 ft-c) facilitates reading or
k. In DIFFUSED ILLUMINATION light comes from all directions,
ing no deep shadows; extreme diffusion gives flat lighting and
y be monotonous. To protect the eye some diffusion is usually
irable, for it tends to remove glaring reflection from objects in view; thus, when reading
n glazed paper or working on polished metals light must not be strongly directional.
DIRECT ILLUMINATION, shadows are dark, contrasts great, and forms are clearly outlined.
tails are best brought out by a light rich in the color corresponding to that of the object
wed. DIRECT LIGHTING unit sends most of its light at once to the object to be

SEMI-INDIRECT unit consists of a diffusing medium between the lamp and the object. It directs most of the light to the walls and ceiling, from which it is reflected. The INDIRECT unit directs all light from the lamp to the walls or ceiling, from which it is reflected. While more light is lost with the indirect system it is easier on the eyes. glare is avoided, the eyes are more sensitive, and vision is improved.

Arc lights are generally in large units (1 000 c p), are among the most efficient sources, but have high first cost and high maintenance cost, compared to incandescent lamps. They operate on either d-c or a-c circuits (but are not interchangeable), and may be of series or multiple type. The d-c lamp is usually a little more efficient and more reliable than the a-c. SERIES-TYPE ARCS are in series on one circuit, supplied by a generator or transformer designed to give a constant current irrespective of the lamps in operation. MULTIPLE-TYPE ARCS operate either singly or in pairs in series, on the usual constant-potential circuits with motors and incandescent lamps.

Arc lamps are also divided into the OPEN and the ENCLOSED type, the latter being preferable and are of the CARBON, FLAME, or MAGNETITE TYPE, depending upon kind of electrodes. The flame and magnetite are more efficient, the magnetite the most recent. Series arcs take from 60 to 80 volts and from 3 to 8 amperes, depending upon size; multiple lamps take 55 or 110 volts and 3 to 12 amperes. Carbon arcs have an efficiency of about 1 watt per m l h c p. Flame arc efficiency is about 0.4 to 0.7 watt per m l h c p. Magnetite arcs operate on d c, and on series circuit have an efficiency of 0.3 to 0.5 watts per m l h c p. They give off some obnoxious fumes. Arc lamps require periodic attention. MERCURY VAPOR lamp is used on d-c multiple circuits, 50 or 100 volts, efficiency 0.8 watts per m l h c p, and gives a blue-green light. No regular attention required.

Incandescent lamps are in many types, forms, sizes, voltages, and currents. There are 2 types now in general use, distinguished by the material of which the filament is made, but lamps of almost any c p and voltage may be obtained of each type. METALLIZED FILAMENT lamp (known as Gem lamp) contains a carbon filament especially treated in vacuum; it is the cheapest and most durable now on the market, the ordinary carbon filament lamp being obsolete. Gem lamp operates at efficiency of about 2.5 watts per m h c p. TUNGSTEN FILAMENT lamp (known as Mazda lamp) contains a tungsten filament giving yellow-white light. First cost is greater than the Gem, but its better efficiency usually saves enough energy to warrant the cost It is less affected by variation in voltage. Filament is more easily broken by mechanical shock or vibration than the metallized filament. Efficiency: 1.0 to 1.25 watts per m h c p. GAS-FILLED LAMPS (known in trade as high efficiency type C) have a tungsten filament coiled into a close spiral in a bulb filled with an inert gas at about atmos press. This retards distintegration of filament and permits operation at a higher temp and better efficiency (0.5 watts per m h c p). These lamps for multiple circuits are made only in large sizes, 200 watts and larger; series lamps in smaller sizes.

Rating of incandescent lamps. All multiple lamps are now rated in watts rather than c p. In speaking of the c p of incandescent lamps the m h c p is usually meant, from which the m s c p may be obtained by multiplying by a reduction factor given by makers. All incandescent lamps are very sensitive to variation in impressed voltage, respecting life, as well as efficiency and c p. Gem lamp is more sensitive than the tungsten. Common sizes of Gem lamps are 40, 50, and 60 watts, giving 16, 20, and 24 m h c p respectively. Common sizes of Mazda lamps are 25, 40, and 60 watts, giving 24, 39, and 60 c p respectively. Average useful life of a Gem lamp is 600 hr; of a Mazda, 1 200 hr. A lamp's useful life is said to be at an end when its c p for rated voltage has decreased to 80% of initial c p.

Costs. Enclosed carbon arc lamps cost $16 to $20; flame arc, $30 to $45. Magnetite arc, $23 to $27; mercury arc, with glass tubes, $33 to $35. Annual cost of OPERATING ARC LAMPS is from $30 to $70 per year, excluding fixed charges. Contracts are often made between municipalities and lighting companies for street lighting at $65 to $100 per year per arc lamp, everything supplied by

ng company. Gem incandescent lamps cost 15 to 25¢ retail; Mazda lamps, \$8. Cost of OPERATING INCANDESCENT LAMPS, including maintenance, st, and renewals, is summarized by A. G. Rakestraw as follows:

Cost of energy per kw-hr, cts	1	5	10
Cost of operation, cts per 1 000 s c p hr:			
25-watt tungsten	5.8	12	20
100-watt tungsten	4	10.5	18
Gem at 2.5 watt per c p	5.8	18	..

st per hr of supplying a given quantity of light comprises cost of lamp f energy. Following formula is convenient for obtaining minimum cost, mparing costs of different types of lamps:

$$\frac{Pn}{L}+\frac{c\,p\times w\,p\,c\times K}{1\,000}$$; C = total cost of light required, cts per hr; P = price of cts per lamp; n = number of lamps required for desired c p, obtained by dividal c p desired by the c p per lamp under consideration; L = average life of lamp, p = total c p desired; $w\,p\,c$ = watts per c p of lamps considered; K = price of cts per kw-hr.

rior Illumination. Interiors are generally illuminated by incandescent lamps on mulircuits, and to supply general illumination of even intensity a number of medium size (60 to 250 watts) are usually preferable to one large unit. Units should be placed or l so as to be out of line of sight of eye. Best method of distributing light is to divide into equal squares, having sides of 10 to 20 ft, depending upon degree of illumination l and height of ceiling, and placing a source of light at ceiling in center of each square t corners). Any local concentrated illumination may be provided by special local fix-

Diffusing globes should be used on all lamps in line of vision, and bowl frosted lamps ixtures unless completely shaded. Customary criterion for good interior lighting of ies by Mazda lamps is to supply 1 to 2 watts per sq ft of floor space; the higher value work is exacting or walls are of dark color. This is based on an illumination of from 7.0 ft-candles on a plane 30 in above floor, using tungsten lamps.

ne lighting. Tunnels, shaft stations, and all places of traffic, are usually d by 40 or 60-watt Mazda lamps, with weatherproof sockets and steel lain enamel reflectors. In gassy mines lamps are enclosed in heavy vapor-glass globes protected by iron guards. 200 to 300-volt circuits are used in , for which voltage lamps are available. Working places may be lighted rtable storage battery lamps, carried on the cap with separate battery or hand with battery combined (Sec 23, Art 10). To be approved by the u of Mines these lamps must be free from possibility of igniting gas, must light weight (not over 5 lb) and must not spill or leak electrolyte. They illuminate a 7-ft circle, give about 0.7 c p, and are usually for 2 volts au of Mines Schedule 6).

ble 11. Usual Intensities of Illumination in Installations Analogous to Those of Mining

Service	Intensity, ft-candles	Service	Intensity, ft-candles
k	4–5	Steel works:	
tory, general	0.8–1.5	Unloading yards	0.1–0.3
al bench for fine work	5–10	Open-hearth floors	0.1–0.3
al bench for coarse work	3–5	Blast furnaces	0.3–0.5
hine tools	2–5	Rolling mills	1–3
tern shop (wood)	3–5	Wire drawing	1–2
tern shop (metal)	4–6	Threading floors, pipe mills	1–2
ver house	2–3	Warehouse	0.5–1
		Wharf	0.1–0.2

Reflectors should be used with all lamps. Reflector for arc lamps for general lighting is an integral part of lamp, and designed to throw the light downward and outward. Reflectors for incandescent lamps may be extensive, intensive, or focussing; extensive throws the light downward, within a solid angle of about 90°, intensive in an angle of 60° and focussing in an angle of 30 to 40°. Industrial reflectors may be made of white enameled steel, aluminum, or painted metal; for interior use, plain glass or prismatic glass. Latter uses the principle of refraction to send downward the rays that would normally pass through the glass. Reflectors and lamps must be cleaned regularly, to prevent loss of efficiency. Tungsten lamp should be cleaned while the current is on, as there is less liability of breaking the filament.

16. Applications of Electric Transmission to Mine Service

(See Sec 16 for details of this subject.)

17. Electrochemistry

Equivalent weight or chemical equivalent of any substance is its molecular weight divided by its highest valency; thus, equivalent weight of copper in $CuSO_4$ is 31.78, being the atomic weight of copper 63.57 divided by its valency 2. Equivalent weight of the SO_4 radical is $[32.07 + (4 \times 16)] \div 2 = 48.03$. Equivalent weight gives the relative weights of various substances deposited in electrolytic work by a given current.

Gram molecule or **mol** is the number of grams of a substance which are equal to its molecular weight; thus, 1 gram molecule of $CuSO_4$ is 159.64 gm. GRAM EQUIVALENT is the number of grams of a substance which are equal to its equivalent weight; or to its gram molecule divided by its valency. The gram equivalent of $CuSO_4$ is $159.64 \div 2 = 79.82$.

Electrolyte. When an electric current passes through certain substances chemical action takes place at the points where the current enters and leaves the substance. Such substances are called electrolytes and the chemical action thus produced is called electrolysis.

Electrodes are the conductors by which the current enters and leaves the electrolyte. ANODE is the electrode by which the current enters; CATHODE that by which it leaves. The anode is connected to the positive terminal of the generator. Hydrogen and metal atoms of salts flow to cathode and are called CATHIONS; oxygen, and acid and basic radicals flow towards anode and are called ANIONS. All anions and cathions are called "ions."

Electrochemical equivalent OF AN ION is the mass in gm of the ion which would be deposited by one coulomb (Art 1) of electricity. ELECTROCHEMICAL CONSTANT or FARADAY (F) is the number of coulombs required to liberate one gram equivalent of any ion; value is 96 450 coulombs for all ions.

Faraday's Laws. Quantity of an electrolyte decomposed by an electric current is directly proportional to total quantity of electricity which passes. Rate of decomposition is directly proportional to current. Quantity decomposed is theoretically independent of voltage, current density, size of electrodes, and concentration of electrolyte. In practice local reactions cause deviation from this law. A given quantity of electricity always decomposes equivalent weights of different electrolytes in definite proportions for different elements. Total weights of different substances deposited by a given current in a given time are proportional to the equivalent weights of those substances. Some elements, as copper, may have different valencies in different combinations. Weight of such substances deposited is inversely proportional to the valency.

Electrochemical equivalent OF A SUBSTANCE is the number of gm of the substance deposited by 1 ampere flowing for 1 sec. The ampere is the unit electric current, which will deposit 0.00118 gm silver in 1 sec. For calculation

ts deposited by different currents in commercial practice it is more conven-
› use the value of grams deposited by 1 ampere in 1 hr (Table 12).
: electrochemical equivalent of any ion may be calculated by dividing
am equivalent of the ion by the Faraday. Thus, grams deposited in 1 sec
ampere equals the gram equivalent of the ion + 96 450. Weight of any
ınce in gm deposited in 1 hr is obtained by multiplying the values given in
12 by the current in amperes.

ible 12. Electrochemical Equivalents of Common Elements, in Grams Deposited per Ampere-hr

	Gm		Gm
ninum	0.3396	Lead	3.863
nine	2.981	Magnesium	0.4531
ium	0.7472	Nickel (val = 2)	1.095
rine	1.322	Oxygen	0.2984
per (val = 1)	2.371	Platinum (val = 2)	3.640
(val = 2)	1.186	Silver	4.025
l (val = 1)	7.353	Sodium	0.8576
(val = 3)	2.451	Tin	1.109
lrogen	0.0376	Zinc	1.219
(val = 2)	1.042		
(val = 3)	0.6946		

ecomposition or critical voltage is the minimum potential which must be essed to cause decomposition of a substance. With lesser potentials no n takes place; with greater potentials action is proportional to current. value depends upon chemical composition of the electrolyte, and is a func- of the chemical energy of the process.

e 13. Decomposition Voltages of Common Solutions used in Metallurgical Industries

	Volts		Volts
d, hydrochloric	1.31	Hydrate, potassium	1.67
nitric	1.69	" sodium	1.69
oxalic	0.95	Nitrate, lead	1.52
perchloric	1.65	" potassium	2.17
phosphoric	1.70	" silver	0.70
sulphuric	1.67	" sodium	2.15
loride, nickel	1.85	Sulphate, cadmium	2.03
loride, sodium	1.98	" nickel	2.09
		" zinc	2.35

alues given in Tables 12 and 13 are useful in checking efficiency of any elec- ytic action; for, with these constants, the theoretical value of grams separated kw-hr may be calculated and compared with the results obtained in practice. is, if A is the value of grams deposited per ampere-hr (Table 12), and B is the ical voltage for that substance (Table 13), then the theoretical grams per hr = $(A + B) \times 1\,000$. Example. Copper is deposited from a cupric solution he rate of 1.186 gm per ampere-hr (A), and requires 1.59 volts to maintain the on (B). Therefore the maximum possible recovery is $(1.186 \times 1\,000) + 1.59 =$ gm per kw-hr. Similarly, zinc is decomposed at rate of 1.219 gm per ampere- A), and requires a critical voltage of 2.35 (B). Hence, a primary battery con- es a minimum of $(1.219 \times 1\,000) + 235 = 520$ gm per kw-hr. The kw-hr re- red to deposit 1 kg of material are given by the formula: kw-hr = B

18. Batteries, Storage and Primary

Lead storage battery consists of 2 lead electrodes in dilute H_2SO_4. The positive plate has lead peroxide supported by a grid of pure lead; the negative is lead sponge similarly supported. The chemical reaction is:

$$\underset{\text{+plate}}{PbO_2} + 2\,H_2SO_4 + \underset{\text{−plate}}{Pb} = \underset{\text{+plate}}{PbSO_4} + 2\,H_2O + \underset{\text{−plate}}{PbSO_4}$$

(charged) (discharged)

Reading from left to right gives the action of discharging; from right to left gives charging. There are 2 types, the Planté and the Faure. PLANTÉ TYPE consists of plates of pure lead with surface increased by roughening by web ribs or grooves, and formed by charging electrolytically. There are 3 forms of construction of Planté plates: central web, cast lead, and pellet construction. In general, Planté cells are heavier and more bulky than Faure cells, but last longer. FAURE TYPE (paste type) has active material consisting of a paste containing litharge (PbO) or red lead (Pb_3O_4) applied to all grids and dried. Plates are then formed by being charged and discharged. Faure cells are lighter and therefore preferable for vehicles. Positive plates are those from which the current flows to the load during discharge. They have a dark chocolate color when charged, with hard smooth surface. Negative plates are gray, with soft surface. The ELECTROLYTE H_2SO_4, diluted with distilled or pure water, should have sp gr of about 1.15 when new, but different values are used for different services. It must be free from chlorine, nitrates, copper, iron, mercury, arsenic, acetic acid, and platinum, and should be made for the particular purpose. When cell is fully charged, the sp gr is 1.21 to 1.3; when discharged, about 1.15.

Capacity of a lead battery is stated in ampere-hr. Normal discharge current is that which the cell will give for 8 hr. Ampere-hr rating is this current multiplied by 8 for stationary work or by 5 for vehicle service.

Special ratings may be stated for a 1-hr or 4-hr discharge. Rate of charge and discharge varies from 6 to 10 amperes per sq ft of plate area, not including area obtained by corrugating. Any current may be obtained from a cell by putting enough plates in multiple. Watt-hr efficiency is the ratio of output to the input necessary to bring the battery back to its original condition. It is 90 to 92% for a battery floating on the line, and 75 to 80% for a battery completely discharged before recharging. A cell is discharged when its terminal voltage while discharging has decreased to 1.8 volts, and is charged when it requires 2.5 volts to force rated current through it. If a cell is discharged so that its voltage at normal current is less than 1.75, $PbSO_4$ may form, making it difficult to charge.

Care of lead storage batteries. All terminals and electric connections of a lead cell must be of lead; joints are formed by burning, or are lead covered. Instructions for care of batteries are given by makers, and should be followed carefully according to conditions and service. Electrolyte must be free from the injurious substances mentioned above, must be of definite sp gr for each condition, and must cover all plates. Cells must be cleaned before any deposit of sediment on bottom is high enough to touch the plates. Exterior of the cell must be cleaned and kept dry to prevent grounding and deterioration of terminals. Any leak must be repaired at once. Charge as soon as possible after cell has been discharged, at a current slightly in excess of the 8-hr discharge current, continuing until the negative plates give off gas steadily; then stop. In discharging stop when terminal voltage drops to 1.8 for normal 8-hr current and 1.6 volts for the 1-hr rate. Keep temp of the battery below 105° F. Cells to be put out of commission for some time should first be fully charged, electrolyte removed and replaced by pure water. Then discharge through a very low resistance, until practically discharged (avoiding heating of cells). Electrolyte is then removed, plates washed, dried, and put away.

ıon storage battery contains positive plates consisting of perforated steel tubes r nickeled and filled with alternate layers of nickel hydroxide and thin flakes of pure ic nickel. Tubes are supported by a grid of cold-rolled nickel steel. Negative consist of a nickel-steel grid, holding rectangular pockets filled with powdered iron

All plates are insulated from each other and from steel container. Electrolyte s of a 20% solution of potash in distilled water. Chemical reaction is:

$$\underset{+\text{plate}}{2\,NiO_2} + \underset{-\text{plate}}{2\,KOH} + Fe = \underset{+\text{plate}}{Ni_2O_3} + \underset{-\text{plate}}{2\,KOH} + FeO$$

d from left to right for discharge; from right to left for charge.

ting. Batteries are rated at the current which they will give for 5 consecu- ır. Ampere-hr rating is this current × 5 hr. Charging current is same as ırging, but time is about 7 hr instead of 5. Ampere-hr capacity is practi- independent of rate of discharge, as battery is not injured by reasonable ɔad and may be discharged to 0 volts. Watt-hr efficiency is 60 to 65%. ıge voltage per cell at normal rate of discharge (5 hr) is 1.2, varying from to 1.0 volt. During charging at normal rate voltage is 1.55 to 1.85.

ıeration. Steel containers must be insulated from each other, kept clean dry, and plates kept covered with electrolyte. Charge at 75 to 85° F; arge at 120 to 125° F. Sp gr of electrolyte should be constant at 1.2. Bat- are shipped with plates discharged, and after setting up should be charged eir normal rate for 12 hr. This overcharge should be repeated after the 30 and 60 days, and thereafter whenever electrolyte is renewed. Battery stand idle indefinitely if level of electrolyte is above the plates. It should ntilated, as the hydrogen gas given off is explosive. As in case of all stor- ıatteries the maker's instructions must be carefully followed.

eight and cost. LEAD STORAGE batteries rated on 8-hr basis cost about: ɔer kw-hr of capacity. Weight: 120 lb per kw-hr for portable style, 260 lb ıss jars, 370 lb in lead-lined tanks. Hence, they cost $300 to $400 per kw ɔity on 8-hr basis. NICKEL STORAGE batteries rated on 5-hr basis cost t $60, and weigh about 71 lb per kw-hr capacity. On this basis the cost w rating is about $300. The nickel-iron storage cell gives about 14 watt-hr ɔ of gross weight per charge; portable type of lead cell, about 10 watt-hr per lb.

imary batteries consume zinc or other metal by H_2SO_4, KOH, or similar electrolyte, ɔroduce electrical energy. Zinc is usual for negative pole, and carbon or copper for ive. Inside a primary battery current flows from zinc to carbon. Polarization ıs from new chemical substances (usually hydrogen) at the poles, causing a back . Depolarizers are chemical reagents used to combine with these substances and nt or reduce back e m f. There are 3 general types: wet, dry, and standard. WET s are being superceded by dry cells on account of their greater convenience. Typical ells are: Daniell, giving 1.07 to 1.14 volt; Gravity, 1 volt; Bunsen, 1.9 volt; Edison- ıde, 0.75 volt; LeClanche, 1.5 volt. The Daniell and Gravity are similar, and are for continuous small currents. LeClanche cell is good for intermittent service, as epolarizer takes time to operate. It is the commonest wet battery. DRY CELLS f LeClanche type, consisting of a zinc container which is also the negative pole. A ɔn rod forms the positive pole, the electrolyte being sal-ammoniac and zinc chloride in blotting paper, saw dust, or the like. Manganese peroxide is used as a depolarizer. age is 1.5 to 1.6. Internal resistance is 0.05 ohm new and 0.5 ohm after 10 months. short-circuit current through an ammeter of 0.01-ohm resistance should be from 18 to mperes. High temp increases the current. Potential decreases with age, due to dry- out. Cells are not good after 10 to 12 months, even if not used. Their ampere-hr ıcity is 24 to 30; higher on intermittent service, but capacity and life are very poor ɔntinuous service. Life of cell depends greatly upon relative external and internal ıtance, a matter requiring careful consideration. By putting 2 cells in parallel their bined life may be much more than twice the life of one alone on the circuit. Usual of cell is 6 in high by 2.5 in diam. Cost is 10 to 25¢ each, depending upon quantity quality purchased. (Standard Cell, see Art 1.)

19. Costs

D-c generators are usually of the compound-wound type, and comm 3 classes: high speed, to be belted to prime mover; moderate speed and l speed, to be direct-driven by an engine. The last class is least impor Table 14 gives costs and weights of the first 2 classes, for 125 or 250 volts.

Table 14. Weight, Cost, and Usual Speed of D-c Generators

Kw	High speed			Moderate speed		
	Speed, r p m	Wt, lb	Cost	Speed, r p m	Wt, lb	Cost
100	1 000	6 000	$ 900	625	10 000	$1 700
200	850	11 000	1 750	450	17 500	2 800
300	700	15 000	2 500	400	24 000	4 000
500	625	25 000	4 100	350	33 000	5 500

Machines to operate at lower speeds cost more. Add about 10% for install

D-c motors. Shunt motors are usually rated at the output they will continuously without injury to themselves. They are divided into classes according to their speed. Table 15 gives costs and weights of small high-speed and moderate-speed motors.

Table 15. Weight, Cost, and Usual Speed of Shunt Motors

H P	High speed			Moderate speed		
	Speed, r p m	Wt, lb	Cost	Speed, r p m	Wt, lb	Cost
0.5	2 200	60	$ 30	.....	.....	.....
1	2 000	100	50	1 675	125	$ 60
2	1 650	220	75	1 175	250	90
5	1 100	550	150	920	600	195
10	850	950	240	700	1 100	280
20	680	1 650	410	550	1 900	470
25	650	1 950	500	500	2 300	600

Cost of installation, exclusive of freight, is from 5 to 10% of first cost, depending upon local conditions.

Table 16. Weight, Cost, and Usual Speed of Series Motors

H P, nominal	Speed, r p m	Wt, lb	Cost
40	750	2 300	$440
50	700	2 650	525
75	650	3 300	640
100	625	3 700	700
150	600	5 000	950

Series motors are usually railway type, rated at output they will give for 1 hr without injury.

A-c generators and synchronous motors cost about the same for given output, speed, and voltage. Curves are given for costs usual types in Art 8. Add about 10% for installation.

Polyphase induction motors. Curves in Art 10 give costs and weights of usual sizes and types. Single-phase motors cost 25 to 30% more for given output and speed. Installation is simpler for most a-c motors, and cost can be taken at from 4 to 8% of first cost.

otor-generator sets, with 2 d-c machines, or 1 a-c and 1 d-c, cost about er kw rated output for a 200-kw set running at 750 rev per min; more for er sets. Flywheel and spe- controlled sets cost more be- of special features.

Table 17. Cost of Synchronous Converters

Kw	Speed, r p m	Wt, lb	Cost
100	750	13 500	$2 000
200	750	19 000	2 800
300	750	23 000	3 400
500	500	35 000	5 200

nverter costs are given in : 17 for a line of 25-cycle ines; 60-cycle at same speed little more expensive.

ansformers. Weights and of oil-cooled transformers for cles are given in Table 18. 25-cycle transformers cost 20 to 30% more. of installation is very small, as there are no moving parts.

18. Cost of Oil-cooled Transformers

:w	Voltage	Wt, lb	Cost
:0	2 200	450	$85
50	2 200	1 300	200
:0	2 200	1 900	275
00	13 000	2 800	450
00	13 000	5 000	700
00	13 000	6 500	900

Transmission lines for 3-phase, 33 000 volts, with No 2 B & S wire, wooden poles and porcelain pin insulators, may be erected for $1 800 to $2 000 per mile, if topography of district is not difficult

Power plants. Costs of building and equipping are: STEAM PLANTS, with reciprocating engines, vary according to size and character ant from $90 to $140 per rated kw. STEAM-TURBINE PLANTS, from $40 to er kw. HYDROELECTRIC PLANTS from $100 to $200 per kw, including dam other equipment. GAS ENGINE PLANTS, with gas producers, from $90 to per kw. In each case the lower figure applies to very large installations.

st of operation of power plants varies so widely with local conditions that detail is necessary for accurate estimates. A value of 0.5¢ per kw-hr uding fixed charges) is low, and is attainable only in large plants. 1¢ per r delivered to bus bars is common where load factor is reasonable, say 0.3 , and is a figure commonly used for preliminary estimates. LOAD FACTOR : ratio of average load for some period (as 24 hr) to maximum load during time. Duration of PEAK LOAD must be specified (as 1 min or 15 min); usually a short period during which the machinery may carry the overload ut injury. Total cost of a kw-hr, including fixed charges, is high when oad factor is low and vice versa.

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SECTION 43

ELEMENTS OF STRUCTURAL DESIGN

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Note. — Numbers in parentheses, except numbers of formulas, refer to Bibliography at end of this section.

1. General Principles

In structural design, one or all of 3 conditions may govern: safety, economy, esthetic effect. Safety involves knowledge of the stresses produced by the external forces acting, mechanical properties of materials of construction, and details of design best suited to the purpose.

'o compare two designs from standpoint of economy, the elements of cost are reduced common basis, using either capitalized or annual cost:

$$P = C + \frac{M}{r} + \frac{E}{(1+r)^n - 1} + \frac{R}{(1+r)^n - 1}$$

re P = capitalized cost, or the sum of money which would build and perpetually main- the structure; C = first cost; M = annual maintenance and operation; E = cost xtraordinary repairs every nth year; R = cost of renewal of structure in nth year; rate of interest. Annual cost = Pr.

line structures often have a useful life not greater than the life of the mine, and a sink- fund should also be included = $S + [(1+r)^x - 1]$, where x = useful life of structure ears. Esthetic principles are seldom considered except in important structures like ges or architectural work. Following case is also common. An addition to a plant be required n years hence and will cost A dollars. What expenditure can be afforded to avoid this future outlay? As much as the present worth of A dollars due in n s, or $p = A \div (1+r)^n$.

MECHANICS OF MATERIALS

2. Stress, Strain, and Elasticity

rain or deformation is the change in dimension produced on any material n external force; usually expressed as a ratio, inches per inch. Poisson's of lateral to direct strain is about 0.25 to 0.3 for metals. STRESS is the ance offered to deformation. For equilibrium it must be equal to the exter- orce, and may be regarded as resisting this force. It is expressed in units rce per unit area, lb per sq in or per sq ft. There are 3 kinds of strain and s: TENSION, COMPRESSION, and SHEAR. Bending and torsion are combina- of these, though sometimes classed separately.

odulus of elasticity or Young's Modulus = E = unit stress ÷ unit strain = stant (within elastic limit); or "stress is proportional to strain" (Hooke's law).

s generally considered the same in tension and compression for all materials in com- ise; a fundamental assumption in the theory of beams and columns. In shear or a = G = about 0.45 E. Hooke's law holds only for "elastic" materials (not plastic), nperfectly for C I, concrete, and some other materials. Thus, a steel bar 1 in diam in long, under tensile load of 23 000 lb (= 29 300 lb per sq in), taking E = 30 000 000, retch 29 300 ÷ 30 000 000 = 0.001 in per in, or a total of 0.012 in. If this law ntil strain = unity, the length of bar would be doubled; hence E is sometimes de- as force which would stretch a rod of unit cross-section to double its length.

stic limit is the limiting usually in lb per sq to which Hooke's law

t easily determined in test- "drop of the beam" of the machine, which occurs at ELD POINT, or commercial limit; higher than true limit (see Fig 1). Yield s not well defined in some als, as C I; it averages 0.5 the ultimate strength.

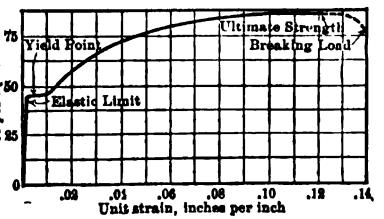


Fig 1. Stress and Strain for High-carbon Steel

mate Strength is the greatest stress, usually in lb per sq in, which a al develops before failure; sometimes greater than BREAKING LOAD. Some als in compression fail to give a true ultimate, but continue to compress crease in cross-section,

Resilience is a measure of DUCTILITY of material, and hence its ability to wit shock or suddenly applied loads. It is the work required to produce a given stres

Direct or static stresses are those acting always in one direction. REPEATED STR vary in intensity in one direction, but never pass through zero. ALTERNATING STR pass through zero, as, from tension to compression. Rupture may be caused not onl force exceeding the ultimate resistance of the material, but also by repeated ac forces of these two types, acting between certain limits which are less than the ulti known as the FATIGUE OF METALS.

Suddenly-applied loads, shock, and impact. A load is considered suddenly ap when its full amount acts instantly upon the material loaded. It produces strains stresses double those produced by a gradually-applied load. Shock is produced wh load moves freely before acting upon the material. The effect is that of a sudde applied load plus the effect of the motion. In practice suddenly-applied loads sel occur; shock is avoided if possible, but, in structures like RR bridges, an allowan effect of impact (a combination of sudden application and of shock due to irregulari roadbed, etc) is always made. It should also be allowed for in designing hoisting and links and pins for couplings.

Working stress is the max stress considered safe by the engineer and use him in designing. FACTOR OF SAFETY = ultimate strength divided by wo stress; it depends on: (*a*) uniformity, reliability, and durability of mate (*b*) kind of stress and character of loading, and certainty with which amo load is known; (*c*) character and life of structure.

For uniform materials like steel, 4 may be used; for variable materials like woo 10. Factor of safety is greater for alternating than for direct stress; less for tem structures than for permanent. In many cases (as for steel columns) safety factor s be computed on basis of elastic limit rather than ultimate strength.

3. External Bending Moments and Shears in Beams

Beams are of 4 kinds: (*a*) simple beams or horiz members, with end supp (*b*) cantilever beams, having but 1 support, or projecting from a suppo constrained beams, those having 2 supports and being rigidly fixed at both, or projecting over one or both, so that moments occur at the suppor continuous beams, with more than 2 points of support. Beams are subje bending and shear; struts or columns are subject to bending and compres

External bending moments and shears in (*a*) and (*b*) are easily determined as The derivation of formula for (*c*) is based on equation of the elastic or deflection while (*d*) involves special treatment, based on Theory of Three Moments. Table formulas for (*a*), (*b*), and (*c*); for (*d*), see *Trans* Am Soc C E, Vol 76, p 785.

End reactions must = total load, from law $\Sigma V = 0$. Also $\Sigma M = 0$; hence may be found by taking moments about either support.

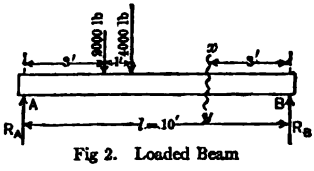


Fig 2. Loaded Beam

Example, Fig 2, M about A = (2 000 + (4 000 × 4) = R_B × 10, or R_B = 2 and R_A = 2 000 + 4 000 − 2 200 = 3 80

Bending moment at any sectio algebraic sum of moments of all for left of the section, calling forces t to cause clockwise rotation positive others negative.

In Fig 2, bending moment at section xy = (3 800 × 7) − (2 000 × 4) − (4 000 = + 6 600 ft lb, and also equals 2 200 × 3, or the moments to the right, if clockwise be called plus.

Vertical shear. At any section in a beam 2 equal and opposite forces ac acting, one on right and one on left, tending to shear the beam at that Vertical shear at any section is equal to algebraic sum of forces to the l

ection, upward forces being taken as positive, downward forces, negative. es to right of section may also be used, changing the sign.

Fig 2, vertical shear at $xy = 3\,800 - 2\,000 - 4\,000 = -1\,200$ lb. Fig 3 shows nts and shears at different sections of a beam.

aximum moments and shears. Max moments and shears determine the -section of a beam. It is generally economical to use same cross-section ughout, hence max moments shears need be uted. Point ax moment is l from princi- own in Fig 3; curs where ar passes gh zero. Max occurs at sup- .

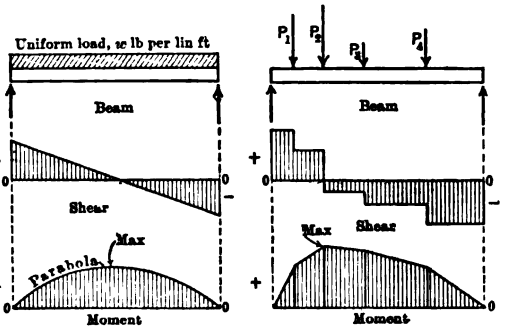


Fig 3. Graphic Representation of Moments and Shears in a Beam

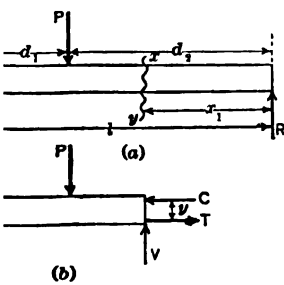
lesigning struc- for moving a series of con- ted loads is similar in spac- nd amount to rs or moving load to be used. These loads may be moved to occupy different posi- and the max moment is determined by methods in Art 19.

le 1 gives formulas for reactions, max bending moments, moments at any section ax deflection (for uniform cross-section), for common forms of beams and loading.

4. Flexure of Beams

beams, when subjected to loading, are stressed in tension, compression and . Vertical loading produces vertical shear, causing equal horiz shearing , which give to the horiz fibers of the beam their tension and compression es. Investigation of beams is based on the 3 laws of equilibrium (Sec 36, 2), the external forces being usually coplanar and at right angles to beam.

s, if beam in Fig 4a be cut at any section xy, the bending moment at $xy = P d_1 x_1$ nd shear at $xy = P d_1 \div l$ (Art 3). This moment must be resisted by the inner forces of beam acting on the right of section xy, or 2 forces C and T (both equal since ΣH must = 0) and acting with lever arm y. Assuming E equal in tension and compression, and that Hooke's law holds, and that plane sections of a beam before bending remain plane after bending (Navier's hypothesis), then it may be shown that C and T are the resultants of uniformly varying stress acting over the upper and lower sections of beam, having 0 intensity at a horiz plane passing through the center of gravity of section (known as the NEUTRAL AXIS), and max intensity at the extreme fibers of beam, directly proportional to distance of these fibers from neutral axis. The internal, or RESISTING MOMENT, is then $= kI \div d = kS$, where M is in inch-lb, k = extreme fiber stress, lb per sq in, I = it of inertia of the section, in^4, d = distance from neutral axis to extreme fibe $S = I \div d$ = SECTION MODULUS.



. Bending Moment in Simple Beam

Table 1. Reactions, Bending Moments, and Deflections of Beams

SIMPLE BEAMS. REACTIONS, BENDING MOMENTS AND MAX DEFLECTION (FOR UNIFORM CROSS-SEC)			
Beams and loading	**Reactions**	**Bending moments**	**Max defl'n $I=\infty$**
w = lb per linear ft	$R_A = R_B = \frac{1}{2} wl$	$M = \frac{1}{2} wx(l-x)$ $M_{Max} = \frac{wl^2}{8}$	$\frac{5wl^4}{384EI}$
	$R_A = R_B = \frac{1}{2} P$	$M = \frac{1}{2} Px$ $M_{Max} = \frac{1}{4} Pl$	$\frac{Pl^3}{48EI}$
	For AC $R_A = \frac{d_2}{l} P$ For BC $R_B = \frac{d_1}{l} P$	For AC $M = \frac{Pd_2}{l} x_1$ For BC $M = \frac{Pd_1}{l} x_2$ $M_{Max} = \frac{Pd_1d_2}{l}$	[illegible]
	For AC $R_A = R_B = P$ For CD $R_A = R_B = P$	For AC $M = Px_1$ For DB $M = Px_2$ For CD $M = Pd$ $M_{Max} = Pd$	$\frac{Pd}{24EI}(3l^2 - 4d^2)$
	$R_A = R_B = \frac{1}{2} W$	$M = Wx(\frac{1}{2} - \frac{x_1}{l^2})$ $M_{Max} = \frac{Wl}{8}$	$\frac{Wl^3}{60EI}$
	[illegible]	[illegible]	[illegible]

CANTILEVER AND RESTRAINED BEAMS. REACTIONS, BENDING MOMENTS AND MAX DEFLECTION (FOR UNIFORM CROSS-SEC)			
Beams and loading	**Reactions**	**Bending moments**	**Max defl'n $I=\infty$**
w = lb per linear ft	$R_A = wl$	$M = \frac{wx^2}{2}$ $M_{Max} = \frac{wl^2}{2}$	$\frac{wl^4}{8EI}$
	$R_A = P$	$M = Px$ $M_{Max} = Pl$	$\frac{Pl^3}{3EI}$
w = lb per linear ft	$R_A = R_B = \frac{wl}{2}$	$M = \frac{wl^2}{6}(\frac{1}{l} - \frac{x}{l} + \frac{x^2}{l^2})$ $M_{Max} = \frac{wl^2}{12}$ at supports	$\frac{wl^4}{384EI}$
	$R_A = R_B = \frac{P}{2}$	$M = \frac{Pl}{2}(\frac{x}{l} - \frac{1}{4})$ $M_{Max} = \frac{Pl}{8}$	$\frac{Pl^3}{192EI}$
w = lb per linear ft	$R_A = \frac{3}{8} wl$ $R_B = \frac{5}{8} wl$	$M = \frac{wx^2}{8}(\frac{3}{4} - \frac{x}{l})$ $M_{Max} = \frac{wl^2}{8}$ at B	$\frac{wl^4}{192EI}$
	[illegible]	For AD $M = [illegible]$ For DB $M = [illegible]$	[illegible]

Example. Assuming a T-section (Fig 5), subjected to external bending moment of 9 000 inch-lb, I is found = 2.81, the center of gravity being 2.09 in from bottom and 4.91 in from top, giving extreme fiber stress in compression, $k = dM \div I = 9 \times 2.09 \div 2.81 = 2\,900$ lb per sq in, while for tension side, $d = 2.09$ and $k = 6\,700$. For a symmetrical section, k is equal in tension and compression. In design, the greatest value of k must be used. For moments of inertia of various sections see Art 26 and Sec 56, Art [illegible]

Working value of k used in design is found by reducing by a proper factor of ty the k found by testing a beam to failure. Value at failure is the MODULUS RUPTURE, and is computed on assumption that above equation holds to failure tually true only to elastic limit), since M, I and d known.

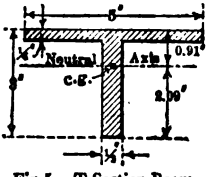


Fig 5. T-Section Beam

he work of actual design is facilitated by tables giving ies of S for different beam sections. Instead of S, the FF OF STRENGTH is often given. Thus, assuming M in , let C = coeff of strength = 8 M (= Wl for uniform ing or 2 Wl for concentrated, where W = total load, lb, l = span, ft), then C = ⅔ kS; and, knowing value of C ny beam, the safe load is easily computed, or vice versa. in ft-lb.

hear. In Fig 4*b*, a force V, equal to the vertical shear, must be applied at ion xy for equilibrium. $V \div$ the area of cross-section of the beam = v = the rage intensity of shearing stress, and the max intensity = 3 $v \div$ 2 or 3 $V \div$, and occurs at the neutral axis, where b = breadth and h = depth of beam.

sually, the limiting factors in design are axial tension and compression, it being unssary to increase size of member because of shear. In short or relatively deep beams, r may become the limiting factor, and in short timber beams especially, due to lamid structure, horiz shear often controls. Diagonal stresses are only important in reinɔd concrete beams (Art 14).

5. Columns and Struts

ompression members may be divided into 3 classes, depending upon mode of ire, and this in turn depends on the ratio of length of column to its least US OF GYRATION (Sec 36, Art 42), or in solid sections to its least diam, exsed as $l \div r$, where l = length, in, and r = least radius of gyration, in (Art 28).

hort columns, in which $l \div r$ is less than 10 or 15, generally fail by compresonly, and are sometimes termed short blocks. The safe working stress in pression is used in their design.

ort blocks are tested for determining strength of materials in compression; most rete and many timber columns fall in this class. They fail by compression or by ring in a plane making an angle of slightly over 45° with the axis of block. Shear re occurs when resistance to shear is less than 0.5 resistance to direct stress, as comion induces shearing stress on all oblique planes and max intensity (= 0.5 that of the t stress) on a 45° plane. Hence length of test specimens should always be stated, and rally will be 3 times breadth; many tests fail to agree, due to use of variable lengths.

rdinary columns, in which $l \div r$ ranges from 40 to 120, fail by combined pression and bending. Two kinds of formulas are used: (*a*) those derived theory, in which proper constants are inserted from tests; (*b*) empirical ulas, from tests only.

or column design, see Art 14, 21, 28. For eccentric loading, see Art 6.

e most widely used of the first kind is Gordon's or Rankine's formula (demonstrated by Tredgold): $P = cA + \left(1 + a\frac{l^2}{r^2}\right)$, where P = ultimate strength, lb; A = area, ; l and r as above, and c and a are constants depending upon kind of material, and l a depending on the condition of fixity of the ends of column. This formula may be for design by reducing P by a proper factor of safety.

eoretically, the strongest column has flat ends, a column with round or pin ends h is free to bend and not restrained at the ends being only 0.25, and a column with end round and one end free only about 0.5 as strong. Practically, pin-end columns ar to be as strong as flat-end, probably due to difficulty in getting an even distribut: ading and the resulting eccentric stress with the latter. Hence a should be d ɔd by tests of both flat and pin-end columns.

The principal formula of type (b) is the STRAIGHT-LINE FORMULA, so called because a graph is a straight line. This line is obtained by plotting, for a series of tests of columns of one type, the ultimate strength and $l \div r$ ratio, and drawing an aver line through the points. The general form is $p = A - B(l \div r)$, where $p = P \div A$ = intensity of stress, and A and B come from the tests. This formula is widely used, often displacing the Gordon. As before, p is properly reduced for design.

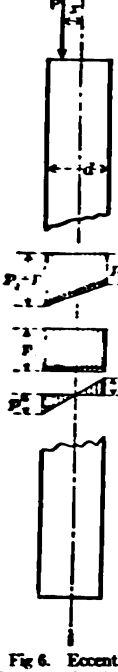


Fig 6. Eccentric Loading of Columns

Long columns, in which $l \div r$ exceeds say 200, are barred in structural practice and will not be discussed (2).

6. Torsion and Combined Stress

Torsion, a type of shearing stress, is fundamental in the design of shafts and other members subjected to a twisting moment. For circular section the formula is $d = 1.72 \sqrt[3]{M \div s}$, where d = diam, in, of a shaft capable of resisting a twisting moment M, in-lb, with a max torsive shear in the extreme fibers of s in-lb per sq in.

Example. Assume a working torsive shearing stress of 8 000 lb per sq in, in designing a circular shaft to carry a twisting moment produced by a force of 2 000 lb acting with a lever arm of 48 in. Then $M = 2\,000 \times 48 = 96\,000$ in-lb, and from the formula $d = 4$ in. nearly.

Combined stress is the condition of stress occurring when a member is simultaneously subjected to 2 of the 3 elementary forms of stress. A common case, compression and bending, occurs in the eccentric loading of columns and joints in foundations of masonry structures.

Eccentric loading of columns. Whenever possible in column design the load is applied to coincide with the axis of the column. Through use of brackets, some columns are loaded at a distance from the axis, as in Fig 6. This increases compression on left and decreases it on the right, and must be provided for by increasing the section until the maximum stress does not exceed the allowable. The unit compression due to direct stress $= p_0 = P \div A$. The unit bending stress caused by the moment $M = Pz$ will be $p' = Md \div 2I$, where I = moment of inertia of the section. Max unit stress $= p = p_0 + p'$, and must not exceed the safe value. Minimum stress $= p_0 - p'$. These formulas, though in common use, are only approx, as increase in stress due to deflection is not considered. (See Art 8 for foundations.) Combined stress also occurs in inclined bridge members, as eye bars and chords, which are subjected to bending due to their own wt as well as the direct stress of tension or compression. The common method of procedure is similar to that given above. Shafts also frequently act under the combined stress of torsion and bending (as in drum shafts of hoisting engines).

FOUNDATIONS

7. General Considerations and Bearing Power of Soils

Failure of foundations is generally caused by use of empirical rules instead of fundamental principles and poor judgment of designer.

Requisites of foundations: (a) They should be leveled off approx at right angles to the direction of loading. This insures that no component of the press will be tangential to the foundation surface, hence no tendency to sliding. Natural inequalities in rock surfaces usually hold the footing in place and make expensive stepping unnecessary. (b) When built on earth, design should be such that center of press and center of base coincide.

ining walls, it is too costly to adhere to this rule; low foundation pressures are required settlement is common. On rock little or no settlement can take place. (*c*) Each idation for separate parts of a structure supported at several points, as walls and in- r columns in a building, should be so designed, when on a compressible material, as to ubjected to pressure intensities that will cause uniform settlement.

afe bearing value of a soil depends on the character, depth, thickness and e of strata, degree of confinement, amount of moisture, type of structure and ribution of pressure.

rdinary soil is more compact as depth increases. Permanent foundations should be ed below frost level, and be well drained. Inclined clay strata frequently "crawls" r heavy loading, especially when moist. Some foundations have been entirely satis- ory on poor materials until nearby excavations have allowed the material to be forced laterally, when excessive settlement has resulted. Some settlement is generally to be cted except on solid rock, and in high structures, as chimneys, where unequal settle- t would be dangerous, very low unit press should be used. In foundations on rock, all portions of rock should be removed and filled with concrete.

earing power of soils. Table 2 is useful in forming an idea of safe unit sures, in absence of data from actual test.

Table 2. Safe Bearing Power of Soils (tons per sq ft)

Character of rock or soil	Authority								
	W H Burr		I O Baker		G P Gebhardt		W W Christie		G B Francis
	Min	Max	Min	Max	Min	Max	Min	Max	
k. Granite, etc, in ard, compact strata...	Any load which it is safe to put on a masonry footing		200	...	Solid bed rock up to 20% net crushing strength		Solid bed rock to 200 tons per sq ft		Ledge rock, 36
k. Limestone, equal best ashlar masonry.			25	30					
k. Sandstone, equal best brick masonry..			15	20					
k. Soft and pliable hale) = poor brick ma- nry.....			5	10					
k. Broken, but well mpacted.....	...	...	...	...	10	15	5 aver	25 aver*	
dpan. Gravel & sand, ell cemented with clay	...	...	8	10	....	.....		8.7	8
y. Thick beds and dry	4	6	6	8	3	4	4	5	3
y. Thick beds and oderately dry.....	2	4	4	6	2	.....	....	.....	..
y. Soft, wet, confined vel. Coarse & dry, ell compacted & con- ned.....	1	2	1	2	1	.....	1	.....	2
d. Dry, compact, well mented with clay....	6	9	...	...	6	8	8	5	5
d. Clear & dry, con- ed in natural beds....	...	...	4	6	1.5	.....	4	5	..
cksand. Marshy and luvial soils, silt, con- ned.....	2	4	2	4	2	.....	2	4.5	4
	...	...	0.5	1	0.5	.....	0.5	2.9	1

* E. L. Corthell

here data is entirely lacking, test pits and borings should be made, and if necess earing power of the soil tested by loading an area to point of settlement. Time rtant factor in such tests, and a safety factor of 3 to 6 should be used.

8. Ordinary Footings and Foundations

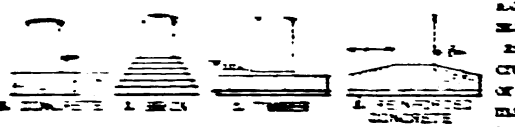


Fig 7. Footings for Masonry Walls

Table 3. Safe Bearing Pressures, Tons per Sq Ft. To get R multiply tabular value by thickness of footing in ft. Factor of safety = 10

Footing	Safe R, lb per sq in	Pressure, tons per sq ft			Footing	Safe R, lb per sq in	Pressure, tons per sq ft		
		0.5	1	2			0.5	1	2
Blue stone	300	4.1	2.9	2.0	Brickwork (Portland cement)	90	1.7	1.2	0.8
Granite	180	3.1	2.2	1.5	Concrete, 1–3–6	20	1.0	0.7	0.5
Limestone	150	2.8	2.0	1.4	Timber	1 000	7.5	5.3	3.7

Fig 7d shows a reinforced concrete footing that may be designed as in Art 14. Experiments indicate that the shear should not exceed the allowable at a distance d from face of wall; hence d is usually greater than required for bending, being fixed by d' which depends on shear. When the required offset l must be large to keep unit pressure low, large excavations of masonry and deep excavations are avoided by using a grillage consisting of 2 or more layers of timber or steel beams (18), or a reinforced footing with rods in both directions.

9. Pile Foundations

Bearing power of a soil may be increased by: (*a*) excavating to greater depth, the soil being then more compact, unless wet; (*b*) by drainage, usually by tile drain; (*c*) by wood or concrete piles.

Wood piles are used for both temporary and permanent foundations, and where the water level will last indefinitely. In temporary work, as trestles, the pile serves the purpose of both foundation and column. Oak, pine and spruce are in common use, depending upon difficulty in driving, length required, and purpose, oak being good for hard driving or where pile is subjected to abrasion. When the soil is very hard, piles are shod with iron shoes and the butt is chamfered off and banded with iron 0.5 to 1 in thick and 3 or 4 in wide. Special driving caps, with a tapered recess, are also effective. Wood piles should generally be at least 6 in diam at point (preferably 8 in) and about 12 in at butt (not over 18 in).

ncrete piles are cast-and-driven or cast-in-place. They are for permanent construc- where excavation to water level, required for wood piles, would be too costly.

fe load for a pile depends upon skin friction on sides of pile, and bearing power of soil r the point. Formula for driving wood piles: P (safe load, lb) $= 2\,Wh + (s + a)$, e W = wt of hammer, lb, h = fall of hammer, ft, s = aver penetration under last blows, in, and a = 1 for a drop and 0.1 for a steam hammer. It is applicable only in rm material, where S decreases at uniform rate. Sometimes S may be very large, after allowing the pile to stand for a day, many blows are necessary to move it at all. sual conditions require tests to determine safe loads. Max load per wooden pile is t 20 tons. Concrete piles are usually jet-driven, and above formula cannot be used. should be tested, and should carry loads of 25 to 80 tons each. Piles driven through to hard material act as columns and are computed as such.

acing of piles. They should not be closer than 2.5 ft centers. Earth is excavated 2 in below top of piles and space between them filled with concrete, which is continued depth of 12 in over top of pile.

st. Wood piles, 15 to 35¢ per ft in place; concrete piles aver 4 or 5 times as much.

MASONRY AND CONCRETE STRUCTURES

10. Lime, Cement, and Mortar (7)

mmon lime should be free from cinder and clinker, and unslaked lime should be in s with little powder or dust. Powder or dust usually indicates air-slaking, which is riment to perfect slaking later, although some fresh limes come in small pieces. Lime ld slake readily to a fine, smooth paste, without residue. Lime should be stored in a place. When slaked the paste may be kept indefinitely under water or protected air. If exposed to air in a thin layer lime may be slaked to a dry powder, sold as LATED LIME. HYDRAULIC LIME is not made in U S, and is rarely used for cement. mon lime costs about 50¢ per bbl in bulk, or 70¢ in bbls.

ment is of 3 kinds, Natural, Portland, and Pozzulan. NATURAL CEMENT is calcined nt rock, containing proper proportions of lime and clay, ground to a fine powder. It t now used for structures requiring great strength, having been superseded by Port- cement, which costs but little more and is stronger. It is made principally at Rosen- N Y, hence sometimes called Rosendale cement. PORTLAND CEMENT is made by ling to a fine powder the clinker from calcining to incipient fusion a powdered mixture operly proportioned lime and clay materials. Powdered limestone mixed with clay be used, also cement rock and clay or limestone, marl and clay, limestone and slag. uction in U S has increased rapidly since 1897, and over 100 million bbl are now pro- d annually. Cement is sold in bbl, paper or cloth bags or in bulk, the aver price at nills being 75 or 80¢ per bbl. There is an extra charge of about 7¢ per bag, which unded if bags are returned. Extra charge for bbls is about 20¢. Wt of Portland nt averages 100 lb per cu ft. A bbl weighs 376 lb; a bag, 94 lb. A bbl contains about u ft. Natural cement weighs 265–300 lb per bbl and 90–100 per bag. POZZULAN NT proper is made by grinding hydrated lime and pozzulan, a volcanic material occur- near Naples, and used by the ancient Romans. SLAG CEMENT is now the most impor- of this class. It is likely to contain much sulphur, and hence is not suited for use e air, though it is satisfactory where constantly exposed to moisture. Its manufac- is decreasing, due to use of slag in making a true Portland cement, with which it d not be confused, and which differs in no essential way from other Portland.

sting cement. Cement for small-scale work is usually bought "on brand"; that is, ke of cement that is known to have a good record for quality. It is well to specify the cement shall meet the specifications of the Amer Soc of C E, and on large works quire that all cement be tested. Complete testing requires considerable skill and ratus. Simple tests in the field may be made as follows: SOUNDNESS. Knead 1 cupful ment for 1.5 min with enough water to make a paste. Make into round pats about 3 im and 0.5 in thick, with thin edges. Make 3 of these on pieces of window glass [illegible] q. (*a*) Cover 1 pat with piece of wire netting and a wet cloth (moist air); (*b*) 1 at to action of steam for 5 hr; (*c*) put 3rd in water. Reject cement if any pat

radial cracks at edge (do not confuse with harmless shrinkage cracks), or curl or crum... (do not confuse curling with lifting of pat from glass due to expansion in setting, harmless). The air pat should not change color. FINENESS. Sift 5 oz through a 100-mesh screen; not over 0.5 oz should be retained. PURITY. Put as much cement as can be placed on a ... piece in a dish; add 0.5 teaspoonful of water and cover with clear HCl acid. Pure Portland cement will effervesce slightly, give off a pungent odor, and form bright yellow ... Powdered limestone used as adulterant will cause violent effervescence. Sand will remain undissolved in form of sediment.

Mortar. The sand should be coarse, clean, free from dust, soft particles, vegetable loam or other organic or deleterious matters.

Lime mortar. The lime should first be slaked, and the sand then spread evenly ... the paste and thoroughly mixed with it, a little water being added occasionally, until ... mortar is of uniform color. Sand reduces cost of mortar and makes it stronger ... grains being stronger than the lime), divides the lime into thin films which may more readily absorb CO_2 from the air, and prevents excessive shrinkage cracks due to water in the paste. Slaking should be done with care. The lime should be spread in 6 or ... layers in a box, or a basin in the sand, and 2.5 to 3 times its volume of water added at once. Cold water added later hinders slaking. The mass should be stirred, if at all, in such manner as to avoid chilling it. Lumps sometimes come to surface, due to formation of ... and are said to "burn." These should be broken up and pushed below surface. Slow slaking gives a volume of paste 3 or more times that of lime, while without care less than ... volumes may be obtained. Unslaked particles are not particularly harmful for masonry, but if cement is used with lime at least 2 days are required after slaking. For plastering work it should be stored 6 or 7 days. Proportions used are 1 volume paste to 3 sand, best obtained by trial. Thus, if mortar cannot be lifted up with the mixing hoe when handle is almost horiz, it contains too much sand and is "short"; if it will not slide from hoe, it contains too little sand. The proper proportions give a mortar that works easily with a trowel in laying up brick or stone. Lime mortar is unsuitable for thick walls or walls not in contact with air. 1 bbl of lime makes about 2.5 bbl or 0.3 yd stiff lime paste and 1 bbl of paste with 3 of sand about 3 bbl or 0.4 cu yd good lime mortar. 1 bbl ... slaked lime therefore makes about 0.95 yd of 1 to 3 mortar.

Cement is used neat in form of GROUT (in setting column bases, etc); in proportions 1 volume cement to 2 of sand as mortar for masonry work requiring great strength, and ... 3 for ordinary foundation work. It is not commonly used for brick walls ... to difficulty of handling it with ... trowel, but addition of 5 to ... hydrated lime makes the mortar "slick" and easy to work, though ... may cause efflorescence. Strength of cement mortar depends upon proportions and kind of sand used. A coarse sand, preferably silicious, should always be used, as fine sand kills ... cement and gives poor mortar. Sand consisting of ¼ coarse particles and ... medium fine is stated to give best results. Sharp, angular sand generally gives a stronger mortar than round sand, but has more voids and hence requires more cement for a ... strength. The denser the mixture, the greater the strength for a given volume of cement; thus the sand which produces the smallest volume of plastic mortar for the same quantity of ... materials is best. Small quantities of clay in the sand may increase the strength of a lean mixture, but decreases the strength of a rich mixture. Fig 8 shows aver strength of mixtures with different sands.

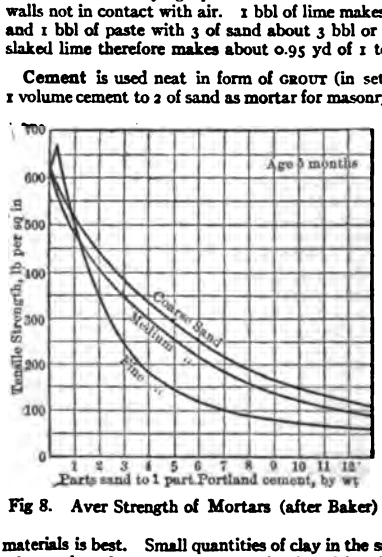


Fig 8. Aver Strength of Mortars (after Baker)

Table 4. Cement and Sand Required for 1 Cu Yd Mortar (37% voids)

Volume of mixture	Portland cement		Natural cement		Volume of mixture	Portland cement		Natural cement	
	Cement, bbl	Sand, cu yd	Cement, bbl	Sand, cu yd		Cement, bbl	Sand, cu yd	Cement, bbl	Sand, cu yd
1	7.59	0	8.05	0	1.4	1.49	1.02	1.47	1.00
1.1	4.00	0.68	4.11	0.70	1.5	1.20	1.03	.....	.....
1.2	2.62	0.91	2.59	0.91	1.6	1.01	1.04	.....	.....
1.3	1.93	1.00	1.79	0.97	.....	.....	.....	.....	.....

11. Stone Masonry (7)

Engineering requirements for building stone are cheapness, durability and strength. n general, building stone should be dense, hard and of uniform texture; it should contain o seams or "drys." A fresh fracture should be bright, sharp and clean, with no loose rains. Destructive agencies acting on stone may be frost, wind, fire, pressure and fric-on; for all of these except fire, granite is the most durable; for fire, sandstone is least ffected. Chemical composition and porosity are also important.

	Wt, lb per cu ft	Aver crushing strength of cubes, lb per sq in	Modulus of rupture, lb per sq in
Granite.....	167	19 400	1 850
Limestone.....	158	9 400	1 370
Marble.....	170	12 700	1 390
Sandstone	139	9 300	1 380

Stone for masonry comrises QUARRY-FACED, used as comes from quarry; PITCHACED, face edges are pitched o a right line; and CUT STONE, epending on finish of front r face of the work. Masonry ay be classified according o continuity of courses into: ANGE, BROKEN RANGE and ANDOM. It may also be divided into classes depending on thickness of joints, that is, ie degree of care with which the stone is dressed and joints made. Cut-stone masonry, ith joints less than 0.5 in thick, is known as ASHLAR; when stones are squared to give ll joints, it is SQUARED-STONE masonry and when composed of unsquared stone it is UBBLE.

For walls the largest stones should be put in the foundation, for strength, large bearing rface and to lessen danger of unequal settlement. The stone should be laid on its oadest face; stratified stone should be laid with its bedding planes normal to the press nd the courses should also be so laid; HEADERS should be placed at frequent intervals to ND the wall transversely, and stones in adjacent courses should break joint vertically o give longitudinal strength. Porous stone should be wet before being laid. All spaces etween stones should be filled with spalls and mortar. Hammering or sliding stones on wall under construction is not allowable.

Following points are usually covered in specifications for rubble masonry: stones ughly squared; laid in irregular courses in natural bedding plane; footings of selected at stones; at least 0.2 of face and back to be headers arranged to interlock; all voids lled with concrete or small stones fully bedded in mortar.

In building walls, a 4-in veneer of cut stone is sometimes tied into the backing (usually rick) by galvanized-iron wall ties or clamps. Riprap is loose stone, piled around piers, r on faces of earth dams, to prevent wash. Hand-laid riprap is a paving of stones, laid on dge. Dry rubble is laid without cementing material. Rubble concrete is concrete in hich large stone have been embedded.

Aver amount of mortar per cu yd required for 1 cu yd of rubble is: for small ugh stones, 0.33 to 0.40; for large stones, roughly dressed, 0.20 to 0.30 cu yd.

Safe loads for best rubble are 10 to 15 tons per sq ft; for squared stone, 15 to 20; for mestone ashlar, 20 to 25; for granite ashlar, 25 to 30 tons per sq ft.

Cost of stone masonary varies with type of masonry, thickness of wall, and location of ork. Rubble usually costs $2 to $6 per cu yd.

12. Brick and Brickwork (7)

Bricks are made from mixtures of lime and sand or cement and sand.

Varieties of clay brick. COMMON BRICK is used for ordinary construction. Best brick, when broken in two, shows a fine, uniform texture, deep salmon color, contains no fissures, air bubbles, pebbles nor lumps of lime. It should also have plane faces and parallel edges (no warping), and give a clear ringing sound when struck (thorough burning). Usual size is 8.25 × 4 × 2.25 in. Overburned brick is dark in color and liable to be warped and brittle. Light or yellow color indicates soft or underburned brick. PRESSED BRICK is brick molded under heavy press. It is uniform in shape and of greater density, absorbing only 10 to 20% of its wt of water, while common brick may absorb 30 or 35%. It is used for exterior faces of walls, or where neat finish is desired. ENAMELED BRICK is made from clay containing a large amount of fire clay, on which a surface finish of enamel is placed either before or after burning. GLAZED BRICK is produced by applying a slip to the unburned brick, which adheres to clay and holds a glaze composed of readily fusible materials giving a vitreous surface. FIREBRICK is of fire clay burned at a high temp. Good fireclay is free from pyrites, lime, and other ingredients which would cause swelling or vitrification. Fireclay brick should be of uniform color, white or white speckled with brown throughout; should be laid in thin paste of fireclay, with joints as thin as possible. SAND-LIME BRICK is generally pure white. Made by molding 4 to 6% of finely-powdered lime with silica sand. The brick is then put in a tight cylinder, and heated by steam for about 10 hr, causing formation of silicate of lime, which gives hardness. Generally not as strong as clay brick, but strong enough for ordinary purposes. CEMENT BRICK is of sand, with Portland cement as the binding material.

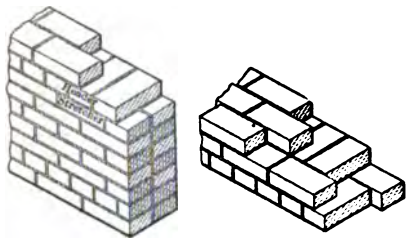


Fig 9. Bonds for Brickwork

Bond results from laying brick in such manner as to tie the wall together longitudinally and transversely. Fig 9 shows the common bond, consisting of 4 to 7 courses of STRETCHERS to 1 course of HEADERS, the latter tying the wall together transversely. For equal strength in both directions use 2 stretcher courses to 1 header course. In laying brick vertical joints in header courses must be made of such thickness that they will not come over a joint in the stretcher course. Brick should be thoroughly wet before laying; otherwise they will absorb moisture from the mortar, and prevent proper setting. Joints should be ¼ to ⅜ in thick, and for best work brick should be placed on a full mortar bed and shoved into place ("shove joints"). All vertical joints should be thoroughly "slushed" with semi-liquid mortar, so the brick will be well embedded. Face of the wall may be finished with either flush, struck or weather joints (Fig 10). For outside walls, exposed to weather, finish (c) is most durable, but more difficult to make than struck joint.

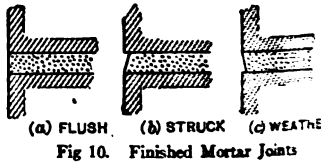


Fig 10. Finished Mortar Joints

Number of brick required depends on size. For common masonry 1 cu ft generally requires 20 brick; 1 cu yd, 500 to 600 brick. A 4-in wall (1 brick thick) requires 7; 8-in (2 bricks thick), 14; 12-in (3 bricks thick), 21 brick per sq ft of wall. An "8-in wall" actually measures 8.25 to 9.5 in, and a "12-in wall" 12.5 or 13 in.

Quantity of mortar. For aver joints (¼ to ⅜-in) 1 cu yd common brickwork requires ¼ to ½ cu yd mortar, or about 0.5 cu yd per thousand brick. For 1 to 3 Portland cement mortar, this requires about 1 bbl cement and 0.5 cu yd sand per thousand.

Working stresses for compression are: for common brick in Portland cement mortar, 175 lb per sq in; in natural cement mortar, 150; in lime and cement mortar, 125 in lime

ortar, 100 lb per sq in. **Brick piers** may be designed by the formula, $p = P - 6H + D$, here p = allowable unit stress, lb per sq in; P = allowable direct compressive stress, per sq in; H = height, in; D = least thickness of pier, in.

Costs. Common brick are sold by the thousand (M), aver price being \$6 to \$7 per M. ne bricklayer will lay 1 500 to 2 000 brick per 10-hr day, in straight thick walls, but only 0 to 500 for work involving many corners or angles. With bricklayers at 60¢ per hr d labor at 17.5¢, labor cost is \$7 to \$9 per M.

13. Concrete (7)

Composition. Concrete is a mixture of broken stone, gravel, cinders, or slag, lled the COARSE AGGREGATE, and sand or stone screenings, known as the FINE GREGATE, with a cementing material, as cement or asphalt. For reinforced important work, Portland cement is used; for unimportant work or where wt the essential element (as anchorages for suspension bridges), natural cement; phalt is used for sidewalks, floor finish and pavements.

Strength of Portland cement concrete, for a given aggregate, increases with increase in ount of cement. Concrete takes say 6 months to attain its full strength. Quantity of ter used in mixing influences the strength, and the rapidity with which full strength is ched. Dry mixtures mature quicker than medium or wet. A medium mixture ulti- tely attains about the same strength as a dry one, but, where strength is important, ry wet mixtures should be avoided as the strength is permanently affected. Strength greatly influenced also by size and grading of the aggregate (Fig 11). Strongest con-

(a)	(b)	(c)	(d)	(e)
Solid Stone Voids = 0% Density=1.00	Sand Voids = 45% Density=0.55	One size stone and sand Voids = 40% Density=0.60	Two sizes of stone and sand Voids = 35% Density=0.65	Graded stone and sand Voids = 30% Density=0.70

Fig 11. Grading of Aggregate in Concrete

te is made by using the densest aggregate, with just enough cement and water to fill the ids. Hence, it is desirable to use: 1, a graded aggregate; 2, as large size stone as prac- able; 3, no more mortar than needed to fill all voids and prevent "honey combing"; 4, more water than necessary. For massive work 3-in maximum size stone is desirable; foundations, 2.5 in; for arch rings, 1.5; for coping, bridge seats and thin walls, 1; for nforced concrete beams or slabs, 0.75 in. Fine aggregate is a relative term, applied to terial less than 0.25-in size. For sand the same rules apply as for mortar (Art 10). The irse aggregate should be graded in size, clean, hard, durable, and contain no soft, flat elongated particles. Broken stone makes slightly stronger concrete than gravel, but as dense, and is an exception to the density rule given above.

Standard proportions for concrete (1 : 2 : 4 indicates 1 volume cement, 2 fine, and 3 vol coarse aggregate). For small work proportions may be selected follows: RICH MIXTURE, 1 : 1.5 : 3; for columns or other parts subjected to h stresses or requiring to be completely watertight. STANDARD MIXTURE, 2 : 4; for reinforced floors, beams and columns, arches, reinforced engine or chine foundations subject to vibrations, tanks, sewers, conduits or other tertight work. MEDIUM MIXTURE, 1 : 3.5 : 5; for ordinary machine founda- ns, retaining walls, abutments, piers, thin foundation walls, building walls, linary floors, sidewalks and sewers with heavy walls. LEAN MIXTURE, 1 : 3 : 6; unimportant work in masses, for heavy walls, for large foundations support- a stationary load, and for backing for stone masonry.

On larger or important work the proportions are determined by measuring the voids, ng 10% excess of cr uired to fill the same voids, and the same amount

of mortar in excess of that required to fill the voids in the stone; or better, by trying various mixtures to determine how much cement and sand will give the smallest volume (greatest density) for a given volume of stone. This will be the strongest possible concrete from the material used.

Table 5. Working Stresses for Concrete

Based on compressive strengths, etc, of Joint Committee of Eng Societies; to be used in absence of better data

Portland cement concrete			Columns			Bending		Diagonal tension and shear				Bond	
Aggregate	Mixture	Ult compressive strength, lb per sq in	(a) Plain, not over 12d in l'gth, or with 1–4% long steel and not over 15d in length, 22.5%	(b) With hooping only when length not over 8X diam of core 20% greater than (a)	(c) 1–4% long steel with hoops if length not over 8X diam of core 45% greater than (d)	Compression in extreme fiber or bearing, 32.5%	Ditto adjacent to support in continuous beams 15% greater	(a) Horiz bars only — no web reinforcement, 2%	(b) Thoro web reinf'c'mt. Bent up bars and stirrups, 6%	(c) Bent up bars only. No stirrups, 3%	(d) Punching shear, 6%	For bars, 4%	For drawn wire, [illegible]
Granite or trap rock	1:1:2	3300	742	891	1077	1071	1232	66	198	99	198	132	[illegible]
	1:1.5:3	2800	630	756	914	910	1047	56	168	84	168	112	[illegible]
	1:2:4	2200	495	594	718	715	822	44	132	66	132	88	[illegible]
	1:2.5:5	1800	405	486	587	585	673	36	108	54	108	72	[illegible]
	1:3:6	1400	315	378	456	455	523	28	84	42	84	56	[illegible]
Gravel or hard limestone or hard sandstone	1:1:2	3000	675	810	979	975	1120	60	180	90	180	120	[illegible]
	1:1.5:3	2500	562	675	815	813	935	50	150	75	150	100	[illegible]
	1:2:4	2000	450	540	653	650	747	40	120	60	120	80	[illegible]
	1:2.5:5	1600	360	432	521	520	598	32	96	48	96	64	[illegible]
	1:3:6	1300	292	350	423	422	486	26	78	39	78	52	[illegible]
Soft limestone or sandstone	1:1:2	2200	495	594	718	715	822	44	132	66	132	88	[illegible]
	1:1.5:3	1800	405	486	587	585	673	36	108	54	108	72	[illegible]
	1:2:4	1500	338	406	490	487	561	30	90	45	90	60	[illegible]
	1:2.5:5	1200	270	324	392	390	448	24	72	36	72	48	[illegible]
	1:3:6	1000	225	270	326	325	374	20	60	30	60	40	[illegible]
Cinders	1:1:2	800	180	216	261	260	299	16	48	24	48	32	[illegible]
	1:1.5:3	700	157	189	228	227	261	14	42	21	42	28	[illegible]
	1:2:4	600	135	162	196	195	224	12	36	18	36	24	[illegible]
	1:2.5:5	500	112	135	163	162	187	10	30	15	30	20	[illegible]
	1:3:6	400	90	108	130	130	149	8	24	12	24	16	[illegible]

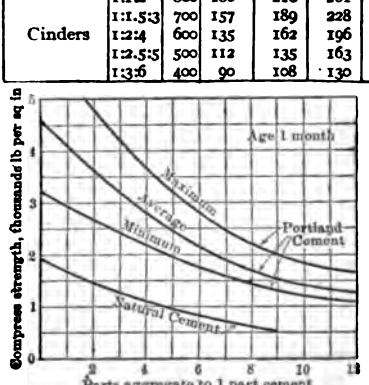


Fig 12. Compressive Strength of Concretes

Tensile strength of concrete, which is low and subject to great variation with slight changes in details of making, is generally neglected in design. For a 1 : 2 : 4 mix, it varies from 300 to 300 lb per sq in. for 1 : 3 : 6, from 100 to 175. Shearing strength is difficult to determine, but it is probably about 0.5 the compressive strength. Modulus of elasticity is also variable, the material not usually showing true elastic behavior. For working loads, it varies from 2 500 000 to 3 500 000 lb per sq in, depending on mixture and age.

Quantities of materials required for 1 cu yd concrete with various

centages of voids may be found by Fuller's rule for the approx amounts of ient, sand and broken stone in 1 cu yd of aver broken-stone concrete: 11 ided by sum of the parts equals number of bbls of cement. Multiply by the number of parts of sand and by the cu ft in 1 bbl cement (3.8 aver) divide by 27. This equals the cu yd of sand required. Similarly for the ken stone.

xample. 1 : 3 : 5 cement concrete. $11 \div 9 = 1.22$ bbl cement per cu yd. $1.22 \times 3 \div 27 = 0.51$ cu yd sand and $1.22 \times 3.8 \times 5 \div 27 = 0.84$ cu yd broken stone.

lixing and depositing. Cement is measured by counting the bags. Sand l stone are best measured by a bottomless box, the size of which is based on ıg an even number of bags per batch.

. 2-bag batch (about 0.5 cu yd) is convenient for hand work and 1 cu yd should be the t. In hand work, the sand and cement should be well mixed dry, spread in a layer the stone placed on top. Water is then put the stone, and the whole turned at least 3 es. Machine mixing gives better results, if amount of water is properly controlled. If , the concrete must be well rammed in layers ut 6 in thick, but in thin walls a wet mixture sed, simply requiring churning with a spade or k to loosen air bubbles and work it into all corners of the forms. Concrete should be tected from too rapid drying, and kept moist il thoroughly set. It may be placed under ter by means of metal pipes, the concrete being in at the upper end as fast as it goes out at the er end, thus preventing it from falling through water, which would wash out the cement. In ning old and new surfaces, care should be taken remove all dirt and laitance (scum which forms surface), and thoroughly to wet the surface of old work. In long walls, expansion joints uld be provided every 30 to 50 ft. Fig 13 shows common forms of joints. Concrete not injuriously affected by oils and acids, unless strong enough seriously to affect other materials. Salt or alkali water causes disintegration at the water line, due to unknown action, and sand containing alkali should not be used.

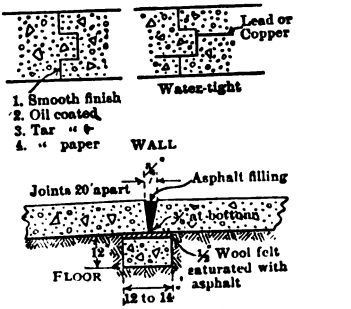


Fig 13. Concrete Joints

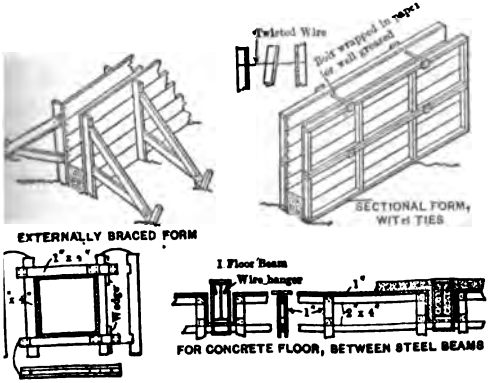


Fig 14. Forms for Concrete

Rates of work and costs. One man will mix, wheel 50 ft, level and ram, about 2.2 cu yd of concrete per 10-hr day. A gang of 15 men will handle as above an aver of about 33 cu yd, and 1 man will level and ram in 6-in layers about 11 cu yd per 10-hr day. Cost aries from $6 to $10 or more per yard, depending on labor and material costs, s well as the kind of work.

Forms for concrete are made of partly seasoned pine or spruce, planed on one side to give smooth surface and even thickness) and for best work tongued and grooved. The lagging is generally 1 in thick, but for heavy work and where the forms are to be used several times 2 in is better. Studding is commonly of 2 by 4 in to 4 by 6 in, spaced 1.5 to 2 ft centers for 1-in lagging, and 4 to 5 ft for 2-in. Pressure due to concrete is about that of a liquid weighing 80 to 140 lb per cu ft, depending upon the amount of water used in mixing and rate of placing. Since forms cost 0.25 to 0.5 of the entire cost of the work, a concrete structure should be designed with a view to simplicity in the form work, a little extra concrete often being economical. Forms should be carefully designed and constructed, so as to be easily removed without injury to the structure or the forms themselves; this is done by making them in units, joined by cleats, the pressure from the concrete being taken up by transverse tie wires, which are left in the concrete, or by external braces and struts. Adhesion of concrete to the forms can be prevented by applying crude oil, soap or grease. Fig 14 shows typical forms. The time which must elapse before forms can safely be removed depends on weather conditions and how soon the structure will receive its load. Time under ordinary conditions: Walls in mass work, 1 to 3 days or until concrete will bear press of thumb without indenting; thin walls and columns, summer 2 days, cold weather 5 days, provided they carry no load; slabs, up to 7 ft span, beams and girders, summer 6 days, cold weather 2 weeks; long-span slabs and girders, summer 10 days, cold weather 3 weeks to 1 month; arches, small 1 week, large 1 month.

Aver cost of form work is 8 to 12¢ per sq ft of surface.

14. Reinforced Concrete (23)

Concrete, reinforced by steel bars, is not yet fully standardized. It is economical for permanent structures, not because of its strength (which averages less than 0.5 that of timber) or low first cost, but because of its great durability, requiring little or no maintenance charges. Reinforced concrete structures can be built for the most part by unskilled labor. Details of design should therefore be simple, completely worked out and detailed on the plans, and construction must be inspected and supervised by competent engineers. Failures have often occurred through neglect of these requirements.

In general, the steel in reinforced work takes the tensile stresses, the concrete takes the compression. Relative economy of the two materials can be shown as follows. Steel costs 60 to 70 times as much as concrete per unit volume. In tension, steel takes a working stress of 16 000 lb per sq in, while concrete is limited to 30 or 40 lb. Hence, for tension, steel is about 500 times as strong per unit, or for same cost about 7 times. In compression concrete takes say 600 lb per sq in, and steel is only 25 times as strong; hence for same cost, concrete is about 2.5 times as strong as steel.

Reinforcing bars are usually 0.25 to 1.5 in. The 0.75-in bar is known as BASE, there being no extra charge for this size or larger, in either rounds or squares, but smaller bars cost more per lb. Odd sixteenths are seldom used. Square bars are more used than rounds, due to the fact that DEFORMED BARS are generally desirable and most of these are square.

Beams. Formulas are based on Navier's hypothesis (23) and on perfect adhesion between the concrete and steel.

Working formulas only are given herewith, and a constant coeff of elasticity is assumed; also the tensile strength of the concrete is neglected, as the concrete in the lower side of a beam develops fine cracks at about the same stress that would cause failure in a plain beam (corresponding to about 3 000 lb per sq in in the steel), and, the steel being computed for 14 000 to 20 000 lb per sq in, the concrete is thus of no tensile value.

For beams under loadings of less than 3 000 lb per sq in in the steel, a straight-line variation is assumed, and the tensile strength of the concrete should be considered. For beams near the ultimate, parabolic variation appears to suit conditions best and the tensile strength of the concrete is neglected. The span of a concrete beam is taken as the distance center to center of bearings, but need not be greater than clear span plus depth of beam.

tandard notation. For convenience the complete notation is given herewith.

Rectangular beams and slabs

= unit tensile stress in steel, lb per sq in
= " compressive stress in concrete, lb per sq in
= modulus of elasticity of steel
= " " " " concrete
$= E_s \div E_c$
= area of steel, sq in
= effective depth of beam, top of beam to cent of grav of tensile steel
= breadth of beam, in
= thickness of concrete below tension steel for protection
= elongation of steel due to stress f_s
= compression of concrete due to stress f_c
= depth of neutral axis below top of beam, in
= ratio of x to d, or $x = kd$
= ratio of lever arm resisting couple to d
= steel ratio $= A_s \div bd$ = % of steel ÷ 100
M_c = resisting moment of concrete
M_s = " " " steel

T-Beams

b = width of flange, in
b' = " " stem, in
t = thickness of flange, in

Shear and bond

V = total shear, lb
v_s = unit aver shearing stress, lb per sq in
u = bond stress per unit surface area of bar, lb per sq in
o = circumference of perimeter of bar, in
Σo = sum of perimeters of all horiz bars
a_s = area of steel in stirrup, sq in
l' = grip or required length of bar or stirrup, in
i = diam of stirrup or bar, in

ctangular beam and slab formulas (Fig 15). From Navier's hypothesis and ke's law (Art 2), $f_s = nf_c(1 - k) \div k$ . . . . (1), and resisting moment of beam based orking strength of the steel $= M_s = f_s pbd\,(d - x \div 3) = C_s bd^2$ . . . . (2), where $f_s p\,(1 - k \div 3)$.

ring similarly on concrete, $M_c = f_c(1 - x \div 3)\,x \div 2$ bd^2 . . . . (3), e $C_c = f_c(1 - k \div 3)\,k \div 2$. If resistoment of a beam d from Eq 2 is ter than that Eq 3, the beam, ted to failure, will

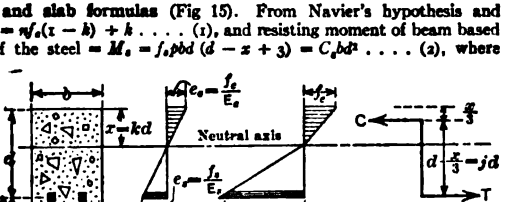


Fig 15. Stresses in Concrete Beams

y crushing of the concrete, and the beam is said to be OVER-REINFORCED. If conditions eversed, it fails by tension in the steel, and is UNDER-REINFORCED. If a beam has equal gth, figuring from either steel or concrete, it is a BALANCED BEAM. Thus, if n, f_s and f_c ixed from Eq 1, k is fixed, and $k = \sqrt{np\,(np + 2)} - np$ . . . . (4); hence, with k , p is fixed, and $C_s = C_c$, as would be required. M_s is then equal to M_c, and must = xternal moment M; hence, from Eq 2 or 3, $d = \sqrt{M \div Cb}$ . . . . (5), where C = e of C_s or C_c, when $C_s = C_c$. Value of d from Eq 5 is the BALANCED DEPTH, as it is the depth that will simultaneously insure full working values of f_c and f_s in a beam with d b fixed.

us, assuming $f_s = 650$, and $f_c = 16\,000$ lb per sq in and $n = 15$; from Eq 1, solving , $k = 1 \div [1 + (f_s \div nf_c)]$ . . . . (6); whence, $k = 0.378$. From Eq 4, which may ritten $p = k^2 \div [2n\,(1 - k)]$ . . . . (7), $p = 0.0077$, and from Eq 2 or 3, $C_s = C_c$ = 107.4. Table 6 gives values for other stresses.

or economy b should be small, since for equal strength, a deep narrow beam a smaller area than a wide shallow one. For proper design this can not be , and it is generally considered that b should not be less than 0.04 of the span e beam is not supported laterally, and that ordinarily b should be between nd 0.75 d. Using proper proportions for b and d, a beam of balanced depth adily designed, and closely approximates the most economical design.

the following examples, if the steel costs 70 times as much as the concrete, the be- cost in the ratio 166 : 208 : 172, the balanced-depth beam being cheapest.

Table 6. Constants for Balanced Beams

(Values of p, k, j, C and S for beams of balanced depth)

f_s	f_c	For $n = 12$			C		S, ft-lb
		k	j	p	in-lb	ft-lb	
14 000	500	0.300	0.900	0.0054	67.5	5.62	1 05
	550	0.320	0.893	0.0063	78 6	6 55	1 24
	600	0.340	0.888	0.0073	90.6	7 55	1 [illegible]
	650	0.358	0.881	0.0083	102.5	8 55	1 [illegible]
16 000	500	0.273	0.909	0.0043	62 0	5.17	1 21
	550	0.292	0.903	0.0050	72.2	6.02	1 2[illegible]
	600	0.310	0.897	0.0058	83.2	6.93	1 [illegible]
	650	0.328	0.891	0.0067	95.0	7 92	1 [illegible]
f_s	f_c	For $n = 15$			C		S, ft-lb
		k	j	p	in-lb	ft-lb	
14 000	500	0.348	0.884	0.0062	76 7	6 40	1 09
	550	0.372	0.876	0.0073	89.5	7 46	1 [illegible]
	600	0.391	0.870	0.0084	102.0	8.50	1 [illegible]
	650	0.410	0.863	0.0095	114.8	9.50	1 [illegible]
16 000	500	0.319	0.894	0.0050	71 3	5 94	1 [illegible]
	550	0.339	0.887	0.0058	82 3	6 85	1 [illegible]
	600	0.358	0.881	0.0067	94 4	7.86	1 [illegible]
	650	0.378	0.874	0.0077	107.4	8.95	1 [illegible]

In getting A_s, two formulas are used. By definition $A_s = pbd$ . . . . (8), also, from F[illegible] $A_s = M + (f_s jd)$ . . . . (9). Eq 8 applies only if balanced depth is used, whereas E[illegible] gives sufficiently close results for other depths, and is used instead of Eq 8. Table 6 gi[illegible] values of the constant $S = f_s j + 12$, which may be used in formula Eq 9 with M in ft-[illegible] thus, $A_s = M + Sd$.

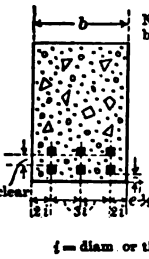
Example 1. Using same values of f_c, f_s and n as above, design a balanced beam f[illegible] load of 1 000 lb per lin ft, span 10 ft. (Load is taken to include wt of beam itself; [illegible] narily, a dead load must be assumed for the beam, and a tentative design made, whi[illegible] modified if the resulting wt of beam does not check the assumed wt.) For this beam, $M = wl^2 \div 8 = 13\,000$ ft-lb. From Table 6, $C = 107.4$ (in-lb units) or 8.95 (ft-lb u[illegible] hence, from Eq 5, $d = \sqrt{13\,000 \div 8.95\, b}$. Value of b is now assumed, so as to give s[illegible] factory proportions. By trial, if $b = 8$ in, $d = 13.5$ in, which is satisfactory. F[illegible] Table 6, $p = 0.0077$, and from Eq 8, $A_s = 0.0077 \times 8 \times 13.5 = 0.83$ sq in; or from table, $S = 1\,165$, and from Eq 9, $A_s = 13\,000 \div (1\,165 \times 13.5) = 0.83$ sq in. Table shows that several combinations of bars may be used to give this area; thus, 2 0.75-in rounds (area = 0.884) or 3 $\frac{5}{8}$-in rounds (area = 0.921).

Table 7. Areas, etc, of Round and Square Bars

Areas in sq in, wts in lb per lin ft, circum, or perimeter in ins

Size, in	Round bars			Square bars			Size, in	Round bars			Square bars		
	Area	Wt	Circum	Area	Wt	Perim		Area	Wt	Circum	Area	Wt	Perim
$\frac{1}{4}$	0.049	0.167	0.785	0.062	0.213	1.000	1	0.785	2.670	3.142	1.000	3.400	4 [illegible]
$\frac{5}{16}$	0.110	0.376	1.178	0.141	0.478	1.500	$1\frac{1}{16}$	0.994	3.390	3.534	1.266	4.304	4 [illegible]
$\frac{3}{8}$	0.196	0.668	1.571	0.250	0.850	2.000	$1\frac{1}{8}$	1.227	4.172	3.927	1.562	5.313	5 [illegible]
$\frac{5}{16}$	0.307	1.043	1.964	0.391	1.328	2.500	$1\frac{3}{16}$	1.485	5.049	4.320	1.891	6.428	5 [illegible]
$\frac{3}{4}$	0.442	1.502	2.356	0.562	1.913	3.000	$1\frac{1}{4}$	1.767	6.008	4.712	2.250	7 65[illegible]	6 [illegible]
$\frac{7}{8}$	0.601	2.044	2.749	0.766	2.603	3.500	....	...	...	...	...	...	...

n choosing the number and size of bars, there are several points. A minimum number esirable (2 for this beam), and if 2 are used the area is less than for 3, resulting in a ng in cost, and there is no extra charge ("size extra") for the 0.75 size, while there is for ⅞. But, if web reinforcement be necessary hereafter), it will be found to be desirable ave more than 2 bars. Also, there should certain MINIMUM SPACING BETWEEN BARS 16). In the above case, if 2 bars are used, am of 7 by 0.75 or 5.25-in is necessary; or 3 bars, 6.25; hence, either combination is factory from this standpoint. Fig 17 shows il of the beam using 2 bars. For slabs, the spacing permitted is 2.5 d.



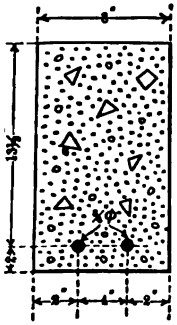
Number of bars in row	Least breadth b
1	4f
2	7
3	10
4	13
5	16
6	19
7	22
8	25
9	28
10	31

f = diam or thickness of bars

Fig 16. Spacing of Reinforcing Bars

nce d is the depth to center of steel, to get otal depth an amount e must be added for ection, etc. Common values of e are: For : when d is 3.25 in and under, e = 0.75 in; 1 d is between 3.25 and 4.75, e = 1 in; 1 d is 4.75 and over, e = 1.25 in. For ns and girders, the clear depth below steel, $-$ 0.5 i, must be 1 in when d is 10 in or under; 1.5 in when d is between 10 and 20 nd 2 in when d is over 20 in.

ample 2. Suppose a depth of 12 in is selected, that is, less than the balanced depth xample 1. The concrete will control and be stressed to full working strength, while the steel will be understressed. To keep f_s below 650, the beam must be over-reinforced. Thus, solving Eq 3, x = 6 in, or k = 0.5. From Eq 7, p = 0.0167 (against 0.0077 for balanced depth), and A_s = 0.0167 × 8 × 12 = 1.60 sq in. Also f_c will be 650 and f_s (from Eq 1) = only 9 750 lb per sq in.



: 17. Beam with 2 Reinforcing Bars

Example 3. Assuming 15 in for d, less steel may be used, and it will be stressed to full value, while the concrete will be understressed. The beam will be under-reinforced. Using S = 1 165, as for a balanced beam, Eq 9 gives a sufficiently close approximation for A_s, or 0.744 sq in. Hence p = 0.0062, and k from Eq 4 = 0.348. Since $j = d\,(1 - k \div 3)$ = 13.26 in, from Eq 9, f_s = 15 850, and from Eq 1, f_c = 570 lb per sq in.

Eq 5 and 9 are also used in designing slabs. EXAMPLE. Required a reinforced-concrete floor slab of balanced depth to carry 500 lb per sq ft on a 10-ft span. Take f_c = 500, f_s = 16 000 and n = 12. M = 6 250 lb for a strip 12 in wide. From Table 6, C = 5.17, hence since b = 12 from Eq 5, $d = \sqrt{6\,250 \div (5.17 \times 12)}$ = 10 in. Also, from Table 6, 1 212 and from Eq 9, A_s = 6 250 ÷ (1 212 × 10) = 0.43 sq in per lin ft of slab. sary spacing of bars of any size, to obtain this area, inches = 12 times area of 1 bar uired area A_s per lin ft. Thus 0.5-in round bars, 5-in centers, could be used, or ⅝ 8-in centers.

fe load. In using these formulas to compute safe load for a beam, first $p = A_s \div bd$, then compute k from Eq 4 and M_s and M_c from Eq 2, 3, using maller value.

mple. Find safe moment for a rectangular beam 8 in wide, 12 in deep to center of and having 0.96 sq in of steel. Take f_c = 650 and f_s = 16 000 lb per sq in, and n = 15. .01. From Eq 4, k = 0.418 and z = 5.0 in. From Eq 2, M_c = 159 000 in-lb. Eq 3, M_s = 134 300 in-lb. This beam is over-reinforced, and would fail by crushing concrete. It could carry 159 000 in-lb moment, and the stress in steel would not l 16 000, but the concrete stress would, hence M must not exceed 134 300, at which g f_s = 13 500 lb per sq in, from Eq 2. Test also for shear and bond.

agonal tension, shear and bond. Besides failing by crushing of the ete or breaking of the steel, a concrete beam may fail by diagonal tens

or by bond failure. Horiz and vert shear, modified by the fiber stress due to bending, produce max tensile stresses having direction shown by curved lines in Fig 18; hence, best position of reinforcing steel would be curved to follow these lines.

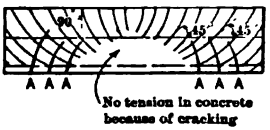


Fig 18

If horiz bars only are used, since concrete has little tensile strength, a beam may fail due to DIAGONAL TENSILE STRESS at ends, causing cracks A,A, as shown. As it is impracticable to compute diagonal tension, which is dependent on shear, shear is used as a measure of the diagonal tension in experiments and design. To prevent failure in this way 3 methods are used (9):

1. Shear is kept below safe value. It is recommended (23), for 1 : 2 : 4 concrete (for other mixes see Table 5) having horiz bars only, that the max intensity of vertical shear v, which $= 8\,v_0 \div 7 = 8\,V \div 7\,bd$ . . . . (10), should not exceed 4c lb per sq in. In designing slabs, where it is impracticable to place web reinforcement such as required in methods 2 and 3, the depth d is increased to keep v within this value.

2. Horiz bars are bent up at ends, usually at 45°, to conform more or less to curve in Fig 18. If this is done, v may reach 60 lb per sq in. This method is sometimes used for beams. Compute distance from support to point where $v = 40$, and beyond which web reinforcement is required. Bend up one or more bars at 45° within this distance. Compute aver total vertical shear V for length of beam s, in which shear is to be taken by bent up bar. Concrete is assumed to take ½ this shear and stress in bar bent up at 45° is $0.54\,Vs \div d$ . . . . (11), which should not exceed safe stress for bar. Note that bar can not be bent up until no longer required for tension reinforcement in bending. Also the BOND STRESS in the remaining horiz bars should not exceed a safe value. The stress u will be greatest at support, and $u = 1.14\,V \div \Sigma o d$ . . . . (12). If this exceeds 80 for plain and 100 or 120 for deformed bars, more bars must be used, or the ends of bars must be hooked to prevent their pulling out, or the length of beam over the support must be increased to obtain a length of embedment, $l' = f_s i \div 4\,u$ . . . . (13).

3. Stirrups are used. If these are figured to take ⅔ the shear (concrete is assumed to take the other third as in method 2), and in addition horiz bars are bent up, v may reach 120 lb per sq in (23), which is the max limit.

Stirrups (Fig 19) should be securely fixed to horiz bars, and not over 6" apart. Design is illustrated by following example for a T-beam. Note that in T-beams the stem only is considered effective for shear; b' is used in Eq 10 for b and computations are made same as for rectangular beams.

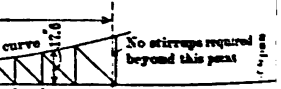


Fig 19. Common Forms of Stirrups

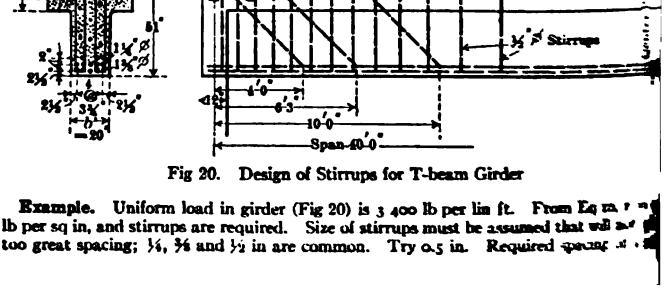


Fig 20. Design of Stirrups for T-beam Girder

Example. Uniform load in girder (Fig 20) is 3 400 lb per lin ft. From Eq 12, v = 60 lb per sq in, and stirrups are required. Size of stirrups must be assumed that will not allow too great spacing; ¼, ⅜ and ½ in are common. Try 0.5 in. Required spacing at end

tirrups (in) $= 1.31 f_s a_s d + V$ . . . . (14). If stirrup is bent as in cross-section in 0, $a_s = 4 \times 0.196$, and spacing at support, where $V = 68\,000$ lb, is 8.8 in. As the is uniform, at a point $l \div 8$, or 5 ft from the support, the spacing will be ½ of 8.8 or in, and at $l \div 4$, or 10 ft, $2 \times 8.8 = 17.6$ in (at $3\,l \div 8 = 4$ times). These values ıid off above the drawing of the beam, and to same scale (as in Fig 20). Draw a th curve through them. For a simply supported beam place first stirrups 0.5 minimum ng from center of support; for a continuous beam, same amount from edge. Draw al line in diagram, and from its intersection with the curve draw line at 45° locating stirrup, as shown. To find how many and where horiz bars may be bent up, from ı for one ⅜-in bar, u at the support would be 485 lb per sq in; hence, to keep this less 120, at least 4 or 5 bars must run to end of beam. All the bottom row, and center f second row, will run to end. Bend up the 1.25-in bars 2 at a time. Find distance support x_2, from formula $x_2 = l\left(1 - \sqrt{m_2 \div m}\right) + 2$ . . . . (15), where m = total of bars and m_2 = area to be bent up. Table 8 is computed from this formula, but ot be used as bars are not of same size. Thus, since area of 2 1.25-in bars is 2.554 and total area is 16 sq in, $x_2 = 12.2$ ft, etc, and bars are bent up, as shown in Fig 20, within these limits.

Table 8. Distance from Support for Bending up Horiz Bars, in decimals of span (uniformly loaded beams)

Total number of bars bent up	Number of bars in center of beam									
	2	3	4	5	6	7	8	9	10	11	12
2	0.114	0.210	0.250	0.276	0.295	0.310	0.321	0.330	0.340	0.348	0.354
3	2	0.93	0.146	0.182	0.210	0.234	0.250	0.265	0.276	0.286	0.296
4		3	0.073	0 114	0.146	0.171	0.193	0.210	0.227	0.240	0.250
5			4	0.053	0.093	0.122	0.146	0.164	0.182	0.197	0.210
6				5	0.044	0.078	0.104	0.127	0.146	0.161	0.178
7					6	0.036	0.068	0.092	0.114	0.130	0.146
8						7	0.032	0.060	0.083	0.101	0.120
9							8	0.028	0.053	0.074	0.093
10								9	0.026	0.047	0.068
11									10	0.023	0.044
										11	0 020

ample: span 20 ft; 6 bars in center; 2 can be bent up at 0.210×20 or 4.2 ft from support

beams. It is noted that the area of concrete below neutral axis in a rectilar beam is of no value against bending. If part of it is removed the re-g T-beam (Fig 20) will be stronger, as it is relieved of some dead load. e, as long as t exceeds x, the beam is essentially rectangular, and the formu-r rectangular beams are used. If t is less than x, use following formulas.

erring to Fig 21 it is seen that rectangular beam Eqs 1, 6, 9, hold equally well for T-. The compression carried by the small portion of the stem above neutral axis is y neglected. Practically $z = x - t/2) + (2x - t)]t + 3$ . . . . and can never exceed $t + 2$; approx, $M_s = f_s A_s[d - (t \div$ . . . (17). Actually $M_s =$ . . . . (9), where $jd = d -$ lso $M_s = [(2x - t)\,b[f_c jd] +$ . . (18), and $x = (2\,nA_s d +$ $(2\,nA_s + 2\,bt)$ . . . . (19).

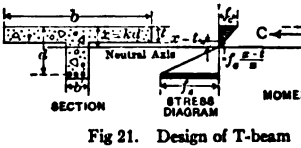


Fig 21. Design of T-beam

lesigning T-beams, a balanced depth is seldom used, but the proper proportions of re arrived at from other considerations, such as area required for shear, headroom, In general, proper proportions for stiffness require that b' shall be between ¼ and ½ so, where the stem forms part of a floor, the slab of thickness t being the flange.

economy the relation of b' and d should be such that $d = \sqrt{rM + f_s b'} + t + z \ldots$ where r = ratio of unit cost of steel to concrete, or 50 to 70. The value of t, where [illegible] forms part of a floor slab, is fixed, and b should not be taken greater than $8t +$ [illegible] distance c to c of T-beams, or over 0.25 the span. For independent T-beams, b should not exceed $3b'$, and t should not be less than $\frac{1}{2}d$. Three problems may arise.

Example 1. T-beam is part of a floor system; $t = 4.5$ in. Design stem for [illegible] of 92 700 ft-lb, max reaction = 16 800 lb. From Eq 11, remembering max allowable shear is 120 lb per sq in, the minimum area of stem $b'd$ for shear = 16 800 ÷ [illegible] = 160 sq in, or 8 by 18 would be satisfactory, as b' would be between $\frac{1}{3}$ and $\frac{1}{2}$ [illegible] required for proper proportions. But, from Eq 20, when $b' = 8$, it will be economical to make $d = 26.75$ in; but b' would then be too small, and if $b' = 10$, $d = 24$, so these proportions will be tried. From Eq 17, $A_s = 3.2$ sq in approx, which can be made [illegible] using several different combinations, as [illegible] and 2 0.75-in rounds, 2¾-in and 6 [illegible] rounds, etc. Using either combination, steel will have to be placed in 2 layers. [illegible] d = depth to center of gravity of steel, total depth must be determined as in Fig. [illegible] where $e = 1.5$ in. This design should be checked up by the exact formula, $8t + b' = 46$ in, and from (19) $z = $ [illegible] hence, this is a true T-beam and Eq 16 [illegible] to find $z = 1.82$ in; also from Eq 9, $f_s = 15\,400$ and from Eq 1, $f_c = 374$ lb per [illegible] which is satisfactory. Stirrups must also be provided.

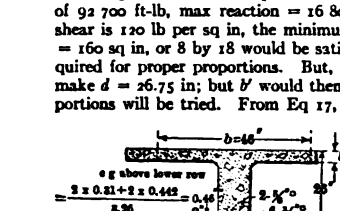


Fig 22. Design of T-beam

Example 2. Design an independent T-beam girder for 40-ft span, to carry live [illegible] of 3 000 lb and assumed dead load of 1 400 lb per lin ft. $M = wl^2 \div 8 =$ [illegible] in-lb. Max $V = 88$ 000 lb; hence, minimum area for shear = 838 sq in, or if [illegible] $d = 53$ or $b' = 18$, $d = 47$, etc. Selecting 18 by 47 as good proportions, and since [illegible] be at least $\frac{1}{2}d$, or say 16 in, this may be designed as a balanced T-beam. Then Eq 6, $k = 0.379$ and $x = 17.8$; hence, a true T-beam. From Eq 16, $z = 5.82$ in 2[illegible] 41.18. From Eq 9, $A_s = 15.80$ sq in, and solving Eq 18, $b = 44.8$ in, which is less [illegible] as required; hence design is satisfactory. To get 16 sq in, 7 1.25 and 5 1¼-in [illegible] would be required, and 3 rows of steel are necessary. To place this steel properly a stem is advisable; hence it is made 20-in instead of 18-in, and steel is placed as in [illegible] Dead load of this section is 1 480 lb per lin ft, which is not enough in excess of [illegible] sumption to require a redesign. END BEARING required is at least 88 000 × [illegible] 450) = 8.8 in. This assumes an allowable compression of 450 lb per sq in, and [illegible] the load by 100% to allow for unequal distribution over the bearing surface.

Example 3. Given T-beam forming part of floor; t, b', d and A_s being known; SAFE LOAD. Find b from data above, and x from Eq 19. If x is less than t, use [illegible] lar beam formulas; if greater, it is a true T-beam and z is found from Eq 16, giving [illegible] is computed from Eq 9 and M_s from Eq 18, and the smaller used. Also, test [illegible] shear and bond.

Columns. Long columns in concrete should be avoided, as variation in quality, almost certain to occur, seriously affect their strength. Usual [illegible] makes limiting area at least 64 sq in, length of plain columns, 12 and rein[illegible] 15 diams. Working loads are computed from tests of cylinders, columns considered as short blocks.

When reinforcement is used, the shortening of the concrete and embedded steel [illegible] the same, hence stress in steel will be n times that in concrete, or usually 8 to 10 [illegible] sq in. It is often held that cement is cheapest reinforcement for concrete [illegible] where space is not particularly valuable, and plain columns of rich mix are [illegible] economical. A 1 : 2 : 4 mix at least is always desirable. The outer 1.5 or 2 in of [illegible] is neglected in estimating strength of column and is considered as protection only. [illegible] able stresses for columns are given in Table 5.

Plain columns are designed for the necessary area, and the requisite amount of [illegible] terial for protection is added. Vertical steel rods, with spacing bands at intervals greatly to the reliability of the column.

einforced-concrete columns comprise: (*a*) those with vertical steel only, (*b*) hooped nns. Since the stress in steel must be n times that in concrete, the formula for longi- ıally-reinforced columns is $f = f_c[1 + (n - 1)\,p]$, where f = the aver allowable ɔression on net area, lb per sq in, f_c, n and p being as already given. Vertical rein- ment from 1 to 4% is used, 2 to 3% being usual economical limit. Vertical reinforcing, t straight and free from bends and symmetrically placed, is of doubtful value. It is to use wire ties, to resist outward flexure of rods, at intervals of not more than 18 in, ı times the rod diam. The compression rods should extend upward or downward into ɔlumn, lapping the reinforcement above or v enough to develop the stress in the rod by llowable unit for adhesion (13), or the ends of may be cut square and faced, and placed on of those above or below, with a short piece pe put over the end as a sleeve connection 23).

ldition of bands or spirals to columns having tudinal reinforcement has little effect upon strength up to the point at which they would vithout hooping. Columns reinforced with tal steel only are brittle, and fail suddenly ı yield point of the steel is reached, while use ops greatly increases reliability and ultimate gth. For this reason a smaller factor of y is used for hooped columns, than for col- ı with longitudinal steel only.

e longitudinal steel is figured as before, using formula. Total amount of hooping must be ss than 1% of volume of the column core en- d, clear spacing of hoops not over ⅙ the diam is core (preferably 0.1) or more than 2.5 in Hoops must be held firmly in place (Fig 23); ffect of hooping, which tends to prevent lat- flow of the concrete and is similar to the n of a pipe filled with concrete, decreases ly as ratio of length of column to diam of increases beyond 5, and it is best to limit hooped columns to 8 diams in length for the es of Table 5. A size for hooping steel can be selected, and the spacing found for d hoops from $S = 3.14\,d^2 + D$, where d = diam of wire. in (usual sizes, ½, ⅝, ¾, ⅞, nd 1 in), and D = diam of core, in. Footing detail for vertical reinforcing rods is im- nt, and, where length of embedment is insufficient to take all the stress, large rods d rest on suitable base plates in the foundation concrete; smaller rods may have a derable length bent outward into the footing. Bending stresses due to eccentric loads ateral forces must be met by increasing the section until the max stress (Art 6) does ceed the values of Table 5. Composite columns of structural steel, filled and covered concrete, form a class by themselves and can not be designed by the above formulas. *Bull* 56, Univ of Ill Expt Sta.)

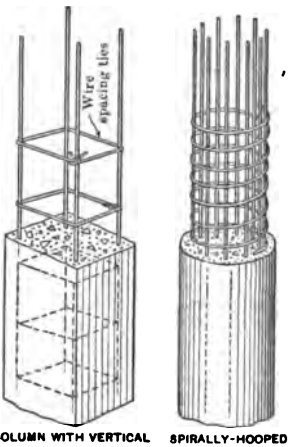


Fig 23. Reinforced-concrete Columns

15. Retaining Walls (9)

neral. Retaining walls are of masonry, sometimes laid dry in unimportant walls, ete or rubble concrete, or reinforced concrete. Principal forces acting are earth t and wt of the wall itself. In gravity walls these are the only forces to be con- d, but in reinforced walls the stability of the various parts must be investigated.

ravity walls (Fig 24). Considering 1 linear ft of wall, the earth thrust E be combined with wt of wall W, producing resultant R, and if R passes de toe of wall A the wall will overturn. R may be resolved into a vertical ɔnent V and horiz component H.

s generally required: (*a*) that R shall cut the base inside the third point; that is, ase will be eccentrically loaded. The formula of Art 6 applies, and can be reduced = $[1 \pm (6\,x \div l)]\,V \div l$ for a rectangular base 1 ft wide, where x = distance from of base (Fig 25), V = as above, and where + sign is used to obtain p_{max} and the —

sign to get p_{min}. When $x = o$, $p_{min} = p_{max} = V + l$; when R acts at third point $x = l + 6$ and $p_{min} = o$, while $p_{max} = 2V + l$; (*b*) that p_{max} shall not exceed the safe foundation pressure; (*c*) that H must not exceed the frictional resistance to sliding of the wall on its base. Resistance offered by the earth in front of the wall is neglected, and considered as simply adding to its safety. These requirements also apply to any horiz joint through the wall, but the base is generally the critical joint.

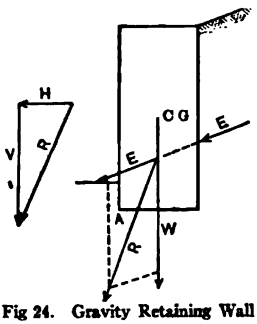


Fig 24. Gravity Retaining Wall

Requirement (*a*) is usually considered standard practice. It insures that there shall be no tension or uplift on the base, but it violates the general principle that center of base and center of press should coincide. In Art 7, safe foundation pressures are given for use in absence of better data. In selecting a safe value and designing the base so this value will not be exceeded as required in (*b*), the fact that the base press will not be uniformly distributed must be allowed for. With dry, compact foundation material, there is little danger of its soaking from the front to the rear of the base, and for walls foundations piles are generally necessary. Most retaining walls on earth settle forward slightly, and for this reason and for appearance the front is battered 0.5 to 3 in per ft. Foundation should be carried below frost line, usually 4 or 5 ft.

Stability against sliding is amply provided for in gravity walls if condition (*a*) is satisfied. Following are coeffs of friction of concrete on foundation materials: dry clay, 0.50 (or $\phi' = 45°$); wet or moist clay, 0.33 (18°); sand, 0.40 (22°); gravel 0.60 (31°); wood with grain, 0.60 (31°); wood across grain, 0.50 (45°). In reinforced retaining walls, sliding requires especial attention, due to their lighter wt, and projection of the base is often necessary (Fig 30, 31). To find the resisting value of this projection draw a line from its lowest point, making an angle ϕ with the horiz; take the wt of material above this line and multiply it by the coeff of friction of the earth.

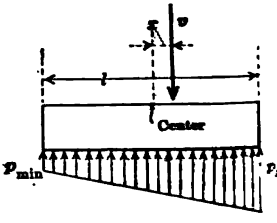


Fig 25. Maximum and Minimum Pressures

Earth pressure. No discussion of the formulas for earth press will be given (see 8). All writers agree that: (*a*) resultant thrust acts ⅓ up from base although experiments indicate that it varies from ⅓ to ½; (*b*) pressure varies directly with depth, or

$$E = Kwh^2 \div 2$$

where E = total press, lb, against a strip of wall 1 ft wide and h ft high, w = wt of filling, lb per cu ft, and K = a coeff, which is independent of the wt or depth of filling but depends on its angle of friction and other factors.

Theories vary in fixing a value of K. Best known formula is that of Rankine:

$$K = \cos\theta \frac{\cos\theta - \sqrt{\cos^2\theta - \cos^2\phi}}{\cos\theta + \sqrt{\cos^2\theta - \cos^2\phi}}$$

which gives value of K for a vertical wall with a fill sloping upward from its top at an angle θ, the material having a coeff of friction represented by angle ϕ. Authorities also differ as to the direction of earth thrust. Rankine's formula is theoretically correct for a granular, dry material, where the thrust acts parallel to surface of fill. If the back face of the wall is inclined, the press against a vertical plane is considered, and this is combined with the wt of the triangle of earth between this plane and the wall, to get the resultant pressure.

Fig 26 gives values of K for various values of ϕ and θ, between limits of $\theta = o$ and $\theta = \phi$. Values of ϕ and w are also given for some aver materials, but in this connection Rankine says: "The properties of earth with respect to adhesion and friction are so variable that the engineer should never trust to tables or to information obtained from books to any

in designing earthworks, when he has it in his power to obtain the necessary data er by observation of existing earthworks in the same stratum, or by experiments." ormula 2 is for active thrust. For passive thrust, or resistance to press of a bank of h, the signs must be changed from $\frac{-}{+}$ to $\frac{+}{-}$; whence $K' = \cos^2\theta + K$. For $\theta = 0$, K io of horiz to vertical press) = 1 for water, since water is frictionless or $\phi = 0$. For

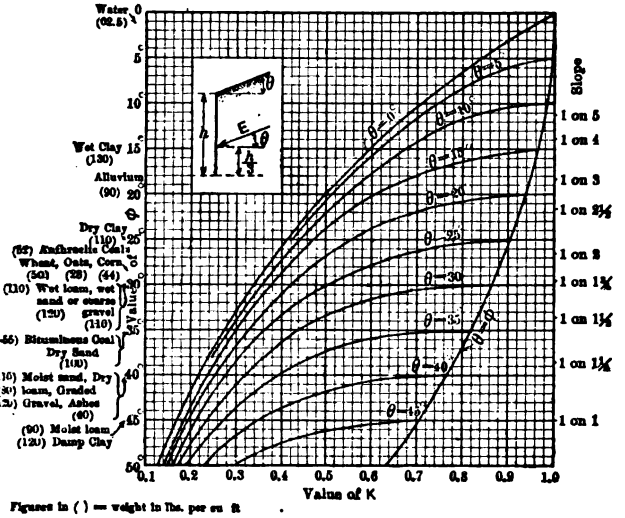


Fig 26. Diagram for Earth Pressure, Vertical Walls

· materials, K depends on the relative fluidity, and many designers use the equivalent r press. This assumes K times w, rather than K and w separately, a relation well h consideration, as it brings out clearly the value of proper DRAINAGE OF FILLING rial.

ɛ 27, *a*, *b*, *c*, shows action of earth thrust. In (*c*) the wall is subjected to a loaded fill h is to be reduced to equiv ft of earth h'. Thus, for a load of 500 lb per sq ft, $w = 100$ r cu ft, $h' = 5$ ft. For RR tracks, h' is usually taken from 6 to 8 ft.

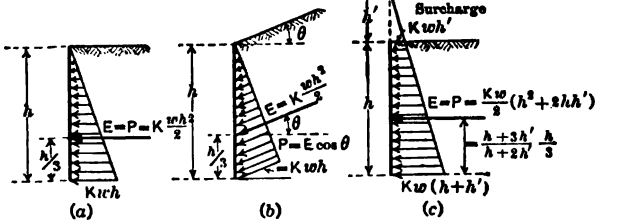


Fig 27. Earth Thrust on Retaining Walls

esign. For small walls, the base is sometimes made simply 0.45 or 0.6 of height, without computing thrust, pressure, etc. For high walls the dimen- s of the base are estimated as above, and then checked by finding the point

where the resultant cuts the base, the intensity of base press, and the stability against sliding. For these computations take a strip of wall 1 ft thick. Rectangular walls are seldom economical, and the trapezoidal form is used. A front batter is assumed of about 1 in per ft, and a top width of 18 to 24 in (largely a matter of judgment). A trial width of base is then taken, 0.45–0.6 h. This fixes the dimensions, and computations are made to see if any modification in base width is necessary.

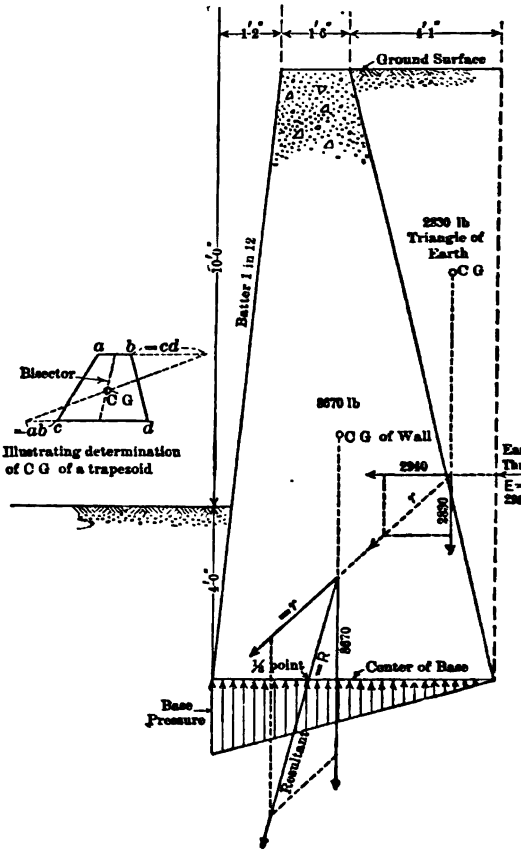


Fig 28. Graphical Design of Retaining Wall

Example. Design wall for 10 ft clear height, borix fill ($\theta = 0$), foundation, 4 ft deep. Assume $w = 100$ and wt of wall 150 lb per cu ft. Material gravel, $\phi = 33°$. Assume top width = 18 in. Take trial width of base $0.48\,h = 6.75$ ft, h being 0.3 (Fig 26). Wt of wall = 8 670 lb, wt of triangle of earth back of wall = 2 830 lb, also $E = 2\,940$ lb (Fig 27a). Fig 28 shows graphical construction, which proves this trial width of base is satisfactory. Max foundation press is 3 400 lb per sq ft, which is very low, and min = 0.

the final design, the back of the wall is usually stepped (Fig 29) to get better contact een the wall and earth, and resist tendency to overturn by friction of earth on earth, not earth on wall, which is less. Also the stepping, which is made to aver up to the sloping back of Fig 28, is convenient in designing sectional forms. The upper 3 ft are often left with a slope, forming a FROST BATTER, to relieve the wall of possible frost thrust. Expansion joints must be provided at intervals of 30 to 60 ft (Art 13). The variation from the trapezoidal form in Fig 29 results in a slight saving of material and smaller base press.

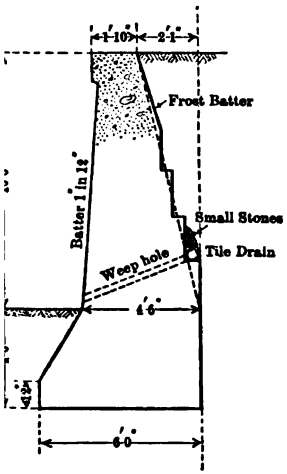


Fig 29. Stepped Retaining Wall

Reinforced-concrete retaining walls are of 3 types: cantilever, counterfort and cellular. Forms for the counterfort wall cost more than for cantilever, but the counterfort is more economical for heights 20 ft, while the cantilever is cheaper than the ity wall, except for very low walls. The cellular has been used where foundations are poor, and piles can not be used, and where wall must be built close to property line. (*Eng-Con*, May 10, 1911.)

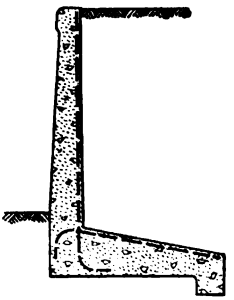


Fig 30. L-type Cantilever Retaining Wall

Cantilever wall of L-type (Fig 30) is required when the front face is on the property line, and the foundation can not extend beyond it. The base press in such walls is liable to be high, requiring the use of piles. Whenever possible, the T-type (Fig 31) is used, as it not only requires a smaller base and is therefore economical, but it gives smaller base press. The vertical stem should be placed at about the third point of the base.

DESIGN OF A CANTILEVER WALL (Fig 31). (*a*) Estimate is made of thickness of footing *ab*, which determines approx height of the vertical wall. Top width is assumed as 12 in, and base *ac* is found by Eq 5 (Art 14), considering the wall as a vertical cantilever subject to the earth thru

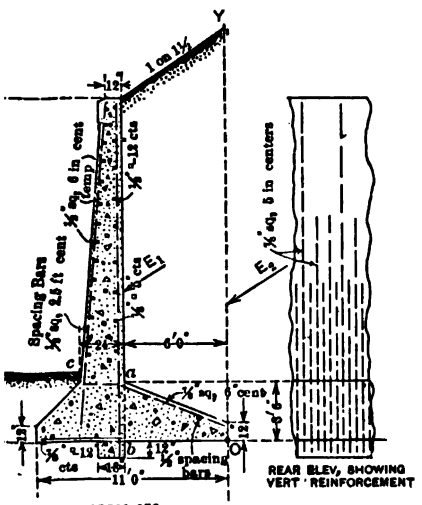


Fig 31. T-type Cantilever Retaining Wall

uniform, and solid rock should generally be required. For dams under 30 or 40 ft a trapezoidal section is economical, with the upstream face vertical or nearly so, a top width of 4 or 5 ft, and a height above extreme high water and waves of 2 or 3 ft. Base dimension is found as for retaining walls, with same 3 conditions for stability (Art 15). For low dams the base is determined by resultant passing inside middle third, but in high dams the allowable press limits the length of base or of a horiz joint through the dam. For rubble masonry 8 to 15 tons press per sq ft is permissible; for concrete, 8 to 10 tons. A force of 20 tons per lin ft of dam, acting at water level, is sometimes allowed for ice thrust. Rarely allowance has been made for possible upward water press on base due to seepage, but it seems unnecessary when the dam is properly bonded to the foundation and when a cut-off trench is made, as is usual (Fig 35). (Wegmann, "Design and Construction of Dams)

Buttress dams have advantages of construction and design that make them desirable and economical in certain locations.

Upstream face may be inclined as in timber dam (Fig 35*a*). In this way a uniform press is obtained at a reasonable cost, and such dams can be built on a footing less for the ordinary gravity type. Deck of the dam is constructed of arches spanning between the buttresses (*Eng News*, 27, 1915), or of a reinforced-concrete slab (*Sch of Mines Quart*, Apr, 1

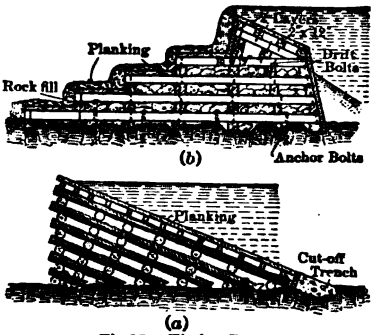


Fig 35. Timber Dams

Timber dams vary from cribwork brush dams to the more costly timber form.

Brush dams were first built as a makeshift, for diversion purposes where funds were lacking for more permanent construction, but several large size are still in use. The brush is tied in bundles with galvanized wire, deposited in layers with the butts downstream and weighted with stones and boulders. Log dams of solid timber are sometimes built, but timber cribs (Fig 35, *a* and *b*) are generally used for higher dams. They are often filled in with boulders, and sheathed with timber. Logs are notched at intersections and secured to each other by wood pins or drift bolts. These dams will last 20 to 50 years, or more if designed to be always wet, that is, the entire width should, if possible, be a spillway. Their max height is 60 or 70 ft.

Spillways, or waste weirs, must be able to pass safely flood flows, and are therefore designed with a liberal safety factor. When possible the spillway should be apart from main dam. Its length is usually fixed by flood width of the stream channel below, and probable depth of flow H can be found as in Sec 38, Art 10.

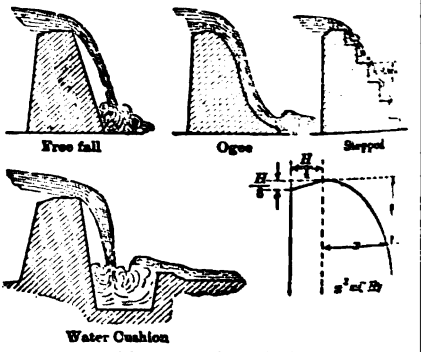


Fig 36. Spillways for Dams

Fig 36 shows types of spillways. When the stream bed is solid rock, if not over about 10 ft high, although a height of 20 ft has been used

/erse-curve section (an American type) the water leaves foot of dam with a direction rallel to stream bed, its velocity being destroyed by the backwater from below. Where backwater exists, and stream bed is of soft or loose material, an apron of concrete or ving is necessary to prevent scour d undermining. Upper part of spill- y curve is full enough to cause the erflowing water to adhere to the face the section; otherwise a partial vac- m might occur, also hurtful vibra- n. Fig 36 shows the curve used, *C* ng 1.6 for high, to 2.3 for low, dams. r gravity spillways, the section must heavy enough for stability. The ver curve is often made the reverse

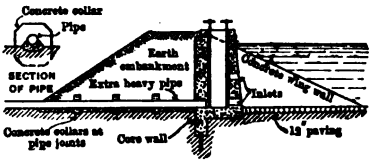


Fig 37. Outlet and Gates for Earth Dam

of the upper. The water cushion, often used by English engineers, is nature's method in many natural waterfalls and is also used with ogee and sloping sections. The depth is generally made about ⅓ the fall; the width, 3 or 4 times the depth.

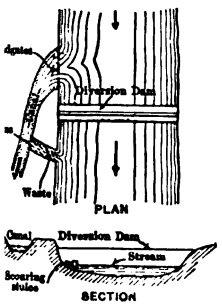


Fig 38. Headworks for Diversion Dam

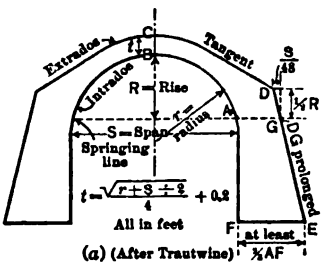
Outlets and gates should be kept away from the dam if possible, so an accident may not endanger the main structure. For low earth dams the arrangement shown in Fig 37 can be used; for higher dams an inlet tower near toe of inner slope is preferable. This allows water to be drawn from different levels, which is desirable for water-supply works. Screens are provided to keep out large rubbish and fish. Area of screen openings should be 2 or 3 times the area of the pipe, and velocity of flow through the screen should be low. Finer screens should be put on portable frames, for ready removal and cleaning.

Gates are generally of the sluice-gate type, obtainable in circular or rectangular form up to 6 ft diam, or y 12 ft, and even larger. They move in tical guides set in the masonry, and are rated by hand wheels, cranks, or electric ors. A blow-off outlet for draining the rvoir is essential.

eepage around the outlet pipes of earth s may cause failure; all parts should be ssible for inspection and repair. Pipes bedded in concrete, with wide collars to ent direct flow of water (Fig 37). In more ly construction the pipe is in a special con- e or masonry tunnel.

leadworks for a dam for diverting water a canal are shown in Fig 38. A scouring e is essential, when much sediment is car- , which is deposited behind the dam due ecrease in velocity of stream at this point. ment can be kept from the canal by open- the scouring sluice during flood flows.

ulverts are small openings of a span of less 1 15 ft, generally built to carry a stream ugh an embankment. Area of waterway nds upon vol be carried, ch in turn ter, area, e, shape e basin 39a). qual to s-sectior of flood,



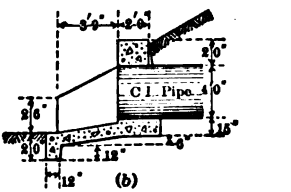


Fig 39. Culverts

which can be found from high-water line. For small openings (up to 4 ft), one or C-I pipes can be used, with concrete end and wing walls to prevent scouring. Fig For larger sizes a concrete or masonry arch can be used.

ANALYSIS OF FRAMED STRUCTURES

17. Simple Frameworks. Loads

Truss is a framework designed to carry loads over larger spans that economical for beams. Fig 40 shows types for bridges; Fig 41, for roofs.

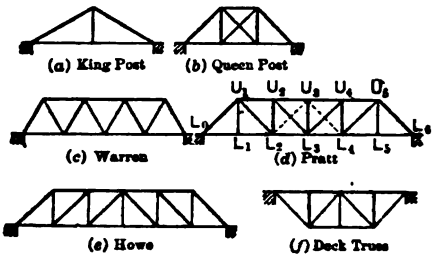


Fig 40. Typical Bridge Trusses

The loads from the roof or a bridge or covering of a truss are transferred to panel points (where members join) by the floor beams or purlins. Hence, true members do not carry loads but are only subjected to longitudinal stresses due to load. But, since their dead wt produces cross bending, the truss members are subjected to combined bending and direct stress. As this is unimportant except in bridge structures, it is customary to consider the dead wt of the truss as concentrated at panel points, and the truss members are designed simply for tension or compression, a − sign indicating compression and + tension.

In bridge work, if the floor system is supported at the panel points of the lower chord it is called a THROUGH TRUSS, if by upper chord a DECK TRUSS. The principle of division into triangles is also used for the bracing of towers and other structures.

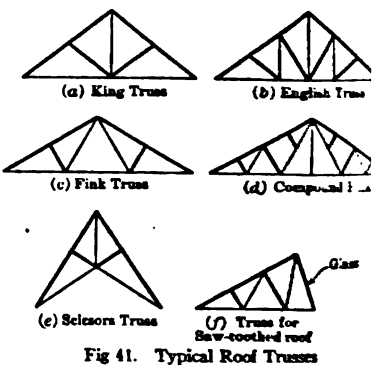


Fig 41. Typical Roof Trusses

In the following analysis only simple trusses will be considered. Structures containing REDUNDANT MEMBERS and other special forms can not be investigated by ordinary methods, as the stresses depend upon the relative elongation of the different members and are computed by the Theory of Least Work. Thus, the braced bent in Fig 40 would be STATICALLY INDETERMINATE, if both diagonals were counterbraced members capable of taking both tension and compression, that is were considered as acting simultaneously. Loading carried by any structure is of 2 kinds, dead load and live load.

Dead load is the wt of the structure itself, and can be closely determined. Table 9 is useful for computing dead loads. In designing, the dead wt must generally be estimated, and the computations revised if the estimate subsequently proves to be in error.

For ROOF TRUSSES the approx wts are found as follows. For wooden trusses, $W = 0.5\, al\,(1 + 0.15\, l)$, where W = total wt of 1 truss, lb, a = distance center to center of trusses

Table 9. Weights of Materials, in pounds

Masonry	lb per
Brickwork	115 cu ft
Cement (Portland)	100 " "
" "	

Metals	lb per
Cast iron	450 cu ft
	480 " "
	490 " "
	505 " "
	711 " "
	438 " "
	41 cu ft
	25 " "
	50 " "
	35 " "
	40 " "
	25 " "
	25 " "
	2—3 sq
	4—6 "
	6—8 "
	6—8 "
	8—10 "
	6 "
	160 c
	160 "
	130 "
	70 "
	200 li
	100 "
	53 c
	48 "
	58 "
	8 "
	62 - 5 "

For
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80
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160

CARS

idges

ors in buildings are usually designed for uniform live loads
arying from 75 to 200 lb per sq ft. For ordinary COUNTRY BRIDGES, a
·100 lb per sq ft is used. Fig 42 shows aver load concentrations due
y cars or steam rollers. For RR BRIDGES a series of concentrated loads

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to passa
follow

a uniform load are used; representing 2 locomotives, followed by a train, and are known as "E 40," "E 50," etc, loadings, indicating the engine axle load. Fig 43 shows E 50 loading (for single-track truss 0.5 of each wheel load would be carried by each truss). E 40 is heavy enough for bridges on small lines, but E 60 is used on major roads.

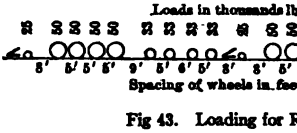

Loads in thousands lb
5 000 lb per lin ft
8' 5' 5' 5' 9' 5' 6' 5' 8' 8' 5' 5' 5' 5' 9' 5' 6' 5' 5'
Spacing of wheels in feet
For load per wheel or axle divide above loads by 2

Fig 43. Loading for RR Bridge

Snow load must be considered in designing roofs in cold climates; 30 lb per sq ft for flat roofs, 20 lb for slopes over 20° and 10 lb (sleet) for roofs over 60° is generally ample. In some structures, temperature stresses must be considered.

Wind load must be allowed for in exposed structures, specifications requiring 30 to 40 lb horiz wind press per sq ft of vertical surface. For sloping roofs the wind press acting normal to the roof may be taken as follows, in lb per sq ft: slope 10°, 9.6 lb; 20°, 13.4; 30°, 26.5; 40°, 33.3; 50°, 38.1; 60° and over, 40 lb. Total press against curved surfaces as chimneys and tanks, is obtained by considering an equivalent flat area of same height and a width equal to ⅔ diam.

18. Methods of Truss Analysis

Algebraic method. Stresses in the members of simple trusses are easily found by applying the 3 laws of equilibrium for coplanar forces. $\Sigma V = 0$, $\Sigma H = 0$, and $\Sigma M = 0$ (Sec 36, Art 30–33). Two procedures are common, method of joints and method of moments, best shown by an example.

Fig 44 shows half of a Fink truss. The loadings P, P, etc, are determined by adding to the roof load, brought to the joints 1, 2, 3, etc, by the purlins, ½ the total wt of the truss which is supposed to be equally divided among the upper panel points. Vertical load 3.5 P, acting downward at A, can be neglected, as it is carried directly by the support. By the method of joints (Fig 44*a*) a section xy is passed, cutting members A 1 and A 5. Since ΣV and $\Sigma H = 0$, by the resolution of forces, and since $R = 3.5\ P$, the stress in A 1 $= 3.5\ P \div \sin\theta$, and stress in A 5 $= 3.5\ P \tan\theta$. By method of moments, a section is passed generally cutting 3 members, the stress in one of which is desired, and the formula $\Sigma M =$ is applied, taking moments about the point of intersection of the other 2 forces. Thus (Fig 44*b*), passing the section ys and taking moments about 4, $3.5\ P\frac{l}{2} - \frac{6}{8}\ Pl -$ (stress in 6–6′) $r = 0$, or stress in 6–6 $= Pl \div r$. The section passed may be either straight or curved. Thus (Fig 44*c*), passing the section mo and taking moments about 4, $(Pl \div r) +$ (stress in 2–7) $r + 2 = 0$, or stress in 2–7 $= Pl + 4\ r$. For the common rise of $l \div 8$ $(= r \div l)$, the stresses in the various members found by the continued application of P. The member 4–8 serves simply to support member 6–6′ at center and is not essential part of the truss and figures as carrying 0 stress. It actually carries 0 wt of 6–6′.

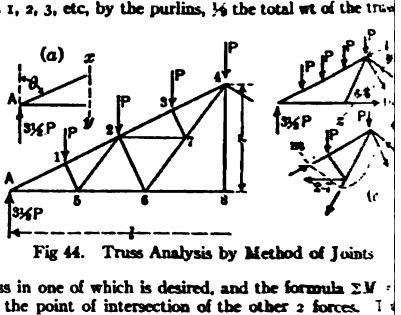


Fig 44. Truss Analysis by Method of Joints

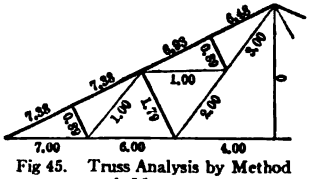


Fig 45. Truss Analysis by Method of Moments

In applying the method of moments to some structures, as the braced bent in g 46, members AC and FD must be prolonged to intersection at O, outside the ıss, and moments taken out O.

Thus, for the wind load own, reactions are equal d may be found by mo- nts about A or F. (Diag- al members are supposed be in tension, and the shed members are not ting stress for the loading own.) To get stress in E, take a section through S, AE, EF (Fig 46a). Tak- ; moments about O, $P_1b - P_2(a + b) - AEs' = 0$, ring stress in AE. To get ess in DE, take section rough DE, BE and BA ig 46b), and moments out B; thus $P_1a - DEy = 0$, giving DE. To get BE, te same section (Fig 46c) d taking moments about $BE \cdot (a + b) - P_1b - P_2(a + b) = 0$, giving BE.

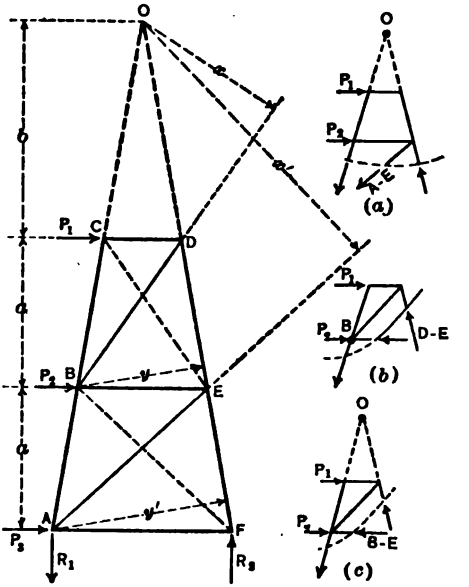


Fig 46. Analysis of a Braced Bent

Graphical method. ıis is often easier and orter than the algebraic ethod, due chiefly to e Bow system of nota- n. There are 2 dia- ams; one a TRUSS DIA- AM, showing lines of action of the forces in connection with the structure elf, and the other simply a FORCE POLYGON. Reactions in many cases are termined graphically, but it may be simpler to obtain them algebraically by ethod of moments (see above).

Thus, take a simple kingpost truss (Fig 47a), carrying a concentrated load at its center. llowing Bow's notation, the spaces between the lines of action of any 2 adjacent forces on the diagram, or the space within any triangle the sides of which are lines of action of any forces, are marked by capital letters. Employing a right-hand system of reading, indicated by the curved arrow, the reaction at 1 is called AB, the concentrated load CA, the member 1–2 is BD, etc. Referring to the force diagram (Fig 47b), the reactions are evidently equal to 0.5 the center concentration. Starting at any point, the line ab is laid off

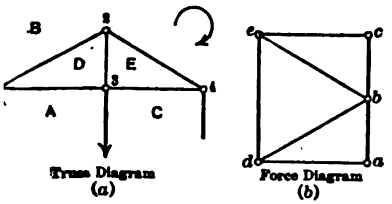


Fig 47. Graphical Method of Truss Analysis

ward from a, representing to scale the left reaction in direction and magnitude. Line bd ben drawn parallel to BD, beginning at b, and line da parallel to DA, beginning at a; their rsection fixes point d. In a similar manner the diagram is completed, as shown. he lines in the force diagram are now scaled off, giving the stresses in different member kind of stress may also be determined as follows: Considering any panel point, say he truss diagram, the member 1–2 is read DB. Noting db in the force diagram, it

seen that the direction from d to b is toward panel point 2, and DB is therefore in compression. Or, at panel point 1, member 1–3 is read DA, and, as direction from d to a is toward right or away from 1, this member is in tension. Analysis of stresses in hoisting headframes by the graphical method is shown in Sec 12.

19. Analysis for Moving Loads

For a fixed system of concentrated loads the max reaction is easily found giving the max shear, and the max moment is then computed by finding the section at which the shear passes through zero (this usually occurs under one of the concentrations known as the critical wheel), the moment at that point being a max. It is usual in small beams to design simply for max moments and shears, and to use the same section throughout. For moving load it must be so placed, with reference to the span, as to produce the max shear or moment.

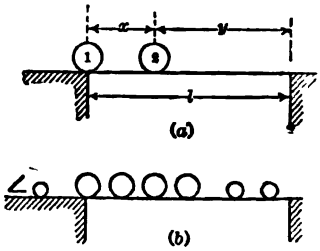


Fig 48. Analysis of Moving Loads

Max reaction or shear occurs at the support when the loads are as near as possible to that end of the span. Thus, in Fig 48a, the 2 loads must be moved so that wheel 1 will be a very small distance from the support, and reaction equals $P_1y + P_1$. For a locomotive (as in Fig 43) the first driver must be over the support (Fig 48b) and be considered as acting just to the right of the support.

Live load on truss bridges is transferred from the longitudinal stringers to transverse floor beams AB which carry it to the panel points (Fig 49a). Beams AB must therefore be designed to carry 2 concentrations, at x and y, equal to the max reaction from the longitudinal members framing in at these points. Sum of reactions from the 2 equal spans will be max when total load is max, and evenly divided between the spans. For a locomotive concentration this generally requires one of the middle drivers at x.

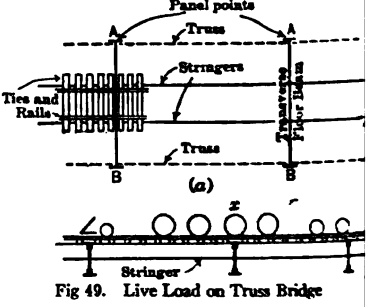


Fig 49. Live Load on Truss Bridge

Max moment at any section in a beam occurs when total load is divided in proportion to distances of this section from each end of beam.

Thus, max moment at section xy (Fig 50) will occur when sum of loads to left ($= W'$) is to sum of all loads ($= W$) as l' is to l. It is sufficiently accurate to determine M by this CRITERION for center of beam, and use it the max moment. But, the max moment will not be at the center, but at a point such that the distance between this point and the center of gravity of entire load is bisected by the center of span (Fig 51c). EXAMPLE. Take 2 equal loads of 1 000 lb, 4 ft apart, on span 10 ft. Max M at center is when 1 load is on either side of center, and equals 3 000 ft-lb (Fig 51a). Max possible moment is when load is placed as in Fig 51b. General formula for 2 equal loads: M occurs where shear $= 0$, when $x = (l + a) - (a + 4)$, and will equal

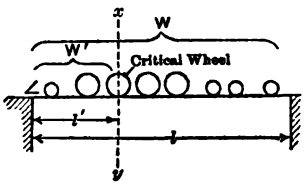


Fig 50. Max Moment on Truss (Approx)

$P + 2b)l - (a + 2)P$. This holds to $a = 0.586\,l$, after which max M is for 1 load at nter of span.

Stresses in truss members due to concentrated moving loads (Bib 11, 2, 13).

Referring to Fig 40*d*, for example, the dead load stresses, or those due to a fixed live ad, may be found by the methods of Art 18. But, if a moving live load is considered, must be moved to positions producing max tresses in each member. Thus, for max stress end post L_0U_1, the reaction of moving load L_0 must be a max, and max stress then quals this reaction + *cos* of angle $L_0U_1L_1$. $_1L_1$ is simply a hanger, carrying the max floor eam reaction at L_1.

For max live-load tensile stress in diagonal $_1L_2$, the load is advanced from right so as produce greatest shear in panel L_2L_3. The RITERION for this (see above) is that the total ad on bridge must equal n times the load on he panel in question. Stress in U_2L_2 will hen = shear + *cos* of angle $L_2U_2L_2$. Max hear in U_2L_2 = reaction at L_0 minus the oor-beam reaction at L_2. This condition of oading also gives the max stress in post U_2L_2, hich will equal the vertical component of $_2L_2$, or the shear. For a load advancing from ft, stress in U_2L_2 will be compression. This ember can be made as a counterbraced mem- er, capable of taking tension or compression, r to take tension only, and another tension ember U_3L_2, called a "counter," can be put , as shown by dotted lines.

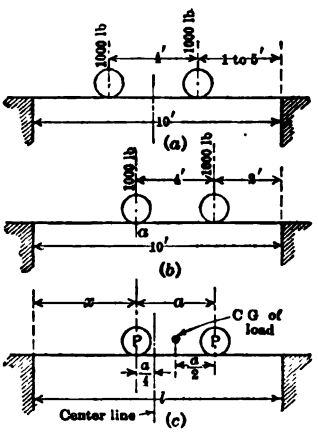


Fig 51. Max Moment in a Truss (Exact)

For max stress in a chord member, the load must be placed so as to produce the max oment in the chord. Criterion for this is the same as for max moment at any section of eam, or $W' : W$ as $n' : n$, where n = total panels in truss, n' = the number to left of ection, W = total load and W' = load to left of section. For U_2U_3, stress = $M \div$ height f truss, where M = reaction at L_0 times its lever arm minus floor beam reactions at L_1 nd L_2, each multiplied by their lever arms, the center of moments being taken at L_0 or ny point in the panel L_2L_3.

TIMBER STRUCTURES

20. Timber

Trade classification of woods: evergreens or soft woods (pines, spruce, emlock, cedar, cypress, fir); hard woods (oak, chestnut, hickory, ash, walnut, aple, birch, whitewood). Some "hard woods" are softer than the so-called ft woods.

Pine. Most useful of all woods. WHITE or northern white, from northern U S and anada: light, soft, not very strong, but easily worked; for interior finish, best grades for ttern making. WESTERN PINE covers timber sold as white pine, coming from Arizona Washington. IDAHO WHITE, from northern Idaho, western Montana and eastern ashington; not as strong as white pine, but resembles it and used for same purposes. OUTHERN YELLOW is of 2 principal kinds. Long-leaf is from southern coast, Virginia Texas, heavy, hard, strong, close-grained, tough; used for heavy framing and floors; n not be used in contact with ground, or built in solidly in masonry, as it decays rapidly ess dry, and dry-rots unless open to air. Short-leaf or Carolina is used as substi- r long-leaf. Variable in quality, the short-leaf being inferior. RED or NORWAY P ed for all construction purposes, is light, hard, coarse-grained. DOUGLAS FIR

OREGON PINE from north Pacific coast is the most valuable lumber of the Pacific region used for structural purposes instead of eastern hard pines; two kinds, red more valuable than yellow; imported to east when large size or length is required.

Spruce. BLACK or eastern from northern U S and Canada; light, soft, close-grained, tough in fiber; desirable for studs, joists and framing in buildings, also for piles, submerged cribs, etc, as it lasts well under water and resists destructive action of crustacea. WHITE: similar to black but not so common. WESTERN or SITKA, north Pacific coast. Giant spruce, soft, satiny, but inclined to be cross-grained; used for interior finish, fences, boats and cooperage.

Hemlock. EASTERN: brittle, splits easily, liable to be "shaky"; used in cheap, rough framing, as substitute for spruce, which it closely resembles in appearance. WESTERN said to be harder and superior in quality, but liable to shrink and warp.

Redwood. Giant tree and most valuable timber of California; used for same purposes in West as pine in East; soft and splinters easily; durable for fence posts, telegraph poles and RR ties (tie plates are used on ties to prevent wear).

Cedar. WHITE or arborvitæ: soft, fine-grained, durable, but lacks strength and toughness; used for tanks, shingles, boats, fencing, cooperage. RED: small size, very durable but brittle; strong, pungent odor; used for fence posts, shingles, etc.

Cypress, similar to cedar, used in northern U S for shingles, but in south as extensively as pine in north.

Oak. WHITE, from eastern U S: heavy, strong, hard, tough and close-grained; liable to check if not well seasoned; used where great strength is required in framing, boat building, etc. RED oak is similar, except in color.

Miscellaneous. HICKORY is hardest, toughest and strongest of all American woods; for tool handles. ASH, for interior work; is unfit for outside, as it decays rapidly. ELM, tough and very close-grained; used in wagons, cars, boats, barrel-staves.

Structure and sawing. The mature, compact inner layers are called HEARTWOOD; the outer 0.25 to 3 or 4 in being SAPWOOD, soft, open, and full of sap which hastens decay. Bark should always be removed. The fibers of the tree are in circular layers separated by ANNUAL RINGS, which are interrupted by radial cells communicating with the pith at center and the sapwood, and known as medullary rays.

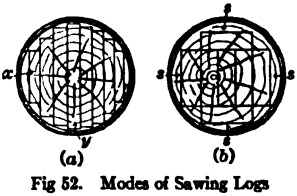


Fig 52. Modes of Sawing Logs

Fig 52 shows modes of sawing common lumber from a log: (*a*) for plank of uniform thickness; (*b*) for joists, etc, of same thickness and width. The part of log section from which lumber comes is of great importance. Thus, in (*a*) piece *x* is liable to warp or curl, due to unequal length of fibers on 2 sides, and if used in flooring is liable to splinter off. *y*, the best cut, is said to be RIFT-SAWED or comb-grained; makes best and most durable flooring. *sss* are SLABS. QUARTER SAWING, in plane of medullary rays, gives handsome finish.

Commercial sizes. FLOORING embraces 4, 5 and 6 quarter-in thicknesses, by 3 to 6 in wide, excluding 1.5 by 6. BOARDS, thicknesses under 1.5 in by over 6 in wide. PLANK, sizes from 1.5 to under 6 in thick, by 6 in and over wide. SCANTLING, sizes above 1.5 and under 6 in thick, and 2 to 6 in wide. DIMENSION TIMBER, sizes over 6 by 6 in. STEPPING, 1 to 2.5 in thick by 7 in and over in width. ROUGH-EDGE, or flitch, all sizes, 1 in thick and up by 8 in and up wide, sawed 2 sides only.

In U S, lengths increase by 2 ft; in Canada, by 1 ft. In dimension timber, sizes above stated may be obtained, but stock sizes vary in both dimensions by even inches; thus 6 by 6, 6 by 8, 8 by 8, etc. All sizes less than 1 in thick are counted as 1 in, in computing quantity. The unit is the BOARD FOOT, 12 by 12 by 1 in.

Finish. Rough timber and lumber, sawed to standard size, shall be not over 0.25 in scant. Thus, 2 by 6 may be 1.75 by 5.75 in. In small sizes, this greatly

reduces strength. Not more than 0.25 in is allowed for dressing each surface. The term $S\,1\,S\,1\,E$ indicates that 1 side and 1 edge are surfaced; $S\,4\,S$ means surfaced on 4 sides, etc. A 2 by 6 $S1\,S\,1\,E$ is usually worked to 1⅝ by 5½ in.

Shrinkage. Shrinkage is greatest tangentially to the fibers, that is, circumferential shrinkage; radial shrinkage is less. Flat-sawed lumber (Fig 52*a*, *x*) may crack and shrink 3 to 10% of width, thus opening spaces between boards or flooring, if not thoroughly seasoned. Longitudinal shrinkage is very small, generally less than 0.1%, which must be carefully considered in designing structures where settling is important; if possible, all timbers with horiz fibers should be eliminated from vertical supports. Effect of moisture is the reverse, and should also be considered.

Seasoning. Natural seasoning, used for lumber for construction purposes, requires 1 to 3 years, depending on kind of lumber.

Object is to remove moisture and dry up natural gums, thus guarding against shrinkage and decay. Lumber should be piled under cover, high and dry, thorough air circulation being allowed by placing strips between layers. WATER SEASONING consists in immersing timber in water about 2 weeks, to dissolve out soluble substances, and then air drying. Liable to make heartwood brittle or sapwood brashy. KILN-DRYING consists in placing lumber, usually only small sizes, in a kiln and exposing to a current of air at 100 to 200° F for 2 to 6 or 8 days. It generally produces inferior product, due to excessive drying of exterior and imperfect drying of interior.

Defects and inspection (5). Two things must be considered: quality and dimensions. Strongest and most durable timber comes from trees of slow growth, indicated by closeness of annular rings. Best timber comes from heartwood; should be straight in fiber, free from large or dead knots, flaws, shakes and blemishes. Dull, chalky appearance, or disagreeable odor, indicates decay and bad timber. Good timber is sonorous when struck.

In inspecting, timber is graded according to number, position and kind of defects visible in any piece. Thus, lumber is graded as first, second and third clear, or as No 1 or No 2, depending upon STANDARD DEFECTS shown, as pin and standard knots, sap stains, pitch pockets or streaks, checks, wanes or shakes, sap. The number of defects allowed depends upon kind of inspection: square-edge, merchantable, prime and clear inspection (5).

Decay. Life of timber depends on manner and time of felling, seasoning and working, and is always subject to attack by animal and vegetable life. Best time to cut timber is in winter and drier summer months, when cells are dormant and amount of sap small. Alternate wetting and drying, heat and confined air, fungi, insects, and worms are the chief agencies producing decay. Well-seasoned wood, in well-ventilated and uniform condition as to moisture, should never decay. Constant immersion weakens and softens, but does not decay.

Dry rot is due to excretion of ferments by a fungus which attacks cell walls, and breaks them down in the presence of moderate warmth, dampness and lack of ventilation. Occurs in wall pockets at ends of timbers and in cores of timber columns. WET ROT is caused by presence of moisture and warmth, which dissolves out cell walls in sapwood. COMMON ROT is due to improper seasoning without thorough ventilation, and is shown by yellow color at points of contact or yellow dust in checks and cracks. The most active worm is the TEREDO, or shipworm. It attacks timber in clear, unpolluted, sea water, from mean tide to ground level, and is especially prevalent on calcareous shores.

Preservation. CREOSOTING is the best preservative for timber, and is also effective against the teredo. It consists in impregnating with dead oil of coal tar or distillates from the wood tars (creosote). Timber is first placed in cylinders and sap vaporized by steam. After 0.5 to several hr, steam and sap are drawn off by a pump and the cylinder filled with oil, at about 160° F. When gage stands constant, showing no more oil is being absorbed, the surplus is drawn off. The entire treatment takes about 24 hr. Amount of oil varies, with kind of wood and seasoning, from 5 to 12 or 18 lb per cu ft for green and hard timber. Cost is from 10 to 20¢ per cu ft. BURNETTIZING consists in impregnation with $ZnCl_2$, in a similar manner as for creosoting. About 0.24 lb of $ZnCl_2$ is used per cu ft. Cost is about 3 to 8¢ per cu ft. Not suitable for timber exposed to moisture, as the $ZnCl_2$ will leach out. ZINC-TANNIN or Wellhouse process is same as Burnettizing, except that

Table 10. Working Unit Stresses for Structural Timber (lb per sq in)

Kind of timber	Bending			Shearing				Compression					
	Extreme fiber stress		Modulus of elasticity	Parallel to grain		Longitudinal shear in beams		Perpendicular to grain		Parallel to grain		For columns under 15 diam, working stress	Formula for working stress in columns over 15 diams long
	Average ult	Working stress	Average	Average ult	Working stress	Average ult	Working stress	Elastic limit	Working stress	Average ult	Working stress		
Douglas fir	6 100	1 200	1 510 000	690	170	270	110	630	310	3 600	1 200	900	1 200
Longleaf pine	6 500	1 300	1 610 000	720	180	300	120	520	260	3 800	1 300	980	1 300
Shortleaf pine	5 600	1 100	1 480 000	710	170	330	130	340	170	3 400	1 100	830	1 100
White pine	4 400	900	1 130 000	400	100	180	70	290	150	3 000	1 000	750	1 000
Spruce	4 800	1 000	1 310 000	600	150	170	70	370	180	3 200	1 100	830	1 100
Norway pine	4,200	800	1 190 000	590*	130	250	100	.....	150	2 600*	800	600	800
Tamarack	4 600	900	1 220 000	670	170	260	100	.....	220	3 200*	1 000	750	1 000
Western hemlock	5 800	1 100	1 480 000	630	160	270*	100	440	220	3 500	1 200	900	1 200
Redwood	5 000	900	800 000	300	80	.....	.....	400	150	3 300	900	680	900
Bald cypress	4 800	900	1 150 000	500	120	.....	.....	340	170	3 900	1 100	830	1 100
Red cedar	4 200	800	800 000	.....	.....	.....	.....	470	230	2 800	900	680	900
White oak	5 700	1 100	1 150 000	840	210	270	110	920	450	3 500	1 300	980	1 300

Multiply the respective fiber stresses by $1 - (l + 60\,d)$

* Partially air-dried.

These unit stresses are for green timber, and are to be used without increasing live-load stresses for impact.

l = length, in. d = least side, in.

Note. Working unit stresses in this table are for RR bridges and trestles. For highway bridges and trestles, unit stresses may be increased 25%. For buildings, in which the timber is protected from the weather and practically from impact, unit stresses may be increased 50%. To compute deflection of a beam under long-continued loading, instead of that when the load is first applied, only 50% of the corresponding modulus of elasticity given in the table is to be employed.

For special information respecting varieties of Pacific Coast woods, with extensive tables giving moment of inertia, section modulus, wt per linear ft and total safe loads, see "Structural Timber Handbook," published in 1910 by The West Coast Lumberman's Association, 1016 White Building, Seattle, Wash.

a small quantity of glue is added to the Zn, and after completing the Zn treatment a solution of tannin is run in under pressure, which forms, with the glue, a leathery coating, closing the pores and preventing leaching out of the $ZnCl_2$. Several other methods and variations in the above methods are in use. KYANIZING, or impregnating with $HgCl_2$ and $CuSO_4$ is also used. Timber may be made incombustible by injecting Fe_2SO_4 and a solution of $CaSO_4$ or Na_2SO_4.

21. Columns and Beams

Tests and strength. Timber shows wide variation in strength depending on kind and age of tree, moisture content, size of test specimen, imperfections and knots.

Table 10 is from recent tests of selected timber, purchased in accordance with standard specifications of Am Ry Engs Assoc, and subjected to careful inspection. Unit stresses vary with class of construction, those in the table being for trestles and bridges. For other structures they may be increased as indicated, depending on character of loading, and freedom from deleterious weather conditions. No values are given for tension, as it seldom occurs without some bending, and the value for bending should be used. For weights see Table 9.

Table 11. Safe, Uniformly-distributed Load for Plank (lb per sq ft)

Based on fiber stress of 1 000 lb and longitudinal shearing stress of 100 lb per sq in. W = safe load, lb per sq ft = $192\ kd^2 \div l^2$ in bending and = $192\ sd \div l$ in shear, where l = depth of beam, in, l = span, in, k = allowable fiber stress and s = allowable shearing stress, both in lb per sq in.

Span, in	Thickness of plank, in							
	⅞	1	1⅛	1¼	2	3	4	6
12	1 020	1 333	1 688	2 000	3 200	4 800	6 400	9 600
14	750	979	1 240	1 713	2 740	4 120	5 480	8 220
16	575	750	950	1 172	2 400	3 600	4 800	7 200
20	368	480	608	750	1 920	2 880	3 840	5 760
24	255	333	422	521	1 333	2 400	3 200	4 800
30	163	214	270	333	852	1 920	2 560	3 840
36	113	148	188	233	593	1 333	2 133	3 200
42	83	109	138	170	435	980	1 740	2 740
48	.....	83	105	130	333	750	1 333	2 400
54	.....	.....	83	103	263	593	1 055	2 130
60	.....	.....	.....	83	213	480	853	1 920
66	.....	.....	.....	.....	176	397	705	1 587
72	.....	.....	.....	.....	148	333	593	1 333

Table 11 for planking is based on a fiber stress of 1 000 lb, and an allowable longitudinal shearing stress of 100 lb per sq in. When ratio of span to depth of beam is less than the ratio of these stresses (10 : 1 in this case), the longitudinal shear determines the safe uniformly-distributed loading. For a single concentration at the center, the limit is twice this value. The limit is indicated by a heavy line, all values above the line being computed for shear and those below for bending. Table 12 is also based on fiber stress of 1 000 lb, but other table may be used for other stresses by changing these values in direct proportion. In using Table 12, the limiting minimum span must not be exceeded. Thus, for short, deep beams, s must not exceed the allowable value and will equal ⅔ the max vert shear, divided by cross-sectional area; or, for uniformly loaded beams or those loaded with symmetrical concentrations, max load in lb = ⅔ the cross-sectional area times the allowable longitudinal shear. Table 13 gives the unit stresses for columns for ratios of $l \div d$ effective length, in ÷ least side or diam, in) between 15 and 30.

Examples. Required safe uniform load for a 6 by 8-in spruce beam, spanning 4 ft. From Table 12, C = 42 667 ft-lb, which ÷ 4 = total safe load of 10 667 lb; or, dividing

Table 12. Coefficient of Strength (C) in Ft-Lb for Rectangular Beams

(Fiber stress, 1 000 lb per sq in) C in ft lb $= 8M = \frac{8}{12}kS$, where k = allowable fiber stress, lb per sq in, and S = section modulus, in

Depth, in	Thickness, in							
	1	2	3	4	6	8	10	12
1	111	222	333	4 444	667	889	1 111	1 333
2	444	889	1 333	1 778	2 667	3 556	4 444	5 333
4	1 778	3 556	5 333	7 111	10 667	14 222	17 778	21 333
6	4 000	8 000	12 000	16 000	24 000	32 000	40 000	48 000
8	7 111	14 222	21 333	28 444	42 667	56 889	71 111	85 333
10	11 111	22 222	33 333	44 444	66 667	88 889	111 111	133 333
12	16 000	32 000	48 000	64 000	96 000	128 000	160 000	192 000
14	21 778	43 556	65 333	87 111	130 667	174 222	217 778	261 333
16	28 444	56 889	85 333	113 778	170 667	227 556	284 444	341 333
18	36 000	64 000	108 000	144 000	216 000	288 000	360 000	432 000
20	44 444	88 889	133 333	177 778	266 667	355 556	444 444	533 333
22	53 778	107 556	161 333	215 111	322 667	430 222	537 778	645 333
24	64 000	128 000	192 000	256 000	384 000	512 000	640 000	768 000

again by 4, a safe uniform load of 2 667 lb per lin ft. But, the depth of this beam is $\frac{1}{16}$ the span, and the longitudinal shear will control; hence, total load may be only $\frac{1}{16} \times 6 \times 8 \times 100 = 6\,400$ lb, or 1 600 lb per lin ft.

Design a longleaf yellow-pine beam to carry concentrated center load of 10 000 lb, span 26 ft. $M = Pl \div 4 = 10\,000 \times 26 \div 4 - 65\,000$ ft-lb, and $C = 8M = 520\,000$. As the allowable stress for longleaf pine is 13 000 lb per sq in, Table 12 must be entered with $520\,000 \times 1\,000 \div 1\,300 = 400\,000$ as the required C, and a 12 × 18-in beam is found to give 432 000, which is more than ample for this load. (Similarly for Table 13.)

Table 13. Unit Working Stresses, Lb per Sq In, for Wood Columns and Struts (Carnegie)

$l \div d$	Longleaf pine or white oak 1 300 $(1 - l \div 60\,d)$	Douglas fir or western hemlock 1 200 $(1 - l \div 60\,d)$	Shortleaf pine, spruce, or bald cypress 1 100 $(1 - l \div 60\,d)$	White pine or tamarack 1 000 $(1 - l \div 60\,d)$	Red cedar or redwood 900 $(1 - l \div 60\,d)$	Norway pine 800 $(1 - l \div 60\,d)$
15	975	900	825	750	675	600
16	953	880	807	733	660	587
17	931	860	788	717	645	573
18	910	840	770	700	630	560
19	888	820	752	683	615	547
20	867	800	733	667	600	533
21	845	780	715	650	585	520
22	823	760	697	633	570	507
23	802	740	678	617	555	493
24	780	720	660	600	540	480
25	758	700	642	583	525	467
26	737	680	623	567	510	553
27	715	660	605	550	495	440
28	693	640	587	533	480	427
29	672	620	568	517	465	413
30	650	600	550	500	450	400

Deepened, built-up, compound or keyed beams. When no timber of required size is available, beams of this kind are used for temporary work and in some cases for permanent construction, though trussed beams are much lighter. If 2 beams are superposed, they have but twice the strength of one, neglecting friction between them. But, if they are

secured together (Fig 53) they act as a single stick of twice the depth, and are 4 times as strong as the single stick. Compound beam in Fig 53a is fastened by sheeting of planks at least ⅓ the width of timbers, with nails extending at least 0.5 their length into the main members; it has a strength of about 70% that of a solid beam of same dimensions. Beams with hardwood or metal keys (Fig 53b), preferably of a wedged type, are 75 to 80% as strong as solid beams. The keys transfer the horiz shear from one beam to the other, and are of such size that the safe end compression is not exceeded. Spacing must be determined for the safe horiz shearing stress. Keys should be about 2.5 times as wide as thick, and beam should be about 0.5.

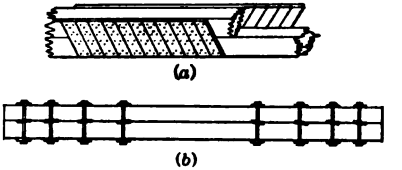


Fig 53. Compound or Keyed Beams

Design. *a.* Design cross-section of beam for bending = 1.25 to 1.33 times that computed (to allow for effic of 80 to 75%). *b.* Compute external vertical shear = V, at governing points. *c.* Compute intensity of horiz shear at neutral axis = $(3 + 2\ V) + bd$. *d.* Compute total horiz shear between end and point of zero shear (total horiz shear = aver intensity between any 2 points × b × distance between points). *e.* Select number of keys (usually 5) and divide (*d*) by this number, designing keys for bearing accordingly. *f.* Draw vertical shear diagram (horiz shear is proportional) and divide into number of equal areas (= number of keys) by vertical lines. Place keys in back of these lines. *g.* Check design for shear between keys. *h.* Design bolts as for tabled fish-plate joint.

Trussed beams are economical for spans over 20 ft, where headroom permits, or for shorter spans where saving in wt is an object.

King and queen-post trusses (Fig 54) are used. Usually the beam is a single unspliced piece, extending from end to end, with iron tie rods passing through oblique holes bored in the ends. If 2 pieces are used, they are separated with tie rods between.

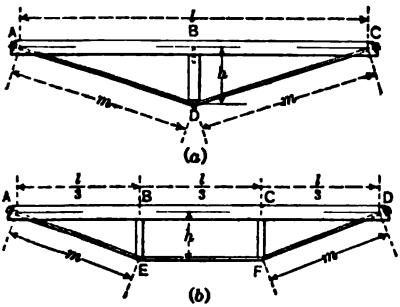


Fig 54. Trussed Beams

When the load is concentrated at the vertical struts, that is, for panel point loading, the stresses are found as in Art 17 and members designed accordingly. The only bending stress that occurs is due to wt of members themselves, which can be ignored except for long spans and small live loads. A uniformly-distributed load is carried partly by the beam and partly by truss action. The stresses can not be computed by ordinary means, but may be found as follows. In Fig 54*a*, king-post truss, W = total load, lb, on beam uniformly distributed. Compression in $BD = 5\ W + 8$; compression in $AB = 5\ Wl + 32\ h$; tension in $AD = 5\ Wm + 16\ h$. Design AD and BD for these stresses. Divide compression in AB by area of beam = bd, giving unit direct compression in AB; add to this the extreme fiber stress due to load acting on AB as a simple beam = $2.25\ Wl + bd^2$, and design AB so this total will not exceed allowable. All lengths are in ft, except b and d = breadth and depth of AB, in. When the truss is above the beam, AD will be in compression, BD and AB in tension, and the latter must be designed for max tension due to direct and bending stress, using same formulas. (Fig 54*b*) queen-post truss. Approximately, compression stress in BE and $CF = W + 3$; compression in AD = tension in $EF = Wl + 9\ h$; tension in AE and $FD = Wm + 3\ h$. Extreme fiber stress in AB, BC, or CD, acting as a beam, = $Wl + bd^2$.

Table 14. Nails and Spikes

Size	Length, in	Cut		Wire		Boat spikes				
		No of nails per lb	No of spikes per lb	No of nails per lb	No of spikes per lb	Length, in	Size, in			
							3/8	7/16	1/2	9/16
2d	1	740	.....	900	.....	3	1 320	....	..	
3d	1.25	460	.....	615	.....	4	1 140	....	..	..
4d	1.5	280	.....	322	.....	5	940	....	....	..
5d	1.75	210	.....	250	.....	6	800	600	450	
6d	2	160	.....	200	.....	7	650	590	380	
7d	2.25	129	.....	154	.....	8	600	510	340	260
8d	2.5	88	.....	106	.....	9	530	400	300	24
9d	2.75	73	.....	85	.....	10	480	360	280	220
10d	3	60	.....	74	37	11	....	320	260	210
12d	3.25	46	.....	57	32	12	....	230	240	190
16d	3.5	33	17	46	29	14	....	....	....	18
20d	4	23	14	29	23	16	....	....	...	16
30d	4.5	16	10	23	18					
40d	5	12	9	17	13					
50d	5.5	10	8	14	10					
60d	6	8	7	10	9					
....	6.5	....	6	.....	8					
....	7	....	5	.....	7					
....	8	....	.....	.....	6					
....	9	....	.....	.....	5					
....	10	....	.....	.....	4					
....	12	....	.....	.....	3					

A keg of nails or spikes weighs 100 lb. Table of boat spikes gives approx number in a keg of 200 lb.

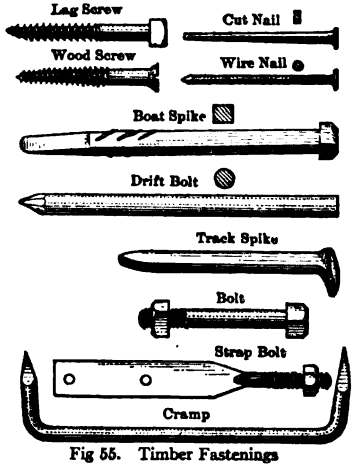


Fig 55. Timber Fastenings

22. Fastenings and Joints

Nails are classified as cut and wire. For lumber liable to split, cut nails are better than wire. They should be driven so that the tapering sides bear against the bent ends of the wood fiber, thus preventing splitting and acting like a wedge. Wire nails may be clinched or bent down; cut nails can not be bent. Table 14 gives common forms, designated in size by the "penny" system (d), probably originally the number of lb to 100 nails. Number per lb varies with different makers. Cost about 2 to 5¢ per lb. Nails should not be depended on for important joints, even in temporary structures. Large cut or wire nails are known as SPIKES. Boat or ship spikes are also made, forged from square bars with square head and chisel point. R R track spikes have a special head.

Drift bolts are long, steel pins, generally round (square are used), with or without heads. ¾ and ⅞-in diam are common sizes, but 1-in diam and up to 30 in long have been used. A hole should be bored 1/16 to ⅛ in less diam than bolt.

Treenails (wooden pins) are hard-wood pins, sometimes used as drift bolts, but commonly in mortise and tenon joints, etc. **DOWELS** are wood or iron pins extending into

but not through, 2 members, to connect them; they have neither point nor head. In heavy work, they are usually 0.75 to 2 in diam and 5 to 12 in long.

Screws comprise wood and lag screws. Wood screws are made to 6 in long, and some lengths in as many as 18 different diams. Thus, 3-in screws are from 6 to 26 gage (diam); 3.5-in, 8 to 26; 4-in, 8 to 30; and 4.5, 5, and 6-in, 12 to 30. Gage refers to special screw-gage, which is approx as follows: 6 = 0.14 in, 10 = 0.19, 15 = 0.26, 20 = 0.32, 25 = 0.39, and 30 = 0.45 in. For all large screws a hole should be bored less than diam of shank and about 0.5 its depth. For lag screws bore a hole equal in diam and length to shank, and then a smaller hole for about 0.5 length of thread. Lag screws have square or hexagonal heads, and are from 5/16 to 7/8-in diam for 3-in length; 5/16 to 1-in for 3 1/2 to 6-in length: 7/16 to 1 for 6.5 to 9-in; 0.5 to 1 for 10 to 12-in. Lengths vary by 0.5 in up to 8 in.

Bolts have round, square or hexagon heads, and are either rough or finished, rough being used for woodwork. Where used to prevent sliding of one member over another, the hole should be bored with a diam of 1/16 in less than the bolt, to give a close driving fit. Bolts have diam differing by 1/16 or 1/8-in. Where subject to vibration, lock nuts are used, or a nail is driven alongside the nut, or the thread may be "burred" to prevent nut working loose. (See Sec 41, Table 22.)

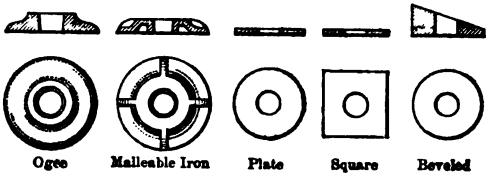


Fig 56. Washers

Washers (Fig 56) are important in timber work, to provide sufficient bearing area so that compression on surface of the timber may not exceed a safe value. They are of C I, steel or W I, and malleable iron. Common C-I washer is the Ogee (marked O G) form. Special washers can be cast when the number required warrants it. Steel and W-I washers are cut from plate, and are often too small and thin properly to distribute the bearing load. They should be about 4 times the diam of the bolt and 0.5 as thick.

Table 15. C-I Ogee Washers (dimensions, in)

Diam of bolt	Diam of washer	Thick-ness	Wt of 100	Bearing area	Diam of bolt	Diam of washer	Thick-ness	Wt of 100	Bearing area
3/8	1 1/2	5/16	8.5	1.57	1	4	7/8	180	11 57
1/4	2	3/8	22	2.83	1	4	1	194	11 57
1/2	2 3/8	1/2	37.5	4.12	1 1/8	4 1/2	1	215	14 68
5/8	2 1/2	1/2	45	4.47	1 1/8	4 1/2	1 1/8	295	14 68
5/8	3	5/8	75	6.63	1 1/4	5	1 1/8	320	18.15
3/4	3	5/8	72	6.47	1 1/4	5 3/8	1 1/4	469	21 21
3/4	3 1/4	3/4	100	7.69	1 3/8	5 1/2	1 1/4	425	21.99
7/8	3 1/2	3/4	115	8.84	1 1/2	6	1 3/8	525	26.20
7/8	3 5/8	7/8	150	9.54	1 1/2	6	1 1/2	688	26.20

Anchor bolts have a thread cut at upper end to take a nut, lower end being embedded in the foundation. Neat cement grout has been shown to be best material for setting the bolt; if there is no vibration, plain rods may be used, proportioned for 15 000 lb per net section of bolt, or 100 to 150 lb per sq in of embedded surface of bolt. Embedment of about 20 diam is needed to develop full strength of bolt. For machine foundations, it is best to use a bolt with nut and large washer at lower end and build it into the concrete.

Resistance to pulling of nails and screws varies with character of wood, depth of penetration, and for screws with size of hole used. Cut nails have higher holding power than wire, and the safe load may be taken at 4d in yellow pine and 7d in oak, where d = penny number of nail, driven across grain, to depth of 0.4 length; for wire nails, 2.5 and 4d. If subject to shock, the resistance is very low. For wood screws the safe resistance of a No 20-gage screw is twice that for a nail of same length. Lag screws show an ultimate resistance varying from 400 for white pine to 1 400 or more lb per sq in for oak; area being

length (exclusive of point) times circumference of screw. Safe lateral resistance of nails is roughly twice the resistance to pulling.

Timber joints. (Fig 57, 58.) Butt joint is effective for compression members, the ends being secured by other connecting members or by fish plates. Notching is widely used, especially for cribs; sometimes called GAINING or "dapping" or "sizing" when used in heavy framing to allow for inequalities in dimensions of timbers. HALVING is common in frame building construction, to join sills and wall plates. MORTISE AND TENON is a favorite joint for strength and rigidity. For 10 to 12-in timbers treenails should be 1 to 1.5 in diam, with a taper of 1/8 to 1/4 in. Hole in tenon should be about 1/4 the length of tenon from shoulder, and slightly nearer the shoulder than that in mortise, so the pin will draw the timbers together. STEPPING is common in all kinds of framing. Metal straps are sometimes used in framing structures subject to shock. Metal plates are used in heavy construction, and for joints in structures like derricks, which are often taken down and set up again. Beam hangers and column caps are used in heavy floor construction. Lap joint is simple, but only used for connections in minor members, due to its eccentricity and consequent bending on pins, etc. Half-lap scarf is widely used for joining members carrying small stresses, as guard rails on bridges, sills in buildings. Oblique scarf has more than double the strength of the half lap. Both joints are often used with fish plates in tension and compression members subjected to bending. Tabled scarf with key will take some tension, and when the amount is known the tables are easily designed for shear. Fish-plate joints, with wood or steel plates, are effective for tension. Bolts should be brought to solid bearing, to prevent timbers from buckling and to exclude moisture. Safe bearing loads per lin in are for safe bearing of 1 000 and 100 lb per sq in respectively. For other unit stresses, safe loads are obtained by proportion. For bending, the bolt is computed as a restrained beam with fiber stress of 22 500 lb per sq in (Table 16).

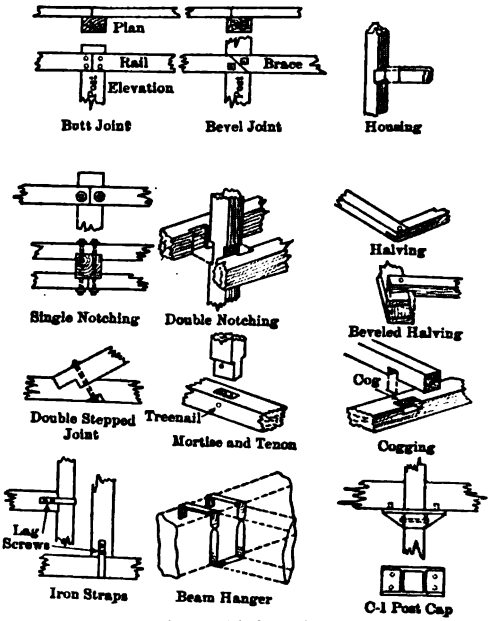


Fig 57. Timber Joints

Design of a plain fish-plate joint (Fig 58). A tension timber to be spliced is 2 by 8 in longleaf yellow pine. Allowable stress, 1 200 lb per sq in net section. 2 fish plates of same material will have to be 1 in thick each. Number and size of bolts are found from Table 16. Thus, try 0.75-in bolts. Safe end bearing for the timber is 1 400 lb per sq in. Hence, safe bearing load on 0.75-in bolt = 750 × 2 (= length of bearing) × 1 400 ÷ 1 000 = 2 100 lb. For bending, safe load = 3 720 lb; hence, bearing controls. Area of net section of member = 2 × (8 − 2 × 7/8) = 12.5 sq in, assuming that bolts are placed in pairs and holes are 1/8 in larger than bolt. Total tensile load = 12.5 × 1 200 = 15 000 lb, and number of 0.75-in bolts required = 15 000 ÷ 2 100 = 7. As an even number of

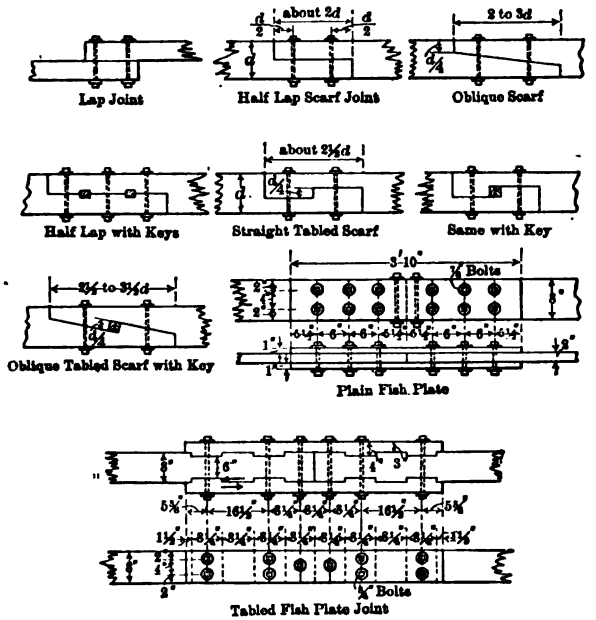


Fig 58. Lap and Scarf Joints

Table 16. Strength of Bolts

Diam of bolt, in	Safe bearing, lb per lin in		Total safe uniform bending load on bolt, lb. a = thickness of member, b = thickness of fish plates					
	End grain	Side grain	$a=2$ $b=1$	$a=3$ $b=1.5$	$a=4$ $b=2$	$a=6$ $b=3$	$a=8$ $b=4$	$a=10$ $b=5$
½	500	50	1 100	740	550	370	280	220
⅝	625	63	2 160	1 440	1 080	720	540	430
¾	750	75	3 720	2 490	1 860	1 250	930	750
⅞	875	88	5 920	3 940	2 960	1 970	1 470	1 180
1	1 000	100	8 820	5 880	4 410	2 940	2 210	1 760
1⅛	1 125	113	12 580	8 390	6 290	4 190	3 140	2 510
1	1 250	125	17 250	11 500	8 640	5 750	4 300	3 450

olts makes a better design, 6 ⅞-in bolts will be used. Length of plate is determined by pacing of bolts, which tend to shear out a portion of the plate between planes tangent o top and bottom of bolt. Round bolts also tend to split the plates, so only 80% of safe hear with grain is used, or 80% of 150 lb per sq in = 120 (Table 10). Hence, distance etween bolt holes should be not less than 14 400 (= total load on net section for ⅞-in olts) ÷ [6 (= number of bolts) × 2 (= thickness of 2 fish plates) × 2 (double shear) × 120 (shearing stress)] = 5 in, or, adding 1 in for bolt hole, = 6 in c to c and 5.5 for ends. The TABLED FISH-PLATE JOINT in Fig 58 was designed for a load of 32 tons in longleaf yellow ine. The design is as follows, using for tension 1 800, for compression on end of grain 1 000, and shear along grain, 250 lb per sq in. a. Find areas required for tension, compression and shear, or 64 000 ÷ 1 800 = 35.6, 64 000 ÷ 2 000 = 32, and 64 000 ÷ 250

= 256 sq in, respectively. 4 tables are usually effective, and the shearing area of each would be 256 ÷ 4 = 64 sq in. In order that the length of table shall be not less than 4 its width (representing good proportions), the width of timber must be about 8 in, and the table a little over 8 in long to allow for bolt holes. Depth of tables must be 32 ÷ (4 × 8) = 1 in, to furnish necessary bearing area at their ends. Assuming ¾-in bolts and ⅞-in holes, the net depth of main member at tension must be 35.6 ÷ (8 − 2 × ⅞) = 5.7 in, making total depth of timber 8 + 5.7 = 13.7 in, or an 8 × 8 is needed. The fish plates, each taking 0.5 the total load and being of same material as main member, should be 0.5 as thick. Size of bolts may be roughly checked. Unlike the plain fish-plate joint, they transmit stress by tension. Thus the pull on fish plate is approx through its center (see arrows in cut), and this is transmitted to the main member by end bearing on the tables, approx at their centers. This causes a moment of (6 400 ÷ 4) × (1.5 + 0.5) = 32 000 in lb per table, tending to cause it to rotate about its edge. This is resisted by tension in the bolt and compression on face of table. The lever arm of this couple may be taken as 0.5 the length of table = 4 in, giving a pull on the bolt of 4 000 lb, assuming 2 bolts per table. A 0.75-in bolt is ample for this stress, using 15 000 lb per sq in, net section. The washers should have sufficient area for bearing, using 550 lb per sq in. A standard 3.25 C-I washer has an area of 7.69 sq in, and will transmit a total stress of 4 230 lb, which is OK. Bolts should be pulled up to a tight bearing.

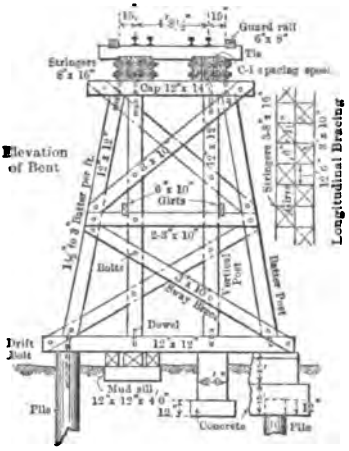


Fig 59. R R Trestle Bent

23. Trestles, Trusses and Bridges

Trestles consist of posts framed together to form bents, supporting a floor system of stringers, ties, and guard rails.

They are used for semi-permanent work, to avoid expenditure for permanent bridge or fill construction in the early days of a new R R, for making large fills, and in connection with mine dumps. Fig 59 shows a trestle bent, with different types of foundations used indicated. Where piles are used they may be long enough to form the posts (pile-bent), being sway-braced as for a framed bent. Bents are built in stories of 15 or more ft high. Fig 59 is a 2-story bent. Posts may be designed as columns, but are made larger than computations require to allow for decay. Stringers should be continuous over 2 spans, and are designed for bending stresses. Bents are spaced 12 to 16 ft apart, depending upon wt of loading. For light mine cars, the construction would be much lighter than that shown, which is for R R work.

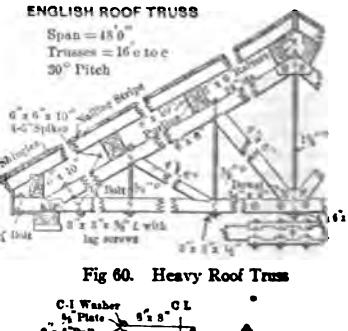


Fig 60. Heavy Roof Truss

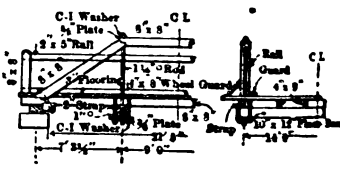


Fig 61. Small Highway Bridge

Roof truss of heavy design is shown in detail by Fig 60. Load from the rafters is rried to the panel points by the purlins; the stresses may be found (Art 17) for any sumed loading, and the members and details designed accordingly.

Small highway bridge, consisting of a queen truss with plank floor, is shown in Fig 61.

24. Frame Buildings

Framing is of 3 kinds: (*a*) MORTISE AND TENON FRAME, in which the members re mortised and pinned together. This requires a large amount of labor and r light buildings has been practically abandoned; (*b*) BALLOON FRAME, com- osed of lighter members, nailed together, and more quickly erected at lower ost; (*c*) COMBINATION FRAME, in which main iembers are mortised as in (*a*) and the stud- ing and bracing is nailed, used in the best odern construction.

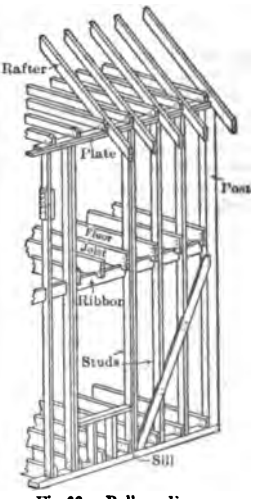


Fig 62. Balloon Frame

Fig 62 shows typical balloon frame for a 2-story ouse. Computations are rarely necessary. Sill is ually 4 by 6 in, halved at corners, on foundation alls or piers. Floor joists are of 2-in plank, 12 or in c to c; for joists over 12 in deep, 3-in thickness ould be used. Joists are braced laterally by cross idging, 1 row for 8 to 16-ft spans, and 2 for 18 to -ft spans, with 1" by 3-in bridging, or 2 by 3 for -in beams, nailed with 2 10d nails. Corner posts, ually 4 by 4 or 4 by 6-in, are set up and temporarily cured by "stay lath." Studs are held by a tempo- ry board nailed against inside. Studs are 2 by 4 . usually 16-in centers; they are spiked to the sills lower ends, and when practicable studs and joists e placed against and spiked to each other. Studs tend from the sill up to the plate, the second floor ists being supported on 1 by 7-in ledger boards, otched into the studs. The plate is usually of 2 2 by -in pieces. Joints in studs are made by nailing a sh plate on each side. Studding is doubled at indow and door openings. 1 by 6-in boards, otched into sill, post and studs, are used for corner aces. Rafters, 2 by 6 in, for lengths of 12 ft, 2 by to 18, 2 by 10 over 18 ft, are spaced 16 to 24 in cen- rs. Combination frame posts are usually 4 by 8 in, and the ledger board is replaced by 4 by 8-in girt, framed into the posts, the studding being separate for each story.

Partitions are usually of 2 by 4-in studding, spaced 12 to 16-in centers. In framing, should be remembered that timber shrinks but little endwise of the grain, and some- mes as much as 0.5 in per ft crosswise. Where settlement is to be avoided, all horiz iembers should be eliminated from vertical supports of the buildings. Partitions should st on the cap of the partition below when possible (not on the floor), or on a sole piece.

Covering. For cheap construction ship-lap is often placed vertically, with battens over e cracks. Beveled siding is commonly 6 in wide and laid 4.5 in to the weather. Molded ding (Novelty siding), is of ⅞-in boards, with ⅝-in lap. In best work, the frame is first overed with 1-in diagonal sheathing, which in turn is covered with building paper before e siding is placed. Shingles or corrugated sheets are also used for siding.

Shingle roof. Shingles vary from 16 to 24 in long, laid with 4.5 to 5.5 in (a little less an ½ the length) to weather. 4 bundles usually contain a "thousand" shingles, that , the equivalent of 1 M shingles 4 in wide, the widths varying from 3 to 14 in. Table 17 ives number of standard 4-in shingles for 100 sq ft of roof. Cost laid, $4.50 to $5 per quare (= 100 sq ft).

Tin roofing. There are 2 common sizes of tin, 14 by 20 in and 20 by 28 in, latter ' ' ost used. Thicknesses most used are I C (No 29 gage) and IX (No 27). Valu-

taken as 6 000 lb per sq in; for malleable iron, 15 000 to 20 000; for C I in compression, 20 000. Ultimate tensile strength of C I is from 16 000 to 30 000, depending on grade etc, some tests showing only 9 200, others up to 46 450; malleable iron averages 43 000. Compression tests show 44 500 to 215 000 lb per sq in, general aver being about 80 000.

Working stresses. CAST IRON is generally used for parts requiring no computation, or for compression members transmitting 10 to 15 000 lb per sq in. Tensile stresses are 1 000 to 4 000, depending on character of load and design. Columns are hollow, of round or square section. The straight-line formula, p (lb per sq in) $= 30\,500 - 160\,l \div d$, where l = length and d = least diam or side, in, represents the aver ultimate strength, and a safety factor of 4 to 6 should be used.

Wrought iron is tough, ductile, malleable, and easily welded, but can not be tempered. Steel has superseded it for bridge and building construction.

It is sold as MERCHANT BAR for blacksmithing and working into wrought shapes. It is said to resist corrosion better than soft steel, which is often sold for it. It may be distinguished from steel by etching for 15 or 20 min (in 9 parts water, 3 H_2SO_4 and 1 HCl); fibers will show up plainly, while steel dissolves uniformly. Principal defects are rough edges, due to poor rolling, spilly or spongy places, caused by imperfect puddling, blisters and excess of slag. Good W I should bend 180° cold around a diam 1 to 2 times the thickness of the test piece; it should show an ultimate strength of at least 46 000 lb per sq in, elastic limit of 25 000, and an elongation in 8 in of at least 20%. Fracture should show fibrous structure, though it may be made to show a coarse crystalline fracture by notching a tension bar all around and breaking suddenly.

Aver W I will test as follows: ultimate strength, 50 000 lb; elastic limit, 30 000 lb; coeff of elasticity, 28 000 000; stretch in 8 in, 20%, reduction area, 30%. It has no true ultimate compresive strength or modulus of rupture, but continues to deflect or compress without breaking. Aver values are taken as 55 000 and 50 000 lb per sq in. Usual working stresses are: tension, 12 000; compression, 12 000; shear, 7 500; bending, 12 000; torsion, 4 000 lb per sq in.

Steel, with a chemical composition intermediate between C I and W I, is produced either by adding carbon to W I or partly removing it from C I. Here it may be divided into 3 classes: soft steel, below 0.15% C; medium, 0.15 to 0.3% C; hard, over 0.3 % C. Soft steel is good for boiler plate, or wherever high ductility is advantageous; medium grades are used for structural purposes; hard grades are valuable for axles, shafts, and service demanding good wearing surface.

Carbon is the controlling element in steel, though other elements affect its strength to a smaller degree. Increase of C decreases ductility and increases strength. H. G. Colbell proposed:

For acid open-hearth steel, $T = 40\,000 + 68\,000\,C + 100\,000\,P + 80\,000\,CM$; for basic open-hearth steel, $T = 38\,800 + 65\,000\,C + 100\,000\,P + 9\,000\,M + 40\,000\,(CM)$; where C = % carbon, P = phosphorus, M = manganese, and T = tensile strength, lb per sq in. Table 19 shows the properties of different grades of carbon steel. Aver shear =

Table 19. Strength of Carbon Steel (after W. L. Turner)

Heat treatment	Kind of steel	Carbon, %	Manganese, %	Elastic limit, lb per sq in	Tensile strength, lb per sq in	Elongation in 2 in, %	Red of area,
R...	Wrought iron.....	0.05	0.05	32 020	49 450	42 5	53 8
R...	Old plate.....	0.24	0.42	58 160	70 840	25 5	33 4
A...	Mild steel.....	0.18	0.40	39 460	60 650	35 0	62 2
OT..	" "	0.18	0.40	49 390	70 036	33 0	64 2
A...	Forging steel.....	0.26	0.28	43 010	61 850	34 0	63 4
OT..	" "	0.26	0.28	52 230	77 310	28 0	65 3
A...	Steel casting.....	0.18	0.65	34 690	58 800	28 0	44 0
A...	Spring steel.....	1.00	0 30	63 800	125 000	8 5	15 2
OT..	" "	1.00	0 30	101 000	186 300	9 5	16 1

r, untreated. *A* = Annealed. *OT* = Oil-tempered.

rength of medium steel is 50 000 lb per sq in. In compression, structural steel acts like I. In general, elastic limit and ultimate strength are same as for tension. Coeff of sticity varies little with the carbon content and averages 30 000 000 for tension and 000 000 in shear.

Specifications (Am Soc Test Materials, 1909) require that for bridge work steel must be ade by open-hearth process, for buildings either the Bessemer or open-hearth, and for ret steel or plates over 0.75 in thick (which are to be punched) the open-hearth. A riation of 4 000 lb per sq in in tensile strength is allowed for structural steel for bridges. r material over 0.75 in thick a deduction of 1% in percentage elongation in 8 in shall made for each increase of 1/8 in over 0.75; also, for material under 5/16 in, a deduction of 5 shall be made for each decrease of 1/16 in under 5/16. Bend-test specimens are usually by 0.5 in section; $d = 3t$, etc, indicates bending around a diam 3 times thickness of bar. least 1 tension and 1 bend test are required from each melt, and if material differs 3/8 in more in thickness, separate tests are made from both thickest and thinnest metal rolled. Variation in cross-sec or wt of 2.5% from that specified is allowed, except for plates dered by wt. If these are over 12.5 lb per sq ft, a 2.5% variation is allowed, if under 100 wide, and 5% if over; for plates over 12.5 lb and under 75 in wide 2.5% is allowed; be- een 75 and 100 in, 3% below and 5% above; over 100 in, 3% below and 10% above. hen ordered by gage, thickness of plate shall not be over 0.01 in less than that ordered.

Inspection and defects comprise: mill inspection, inspection of rolled steel, and shop spection. In mill inspection steel ingots are examined to discover such defects as seg- gation, pipes and cracks, and blow holes; the first being generally found in upper part ingot, due to gathering together of certain chemical constituents, and the others due to o rapid cooling of outside of ingot. Good steel should show a bluish grey color on frac- re, with uniform grain, slight luster and silky appearance. In inspecting rolled steel, e chief points are the size and freedom from pipes, stars, pits, laps, seams, etc, caused by ove-mentioned defects in the ingot, and the failure to discard a sufficient portion of top ingot, where they generally occur, to assure sound metal. Shop inspection is equally portant, covering: straightening, marking and punching, assembling, reaming, riveting, cing, boring, finishing, and painting. Punch dies should be sharp and not over 1/16 in ater diam than the punch. Drifting is allowable only to bring parts together, and ould not be permitted for enlarging or matching up rivet holes. Facing and boring ould be carefully done and at proper angle to axis of the piece.

Alloy steels. NICKEL is used as an alloy for steel for guns, armor plate, and in some ructural work. Structural nickel steel has an elastic limit about 15% and ultimate rength about 25% higher than common structural steel, but nickel seems to decrease sistance to dynamic stresses. An alloy containing 36% nickel ("invar" steel) has a eff of expansion of about 4% that of steel; used for tapes and scientific instruments. HROME-NICKEL and CHROME-VANADIUM STEEL alloys have high elastic and ultimate rength, which, like most steel, may be greatly increased by tempering, at a sacrifice, how- er, of ductility (elongation and reduction of area). They are used in automobile con- ruction, machine parts, locomotive forgings, shafts, axles, etc. Manganese steel con- ining from 6 to 12% manganese is very tough and malleable and is used for frogs and vitches, crusher rolls and teeth for steam-shovel dippers. TUNGSTEN STEEL is used for ols for cutting metal and some varieties will cut when red hot.

26. Tables and Specifications

Standard rolled shapes. Following tables give the elements of the sections common use. Most of those listed may be termed STANDARD, in the sense at they are gen- rally kept ready or quick shipment the steel com- anies' ware- ouses. Beams re designated by epth and wt per t. Thus 15-in *I* 45 indicates an

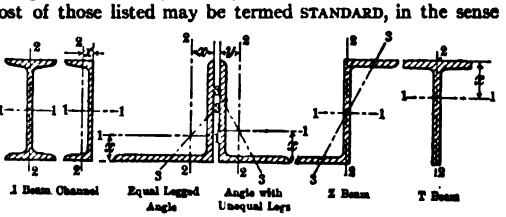


Fig 63. Standard Rolled Shapes

I-beam 15 in deep, weighing 45 lb per lin ft. Wt of rails is in lb per yd. Beams are cut in proper lengths and used singly, or combined by riveting with plates etc, as structural units.

Tables. (For completer tables see Bib 18, also Bethlehem Steel Co's Handbook.) Fig 63 shows cross-sec of the beams listed in Tables 20–25. Fig 64 shows other rolled sections, the H-beam being largely used for columns in building construction. The various factors given in the tables are used in computations and design. SECTION MODULUS is used in obtaining the fiber stress in beam (Art 4), or in selecting a beam for a given load. Coeff of strength (Art 4) gives a simple method of determining the safe load a beam will carry, as it equals 8 M, and for a uniform load = $\frac{8M}{l}$, where W = total load and l = span. Thus, the safe total uniform load on a 20-in I @ 65 lb for a 20-ft span, would be 1 247 600, the corresponding coeff, divided by 20 or 62 380 lb. Again, dividing by 20 gives a safe load of 3 119 lb per lin ft; or, dividing by 2, a safe concentrated center load of 31 190 lb. An approx rule, useful in absence of tables, is for standard I-beams the coeff of strength in ft ton = 0.5 the product of the depth times the wt per ft. Thus, for a 20-in I @ 65 C in ft-ton = about 650, or in ft-lb, about 1 300 000, against the actual value of 1 247 600.

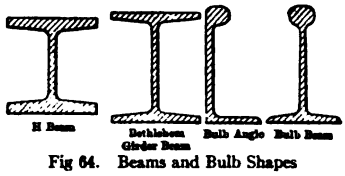


Fig 64. Beams and Bulb Shapes

Table 20. Elements of Selected Equal Angles (Carnegie)

Size, in	Wt per ft, lb	Area of sec, in²	Axis 1–1 and axis 2–2				Axis 3–3
			I in⁴	r in	S in³	z in	r=min in
8 ×8 ×1	51.0	15.00	89.0	2.44	15.8	2.37	1.56
8 ×8 ×⅞	45.0	13.23	79.6	2.45	14.0	2.32	1.58
8 ×8 ×¾	38.9	11.44	69.7	2.47	12.2	2.28	1.57
8 ×8 ×⅝	32.7	9.61	59.4	2.49	10.3	2.23	1.58
8 ×8 ×½	26.4	7.75	48.6	2.51	8.4	2.19	1.58
6 ×6 ×1	37.4	11.00	35.5	1.80	8.6	1.86	1.16
6 ×6 ×⅞	33.1	9.73	31.9	1.81	7.6	1.82	1.17
6 ×6 ×¾	28.7	8.44	28.2	1.83	6.7	1.78	1.17
6 ×6 ×⅝	24.2	7.11	24.2	1.84	5.7	1.73	1.17
6 ×6 ×½	19.6	5.75	19.9	1.86	4.6	1.68	1.18
6 ×6 ×⅜	14.9	4.36	15.4	1.88	3.5	1.64	1.19
4 ×4 ×¾	18.5	5.44	7.7	1.19	2.8	1.27	0.77
4 ×4 ×⅝	15.7	4.61	6.7	1.20	2.4	1.23	0.77
4 ×4 ×½	12.8	3.75	5.6	1.22	2.0	1.18	0.78
4 ×4 ×⅜	9.8	2.86	4.4	1.23	1.5	1.14	0.79
3½×3½×⅜	13.6	3.98	4.3	1.04	1.8	1.10	0.68
3½×3½×½	11.1	3.25	3.6	1.06	1.5	1.06	0.68
3½×3½×⅜	8.5	2.48	2.9	1.07	1.2	1.01	0.69
3 ×3 ×½	9.4	2.75	2.2	0.90	1.1	0.93	0.58
3 ×3 ×⅜	7.2	2.11	1.8	0.91	0.83	0.89	0.58
3 ×3 ×¼	4.9	1.44	1.2	0.93	0.58	0.84	0.59
2½×2½×⅜	5.9	1.73	0.98	0.75	0.57	0.76	0.48
2½×2½×¼	4.1	1.19	0.70	0.77	0.39	0.72	0.49
2 ×2 ×⅜	4.7	1.36	0.48	0.59	0.35	0.64	0.39
2 ×2 ×¼	3.19	0.94	0.35	0.61	0.25	0.59	0.39
1½×1½×¼	2.34	0.69	0.14	0.45	0.13	0.47	0.29
1½×1½×⅛	1.23	0.36	0.08	0.46	0.07	0.48	0.30

Table 21. Elements of Selected Unequal Angles (Carnegie)

Size, in	Wt per ft, lb	Area of sec, in²	Axis 1-1				Axis 2-2				Axis 3-3, r = min in
			I in⁴	r in	S in³	z in	I in⁴	r in	S in³	y in	
×4 ×⅞	27.2	7.98	27.7	1.86	7.2	2.12	9.8	1.11	3.4	1.12	0.86
×4 ×¾	23.6	6.94	24.5	1.88	6.2	2.08	8.7	1.12	3.0	1.08	0.86
×4 ×⅝	20.0	5.86	21.1	1.90	5.3	2.03	7.5	1.13	2.5	1.03	0.86
×4 ×½	16.2	4.75	17.4	1.91	4.3	1.99	6.3	1.15	2.1	0.99	0.88
×4 ×⅜	12.3	3.61	13.5	1.93	3.3	1.94	4.9	1.17	1.6	0.94	0.88
×3½×⅞	25.7	7.55	26.4	1.87	7.0	2.22	6.6	0.93	2.6	0.97	0.75
×3½×¾	22.4	6.56	23.3	1.89	6.1	2.18	5.8	0.94	2.3	0.93	0.75
×3½×⅝	18.9	5.55	20.1	1.90	5.2	2.13	5.1	0.96	1.9	0.88	0.75
×3½×½	15.3	4.50	16.6	1.92	4.2	2.08	4.3	0.97	1.6	0.83	0.76
×3½×⅜	11.7	3.42	12.9	1.94	3.3	2.04	3.3	0.99	1.2	0.78	0.77
×3½×¾	19.8	5.81	13.9	1.55	4.3	1.75	5.6	0.98	2.2	1.00	0.75
×3½×⅝	16.8	4.92	12.0	1.56	3.7	1.70	4.8	0.99	1.9	0.95	0.75
×3½×½	13.6	4.00	10.0	1.58	3.0	1.66	4.0	1.01	1.6	0.91	0.75
×3½×⅜	10.4	3.05	7.8	1.60	2.3	1.61	3.2	1.02	1.2	0.86	0.76
×3 ×⅝	15.7	4.61	11.4	1.57	3.5	1.80	3.1	0.81	1.4	0.80	0.64
×3 ×½	12.8	3.75	9.5	1.59	2.9	1.75	2.6	0.83	1.1	0.75	0.65
×3 ×⅜	9.8	2.86	7.4	1.61	2.2	1.70	2.0	0.84	0.89	0.70	0.65
×3 ×⅜	13.6	3.98	6.0	1.23	2.3	1.37	2.9	0.85	1.4	0.87	0.64
×3 ×½	11.1	3.25	5.0	1.25	1.9	1.33	2.4	0.86	1.1	0.83	0.64
×3 ×⅜	8.5	2.48	4.0	1.26	1.5	1.28	1.9	0.88	0.87	0.87	0.64
½×3 ×⅝	12.5	3.67	4.1	1.06	1.8	1.17	2.8	0.87	1.3	0.92	0.62
½×3 ×½	10.2	3.00	3.5	1.07	1.5	1.13	2.3	0.88	1.1	0.88	0.62
½×3 ×⅜	7.9	2.30	2.7	1.09	1.1	1.08	1.8	0.90	0.85	0.83	0.62
½×2½×½	9.4	2.75	3.2	1.09	1.4	1.20	1.4	0.70	0.76	0.70	0.53
½×2½×⅜	7.2	2.11	2.6	1.10	1.1	1.16	1.1	0.72	0.59	0.66	0.54
½×2½×¼	4.9	1.44	1.8	1.12	0.75	1.11	0.78	0.74	0.41	0.61	0.54
×2½×⅜	6.6	1.92	1.7	0.93	0.81	0.96	1.0	0.74	0.58	0.71	0.52
×2½×¼	4.5	1.31	1.2	0.95	0.56	0.91	0.74	0.75	0.40	0.66	0.53
½×2 ×⅜	5.3	1.55	0.91	0.77	0.55	0.83	0.51	0.58	0.36	0.58	0.42
½×2 ×¼	3.62	1.06	0.65	0.78	0.38	0.79	0.37	0.59	0.25	0.54	0.42

Table 22. Elements of Selected Equal Tees (Carnegie)

Size				Wt per ft, lb	Area of sec, in²	Axis 1-1				Axis 2-2		
Flange, in	Stem, in	Min thickness				I in⁴	r in	S in³	z in	I in⁴	r in	S in³
		Flange, in	Stem, in									
4	4	½	½	13.5	3.97	5.7	1.20	2.0	1.18	2.8	0.84	1.4
3½	3½	½	½	11.7	3.44	3.7	1.04	1.5	1.05	1.9	0.74	1.1
3	3	½	½	9.9	2.91	2.3	0.88	1.1	0.93	1.2	0.64	0.80
3	3	⅜	⅜	7.8	2.27	1.8	0.90	0.86	0.88	0.90	0.63	0.60
2½	2½	⅜	⅜	6.4	1.87	1.0	0.74	0.59	0.76	0.52	0.53	0.42
2¼	2¼	¼	¼	4.1	1.19	0.52	0.66	0.32	0.65	0.25	0.46	0.22
2	2	¼	¼	3.56	1.05	0.37	0.59	0.26	0.59	0.18	0.42	0.18
1¾	1¾	¼	¼	3.09	1.91	0.23	0.51	0.19	0.54	0.12	0.37	0.14
1½	1½	¼	¼	2.47	0.73	0.15	0.45	0.14	0.47	0.08	0.32	0.10
1¼	1¼	¼	¼	2.02	0.59	0.08	0.37	0.10	0.40	0.05	0.28	0.07
1	1	⅛	⅛	0.89	0.26	0.02	0.30	0.03	0.29	0.01	0.21	0.02

Table 23. Elements of Selected Channels (Carnegie)

Depth, in	Wt per ft, lb	Area of sec, in²	Width of flange, in	Thickness of web, in	Axis 1-1			Axis 2-2			z in	Coeff of strength for $k=16\,000$ lb axis 1-1 ft-lb
					I in⁴	r in	S in³	I in⁴	r in	S in³		
15	55.0	16.18	3.818	0.818	430.2	5.16	57.4	12.2	0.87	4.1	0.82	611 900
	45.0	13.24	3.622	0.622	375.1	5.32	50.0	10.3	0.88	3.6	0.79	533 500
	33.0	9.90	3.400	0.400	312.6	5.62	41.7	8.2	0.91	3.2	0.79	444 500
12	40.0	11.76	3.418	0.758	196.9	4.09	32.8	6.6	0.75	2.5	0.72	450 200
	30.0	8.82	3.173	0.513	161.7	4.28	26.9	5.2	0.77	2.1	0.68	287 400
	20.5	6.03	2.940	0.280	128.1	4.61	21.4	3.9	0.81	1.7	0.70	227 800
10	35.0	10.29	3.183	0.823	115.5	3.35	23.1	4.7	0.67	1.9	0.70	246 400
	25.0	7.35	2.889	0.529	91.0	3.52	18.2	3.4	0.68	1.5	0.62	194 100
	15.0	4.46	2.600	0.240	66.9	3.87	13.4	2.3	0.72	1.2	0.64	142 700
9	25.0	7.35	2.815	0.615	70.7	3.10	15.7	3.0	0.64	1.4	0.62	167 600
	13.25	3.89	2.430	0.230	47.3	3.49	10.5	1.8	0.67	0.97	0.61	112 200
8	21.25	6.25	2.622	0.582	47.8	2.77	11.9	2.3	0.60	1.1	0.59	127 400
	16.25	4.78	2.439	0.399	39.9	2.89	10.0	1.8	0.61	0.95	0.56	106 400
	11.25	3.35	2.260	0.220	32.3	3.11	8.1	1.3	0.63	0.79	0.58	86 100
7	19.75	5.81	2.513	0.633	33.2	2.39	9.5	1.9	0.56	0.96	0.58	101 100
	14.75	4.34	2.303	0.423	27.2	2.50	7.8	1.4	0.57	0.79	0.54	82 800
	9.75	2.85	2.090	0.210	21.1	2.72	6.0	0.98	0.59	0.63	0.55	66 800
6	15.5	4.56	2.383	0.563	19.5	2.07	6.5	1.3	0.53	0.74	0.55	69 500
	8.0	2.38	1.920	0.200	13.0	2.34	4.3	0.70	0.54	0.50	0.52	46 200
5	11.5	3.38	2.037	0.477	10.4	1.75	4.2	0.82	0.49	0.54	0.51	44 400
	6.5	1.95	1.750	0.190	7.4	1.95	3.0	0.48	0.50	0.38	0.49	31 600
4	7.25	2.13	1.725	0.325	4.6	1.46	2.3	0.44	0.46	0.35	0.46	24 400
	5.25	1.55	1.580	0.180	3.8	1.56	1.9	0.32	0.45	0.29	0.46	20 200
3	6.0	1.76	1.602	0.362	2.1	1.08	1.4	0.31	0.42	0.27	0.46	14 700
	4.0	1.19	1.410	0.170	1.6	1.17	1.1	0.20	0.41	0.21	0.44	11 600

Table 24. Elements of Selected Zees (Carnegie)

Size			Wt per ft, lb	Area of sec, in²	Axis 1-1			Axis 2-2			Axis [illegible] $r=$ [illegible] in
Depth, in	Flange, in	Thickness, in			I in⁴	r in	S in³	I in⁴	r in	S in³	
6	3½	⅝	29.4	8.63	42.1	2.21	14.0	15.4	1.34	4.9	[illegible]
6	3½	9/16	22.8	6.68	34.6	2.28	11.5	12.6	1.37	3.9	[illegible]
6	3½	⅜	15.7	4.59	25.3	2.35	8.4	9.1	1.41	2.8	[illegible]
5	3¼	½	17.9	5.25	19.2	1.91	7.7	9.1	1.31	3.0	[illegible]
5	3¼	7/16	11.6	3.40	13.4	1.98	5.3	6.2	1.35	2.0	[illegible]
4	3⅛	⅝	18.9	5.55	12.1	1.48	6.1	8.7	1.25	3.2	[illegible]
4	3⅛	11/16	13.8	4.05	9.7	1.55	4.8	6.7	1.29	2.4	[illegible]
4	3⅛	¼	8.2	2.41	6.3	1.62	3.1	4.2	1.33	1.4	[illegible]
3	2 1/16	½	12.6	3.69	4.6	1.12	3.1	4.9	1.15	2.0	[illegible]
3	2 1/16	3/8	9.8	2.86	3.9	1.16	2.6	3.9	1.17	1.6	[illegible]
3	2 1/16	¼	6.7	1.97	2.9	1.21	1.9	2.8	1.19	1.1	[illegible]

Following is an outline of the Am Bridge Co's specifications. Unit stresses, lb per sq in:

Tension, net section, rolled steel 16 000

Direct compression, rolled steel and steel castings 16 000

Bending, on extreme fibers of rolled shapes, built sections, girders, and steel castings 16 000

Bending, on extreme fibers of pins	24 000
Shear on shop rivets and pins	12 000
Shear on bolts and field rivets	10 000
Shear (aver) on webs of plate girders and rolled beams, gross sec	10 000
Bearing press on shop rivets and pins	24 000
Bearing press on bolts and field rivets	20 000
Expansion rollers, press per lin in, not over 600 times diam of roller, in	
Axial compression on gross section of columns for $l + r$ up to 120 = 19 000 − 100 $l + r$, with a max of 13 000 (l = length of member, in, r = radius of gyration, in)	

Table 25. Elements of Selected I-Beams (Carnegie)

epth, in	Wt per ft, lb	Area of sec, sq in	Width of flange, in	Thickness of web, in	Axis 1−1			Axis 2−2			Coeff of strength for k = 16 000 lb per sq in, axis 1−1 ft-lb
					I in⁴	r in	S in³	I in⁴	r in	S in³	
24	110.0	32.48	7.938	0.688	2 883.5	9.42	240.3	81.0	1.58	20.4	2 563 200
	100.0	29.41	7.254	0.754	2 379.6	9.00	198 3	48.6	1.28	13.4	2 114 700
	90.0	26.47	7.131	0.631	2 238.4	9.20	186 5	45.7	1.31	12.8	1 989 200
	80.0	23.32	7.000	0.500	2 087.2	9.46	173.9	42.9	1.36	12.3	1 854 800
20	95.0	27.94	7.210	0.810	1 606.6	7.58	160.7	50.8	1.35	14.1	1 713 900
	85.0	25.00	7.063	0.663	1 508.5	7.77	150.9	47.3	1.37	13.4	1 609 300
	75.0	22.06	6.393	0.649	1 268.8	7.58	126.9	30.3	1.17	9.5	1 353 500
	65.0	19.08	6.250	0.500	1 169.5	7.83	117.0	27.9	1.21	8.9	1 247 600
18	85.0	25.00	7.163	0.725	1 220.7	6.99	135.6	50.0	1.42	14.0	1 446 400
	75.0	22.05	7.000	0.562	1 141.3	7.19	126.8	46.2	1.45	13.2	1 352 500
	65.0	19.12	6.177	0.637	881.5	6.79	97.9	23.5	1.11	7.6	1 044 800
	55.0	15.93	6.000	0.460	795.6	7.07	88.4	21.2	1.15	7.1	943 000
15	70.0	20.59	6.194	0.784	663.7	5.68	88.5	29.0	1.19	9.4	943 800
	60.0	17.67	6.000	0.590	609.0	5.87	81.2	26.0	1.21	8.7	866 100
	50.0	14.71	5.648	0.558	483.4	5.73	64.5	16.0	1 04	5.7	687 500
	42.0	12.48	5.500	0.410	441.8	5.95	58.9	14.6	1.08	5.3	628 300
12	50.0	14.71	5.489	0.699	303.4	4.54	50.6	16.1	1.05	5.9	539 200
	40.0	11.84	5.250	0.460	269.0	4.77	44.8	13.8	1 08	5 3	478 100
	31.5	9.26	5.000	0.350	215.8	4.83	36.0	9.5	1.01	3.8	383 700
10	40.0	11.76	5.099	0.749	158.7	3.67	31.7	9.5	0.90	3.7	338 500
	35.0	10.29	4.952	0.602	146.4	3.77	29.3	8.5	0 91	3.4	312 400
	25.0	7.37	4.660	0.310	122.1	4.07	24.4	6.9	0.97	3.0	260 500
9	35.0	10.29	4.772	0.732	111.8	3.29	24.8	7.3	0.84	3.1	265 000
	21.0	6.31	4.330	0.290	84.9	3.67	18.9	5.2	0.90	2.4	201 300
8	25.5	7.50	4.271	0.541	68.4	3.02	17.1	4.8	0.80	2.2	182 500
	18.0	5.33	4.000	0.270	56.9	3.27	14.2	3.8	0.84	1.9	151 700
7	20.0	5.88	3.868	0.458	42.2	2.68	12.1	3.2	0.74	1.7	128 600
	15.0	4.42	3.660	0.250	36.2	2.86	10.4	2.7	0.78	1.5	110 400
6	17.25	5.07	3.575	0.475	26.2	2.27	8.7	2.4	0.68	1.3	93 100
	12.25	3.61	3.330	0.230	21.8	2.46	7.3	1.9	0.72	1.1	77 100
5	14.75	4.34	3.294	0.504	15.2	1.87	6.1	1.7	0.63	1.0	64 600
	9.75	2.87	3.000	0.210	12.1	2.05	4.8	1.2	0.65	0.82	51 600
4	10.5	3.09	2.880	0.410	7.1	1.52	3.6	1.0	0.57	0.70	38 100
	7.5	2.21	2.660	0.190	6.0	1.64	3.0	0.77	0.59	0.58	31 800
3	7.5	2.21	2.521	0.361	2.9	1.15	1.9	0.60	0.52	0.48	20 700
	5.5	1.63	2.230	0.170	2.5	1.23	1.7	0.46	0.53	0.40	17 600

For structures carrying traveling machinery, as cranes, conveyors, etc., allowance is to be made for stresses resulting from such live load, to allow for impact.

Unsupported length of main compression members shall not exceed [illegible] gyration, or for secondary, 200. In getting cross-section of tension members [illegible] must be used, allowing for holes ⅛ in larger than rivets. In members subject to alternate stress, the greatest extreme fiber stress shall not exceed that given above. The unsupported length of beams and girders not to exceed 40 times width of flange.

Avoid adjustable members. No connection shall have less than 2 rivets. The ends of plates in girders shall not extend more than 8 times the thickness of the plate beyond the outermost row of rivets connecting them to plate. Minimum distance c. to c. of rivets not less than 3 diams of rivet, or preferably 3 in for ⅞ in rivets, 2½ for ¾, 2 for ⅝, 1¾ for ½. Max pitch in line of the stress, for members composed of plates and shapes, 6 in for ⅞ and ¾, 4½ for ⅝, and 4 for ½-in rivets. Where 2 or more plates are in contact, rivets to be not more than 12 in apart in either direction.

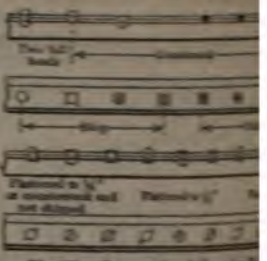


Fig 66. Conventional Signs for Rivets, Bolts, and Rivet Heads.

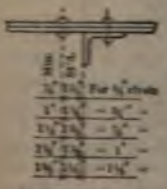
27. Rivets, Bolts, Connections

Rivets are of special steel (Art 25) and are computed for certain stresses (Art 26). They have a head already formed, and a [illegible] dependent on thickness of metal they are to go through.

Gages for Angles, in.

Leg	6	5	4	3½	3	2½	2	1¾	1½
g_1	3½	3	2½	2	1¾	1½	1¼	1	⅞
g_2	2½	2							
g_3	2¼	1¾							
Max. Rivet	⅞	⅞	⅞	⅞	⅞	⅞	⅞	⅝	⅝

Clearances



Diam. of rivet	⅞	¾	⅝	½	⅜	⅝	¾	⅞
Min. g_1 or g_2	1	1¼	1⅛	1	⅞	1⅛	1¼	1⅜

Fig 68. Spacing of Rivets

[illegible] portion of the 3.6-in strip between dotted lines: a, the [illegible] in = 1 rivet hole; b, ⅞ = 3.36, 10.72, 16.08, 21.44 kips. Plate can take 21,000 lb per [illegible]

(or bearing = 2 × 24 000 × ⅞ × ⅝ = 26 200 lb per 3.5-in strip. rivets in double shear. At an allowable shear of 12 000 lb per sq ... strip will be 2 × 2 × 12 000 × 0.785 × (⅞)² = 28 800 lb. The ...es is 25 000 lb, which must be taken as the safe strength. As a ...e will take 3.5 × ⅝ × 16 000 = 35 000 lb, the efficiency = 25 000

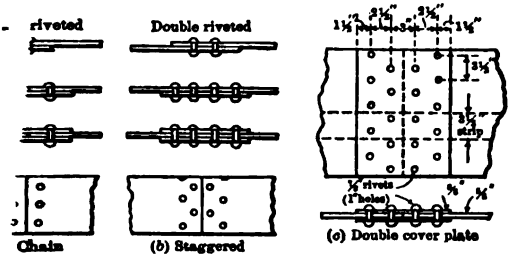


Fig 67. Riveted Joints

...uble-riveted butt joints should have effic of 65 to 75%; single-riveted, ...-riveted lap joints, 65 to 75%, treble and quadruple, 70 to 80%. The ... is that in which the safe load from 1, 2, and 3, above, are equal. For ..., this requires a rivet diam of about 1.6 to 2 *t*. Cover plate is made a ... main plates, to allow for bending. A butt joint with single cover is ..., brought close together. In double-cover butt joint, covers are made

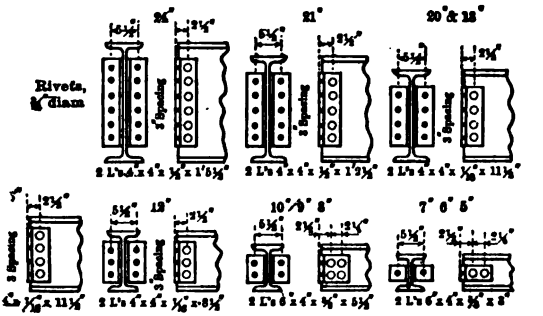


Fig 68. Standard Beam Connections (Carnegie)

... thickness of main plates, and rivets are smaller in diam, about 1.1 *t*. Dis... rows of chain riveting should be 2.5 to 3 *d*, and zigzag about 0.75 of this. ...te beyond rivet holes = 1.5 *d*. Minimum spacing of rivets (Art 26) and max ...alked edges should not exceed 10 times thickness of plate or cover.

...ons for beams. Fig 68 shows STANDARD CONNECTIONS, strength of which is ... above. Special connections may be used for heavy beam reactions. It is ...n designing such joints, that center lines of all connecting members should ... point, thus avoiding bending in the connection. Fig 69 shows a connection ...late and channel columns. Ends of columns are milled at right angles to their ...plate is placed between the ends, and splice and filler plates complete the con... SPLICE PLATES and their rivets are sometimes computed to transmit the entire ...en members, but the chief reliance is on the end bearing. SHOP RIVETING. ...f the total riveting as possible should be done in the shop, resulting in better

work and saving in cost. On small jobs, it may be economical to use bolts instead of rivets for connections. In BRIDGE WORK, pin connections are common, pins being calculated for shearing, bending, and bearing stresses, of the latter usually controlling. For bending, resisting moment = $frd^3 \div 32$, where d = diam, in; f = fiber stress, lb per sq in.

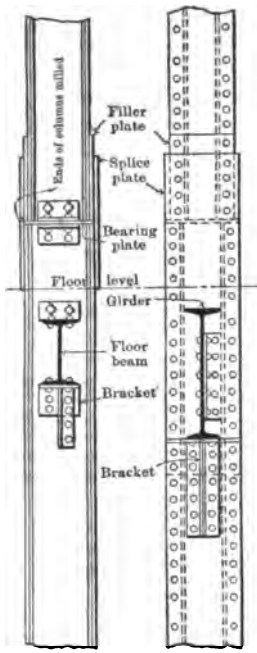


Fig 69. Connections for Plate and Channel Column

28. Columns and Girders

Columns may be of single, rolled sections, built up of plates, channels, and angles. The H-SECTION is widely used and saves in cost of fabrication. Design of a built-up column is simple, as shown below (see also Art 29).

Design a 45-ft latticed plate and angle column for a load of 250 000 lb (Fig 70). In general the unsupported length should not exceed 120 times the least radius of gyration r, or for secondary members, 200. Hence, smallest r for this column would be about $45 \times 12 \div 120$, or 4.5 in, and since, from type 10 in Fig 71, $r_x = 0.36\,h$, the plates should be at least $4.5 \div 0.36 = 12.5$ in. But better proportions require a width of $\frac{1}{15}$s to $\frac{1}{40}$ the length, or about 16 in, in which case r_x = about $16 \times 0.36 = 5.76$; using the formula $p = 19\,000 - 100\,l \div r$, $p = 9\,600$ lb per sq in, and necessary area = $250\,000 \div 9\,600 = 26.1$ sq in. Selection of plates and angles to make up this area is largely a matter of judgment. A plate should not be wider than 55 times its thickness, or angle legs longer than 9 times the thickness. Try 2 16 by 0.5-in plates and 4 3 by 3 by 0.5-in *Ls*. Total area = 27 sq in (Table 21). This design should be checked for axis $x - x$, and distance b determined so as to make r at least equal for axis $y - y$. Since moment of inertia (I) of a section about any axis = I about the axis through its cent grav + area times square of distance between axes, the value of I about $x - x$ is found as follows: For plates, $I = bh^3 \div 12 = 341.3$; for angles (Table 21) about own axis $I = 2.2 \times 4 = 8.8$ or plus $4 \times 2.75 \times (8 - 0.93)^2 = 550$, or total = 890.1; and, since $r = \sqrt{I \div A} = 5.75$, it checks closely with that used in determining p. From Fig 71, $r_y = 0.53\,b$, or $b = 5.75 \div 0.53 = 11$ in. Taking I about $y - y$ axis, this may be checked as before, and r is found to be satisfactory. Rivets for angles and plates in this design must be spaced not over 6 in, or 16 times the thickness of thinner plate. In this case 6 in controls, and 0.75-in rivets will be used. For about 1.5 times the width of column, or 2 ft at each end, rivets are pitched at 4 times their diam (= 3 in) to give extra strength where load is applied.

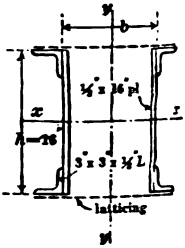


Fig 70. Latticed Column

Component parts of a built-up column must be rigidly connected and held in place, to maintain the radius of gyration r. This is done in columns of types 8, 9, 15, 16, etc (Fig 71), by means of solid plates or diaphragms. It is sometimes undesirable. Thus, for bridge posts, types 10, 13, 14 of Fig 71 are used, to make the entire section accessible for painting and connections. Upper chords are usually made with plate on top and open below (type 11). To maintain rigidity of open members single or double LATTICE BARS are usually applied to the main part of the column, with BATTEN PLATES to give greater rigidity at ends and intermediate connection points, or where latticing is interrupted (Fig 28, 29, 30).

Joint for a column shown in Fig 69. here columns rest masonry footings C-I base or steel se plate is used.

Girders. When ie bending moent is too large r a standard lled beam, a spe-al beam or girder ust be built up. irders are usually ade with flanges rmed by angles iveted back to ack, with cover lates to give re-uired flange area hen necessary, nd a web consist-ig of a plate PLATE GIRDER) or ı light construc-ion, latticing (LAT-ICED GIRDER).

In plate girder de-ign (19), the web is ssumed to resist all he shear and the anges the bending ıoment. Upper ange of a girder is ı compression and ower in tension, and

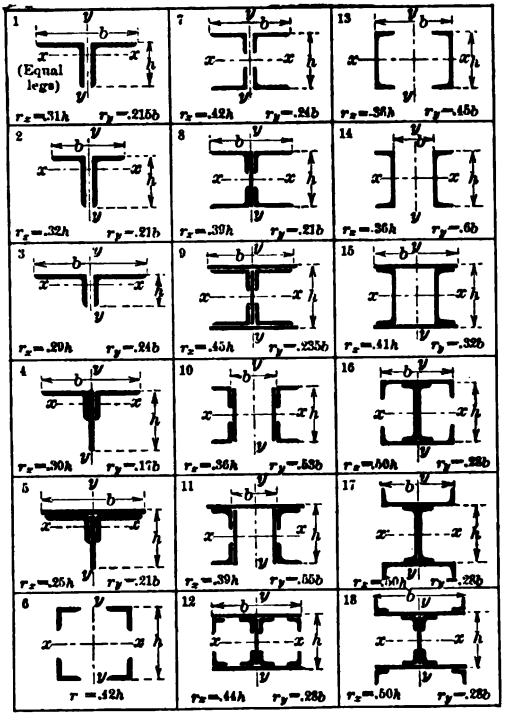


Fig 71. Approx Radii of Gyration of Column Sections

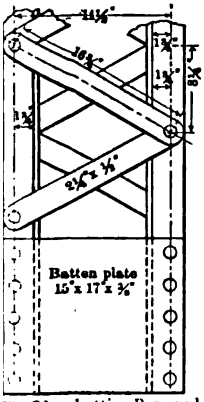


Fig 72. Lattice Bars and Batten Plate

resisting moment, in-lb, $= fdA$, where f = allowable unit stress, lb per sq in, d = EFFECTIVE DEPTH of girder, in, and A = net flange area, sq in. Since the web can resist a small amount of bending, allowance is often made for this by considering a fractional part (usually ⅙) of the web as equivalent to flange area and adding it to the net area of flange angles and cover plates. Only the tension flange is usually designed, the compression flange being made the same. Effective depth is distance between centers of gravity of upper and lower flange. It can only be obtained accurately by series of trials, and for small girders is often taken as equal to depth of web; it will generally be greater when cover plates are used. (For details of design of built-up girders, see Bib 19.)

29. Bridges and Buildings

Bridges. Various forms of trusses (Fig 40) are used to carry loads on spans too great for plate girders, or where there is insufficient headroom for a deck girder and a truss is economical.

No definite limiting span can be set for different types of bridge construction. For ordinary loads, roll-

can generally be used up to spans of about 20 ft, and plate girders up to spans of 120 ft and over under certain conditions, but the aver limit is about 50 or 60 ft. In U S, trusses of 150-ft span or less generally have riveted connections; trusses of larger span are usually connected. Riveted connections are stiffer, but harder to erect. (For details of design of small and large bridges, see Bib 19.)

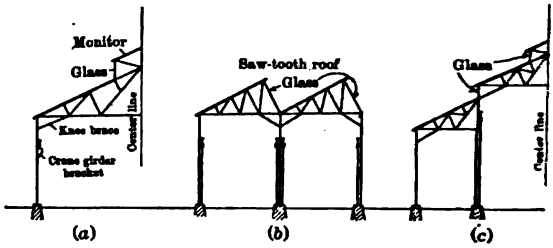


Fig 73. Forms of Steel Buildings

Buildings. Steel mill buildings are of 3 types: (*a*) steel truss and braced columns, covered with corrugated iron (Art 24), or thin concrete walls, supported by the steel frame; (*b*) same as above, except columns are braced by thin walls supporting masonry walls; (*c*) steel roof trusses, supported on masonry walls.

Fig 73 shows 3 common forms of construction. Saw-tooth roof is usually so placed that the glass has a northern exposure. Fig 74 shows the cross-sec of a building of type *a*, with wind force acting against it. Dead and snow-load stresses in the roof truss are computed as in Art 17, but the column, brace, and wind stresses depend largely on wind load and on condition of fixity of columns. Fig 74 illustrates conditions when columns are

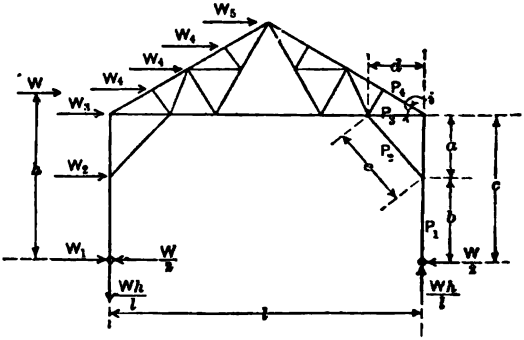


Fig 74. Distribution of Wind Pressures on Roof Truss

assumed as fixed at upper and hinged at lower end, which is usual assumption. Wind load is taken as acting at panel points, the individual loads being W_1, W_2, W_3, W_4, and W_5. The resultant W, acting at height h, is found by taking moments about the column base, the forces required for equilibrium being shown. In column P_1 the max moment producing bending is just below the knee brace, and $= Wb + 2$; the column must be designed (Art 6) to allow for this, in addition to direct load from truss. The shear above brace $= Wb + 2a$, and below brace $= W + 2$. The direct stress due to wind below the brace is $Wh + l$; above brace $= (Wc + 2d) - (Wh + l)$. In brace, $P_2 = Wcs + 2sd$. Member $P_4 = (Wc + 2d) - (Wh + l)\ cosec\ i$. Chord $P_3 = P_4 \cos i - Wb + 2a$. Stresses in other members of the truss are found by applying the forces P_2, P_3, and P_4 to the truss and methods of Art 18.

Fig 75 shows crane girders for carrying the rails for traveling cranes. If the girder is pported by a bracket as in (*a*), the resulting eccentric stress must be allowed for (Art 6). ith a wider column below the girder in (*b*), the connection with the aller upper section must be de- ned for the moment due to wind. aveling cranes are liable to produce ide thrust, usually taken as 0.05 the ing capacity of the crane acting on ch of the 4 wheels; 25% should also added to crane stresses for impact d vibration. (For details of monitor, ss construction, etc, see Bib 22.)

The design of a steel roof truss made in an entirely similar anner to the small bridge given ove. The purlins are either of nber, with wooden roofing as in g 60 (in which case a short ece of angle is used as a brace attach the timber purlin to e steel truss at the panel point), for a corrugated iron roof the rlins would be of steel, giving eproof construction.

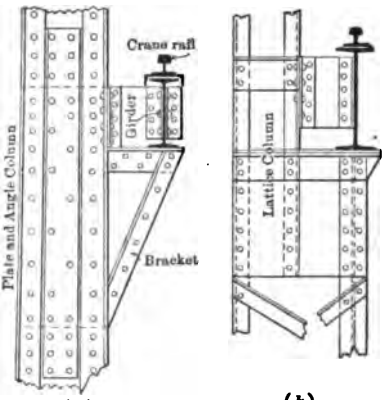


Fig 75. Crane Girders

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SECTION 44

ENGINEERS' TABLES

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Note.— Tables 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 13, 19, and 23 are by permission reprinted here from Merriman's "Civil Engineers' Pocketbook."

1. Squares, Cubes, Square and Cube Roots

No	Square	Cube	Square root	Cube root	No	Square	Cube	Square root	Cube root
1	1	1	1	1	50	2 500	125 000	7.071	3.684
2	4	8	1.414	1.260	51	2 601	132 651	7.141	3.708
3	9	27	1.732	1.442	52	2 704	140 608	7.211	3.733
4	16	64	2	1.587	53	2 809	148 877	7.280	3.756
5	25	125	2.236	1.710	54	2 916	157 464	7.348	3.780
6	36	216	2.449	1.817	55	3 025	166 375	7.416	3.803
7	49	343	2.646	1.913	56	3 136	175 616	7.483	3.826
8	64	512	2.828	2	57	3 249	185 193	7.550	3.849
9	81	729	3	2.080	58	3 364	195 112	7.616	3.871
10	100	1 000	3.162	2.154	59	3 481	205 379	7.681	3.893
11	121	1 331	3.317	2.224	60	3 600	216 000	7.746	3.915
12	144	1 728	3.464	2.289	61	3 721	226 981	7.810	3.936
13	169	2 197	3.606	2.351	62	3 844	238 328	7.874	3.958
14	196	2 744	3.742	2.410	63	3 969	250 047	7.937	3.979
15	225	3 375	3.873	2.466	64	4 096	262 144	8	4
16	256	4 096	4	2.520	65	4 225	274 625	8.062	4.021
17	289	4 913	4.123	2.571	66	4 356	287 496	8.124	4.041
18	324	5 832	4.243	2.621	67	4 489	300 763	8.185	4.062
19	361	6 859	4.359	2.668	68	4 624	314 432	8.246	4.082
20	400	8 000	4.472	2.714	69	4 761	328 509	8.307	4.102
21	441	9 261	4.583	2.759	70	4 900	343 000	8.367	4.121
22	484	10 648	4.690	2.802	71	5 041	357 911	8.426	4.141
23	529	12 167	4.796	2.844	72	5 184	373 248	8.485	4.160
24	576	13 824	4.899	2.884	73	5 329	389 017	8.544	4.179
25	625	15 625	5	2.924	74	5 476	405 224	8.602	4.198
26	676	17 576	5.099	2.962	75	5 625	421 875	8.660	4.217
27	729	19 683	5.196	3	76	5 776	438 976	8.718	4.236
28	784	21 952	5.292	3.037	77	5 929	456 533	8.775	4.254
29	841	24 389	5.385	3.072	78	6 084	474 552	8.832	4.273
30	900	27 000	5.477	3.107	79	6 241	493 039	8.888	4.291
31	961	29 791	5.568	3.141	80	6 400	512 000	8.944	4.309
32	1 024	32 768	5.657	3.175	81	6 561	531 441	9	4.327
33	1 089	35 937	5.745	3.208	82	6 724	551 368	9.055	4.344
34	1 156	39 304	5.831	3.240	83	6 889	571 787	9.110	4.362
35	1 225	42 875	5.916	3.271	84	7 056	592 704	9.165	4.380
36	1 296	46 656	6	3.302	85	7 225	614 125	9.220	4.397
37	1 369	50 653	6.083	3.332	86	7 396	636 056	9.274	4.414
38	1 444	54 872	6.164	3.362	87	7 569	658 503	9.328	4.431
39	1 521	59 319	6.245	3.391	88	7 744	681 472	9.381	4.448
40	1 600	64 000	6.325	3.420	89	7 921	704 969	9.434	4.465
41	1 681	68 921	6.403	3.448	90	8 100	729 000	9.487	4.481
42	1 764	74 088	6.481	3.476	91	8 281	753 571	9.539	4.498
43	1 849	79 507	6.557	3.503	92	8 464	778 688	9.592	4.514
44	1 936	85 184	6.633	3.530	93	8 649	804 357	9.644	4.531
45	2 025	91 125	6.708	3.557	94	8 836	830 584	9.695	4.547
46	2 116	97 336	6.782	3.583	95	9 025	857 375	9.747	4.563
47	2 209	103 823	6.856	3.609	96	9 216	884 736	9.798	4.579
48	2 304	110 592	6.928	3.634	97	9 409	912 673	9.849	4.595
49	2 401	117 649	7	3.659	98	9 604	941 192	9.899	4.610
					99	9 801	970 299	9.950	4.626

Squares, Cubes, Square and Cube Roots (*Continued*)

No	Square	Cube	Square root	Cube root	No	Square	Cube	Square root	[illegible]
100	10 000	1 000 000	10	4.642	150	22 500	3 375 000	12 247	[illegible]
101	10 201	1 030 301	10.050	4.657	151	22 801	3 442 951	12.288	[illegible]
102	10 404	1 061 208	10.100	4.672	152	23 104	3 511 808	12.329	[illegible]
103	10 609	1 092 727	10.149	4.688	153	23 409	3 581 577	12 369	[illegible]
104	10 816	1 124 864	10.198	4.703	154	23 716	3 652 264	12 410	[illegible]
105	11 025	1 157 625	10.247	4.718	155	24 025	3 723 875	12 450	[illegible]
106	11 236	1 191 016	10.296	4.733	156	24 336	3 796 416	12.490	[illegible]
107	11 449	1 225 043	10.344	4.748	157	24 649	3 869 893	12 530	[illegible]
108	11 664	1 259 712	10.392	4.762	158	24 964	3 944 312	12 570	[illegible]
109	11 881	1 295 029	10.440	4.777	159	25 281	4 019 679	12.610	[illegible]
110	12 100	1 331 000	10.488	4.791	160	25 600	4 096 000	12.650	[illegible]
111	12 321	1 367 631	10.536	4.806	161	25 921	4 173.281	12.689	[illegible]
112	12 544	1 404 928	10.583	4.820	162	26 244	4 251 528	12 728	[illegible]
113	12 769	1 442 897	10.630	4.835	163	26 569	4 330 747	12.767	[illegible]
114	12 996	1 481 544	10.677	4.849	164	26 896	4 410 944	12.806	[illegible]
115	13 225	1 520 875	10.724	4.863	165	27 225	4 492 125	12.845	[illegible]
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121	14 641	1 771 561	11	4.946	171	29 241	5 000 211	13 077	[illegible]
122	14 884	1 815 848	11.045	4.960	172	29 584	5 088 448	13 115	[illegible]
123	15 129	1 860 867	11.091	4.973	173	29 929	5 177 717	13 153	[illegible]
124	15 376	1 906 624	11.136	4.987	174	30 276	5 268 024	13 191	[illegible]
125	15 625	1 953 125	11.180	5	175	30 625	5 359 375	13 229	[illegible]
126	15 876	2 000 376	11.225	5.013	176	30 976	5 451 776	13 267	[illegible]
127	16 129	2 048 383	11.269	5 027	177	31 329	5 545 233	13 304	[illegible]
128	16 384	2 097 152	11.314	5.040	178	31 684	5 639 752	13 342	[illegible]
129	16 641	2 146 689	11.358	5.053	179	32 041	5 735 339	13 380	[illegible]
130	16 900	2 197 000	11.402	5.066	180	32 400	5 832 000	13 416	[illegible]
131	17 161	2 248 091	11.446	5 079	181	32 761	5 929 741	13 454	[illegible]
132	17 424	2 299 968	11.489	5.092	182	33 124	6 028 568	13 491	[illegible]
133	17 689	2 352 637	11.533	5.105	183	33 489	6 128 487	13 528	[illegible]
134	17 956	2 406 104	11.576	5.117	184	33 856	6 229 504	13 565	[illegible]
135	18 225	2 460 375	11.619	5.130	185	34 225	6 331 625	13 601	[illegible]
136	18 496	2 515 456	11.662	5.143	186	34 596	6 434 856	13 638	[illegible]
137	18 769	2 571 353	11.705	5.155	187	34 969	6 539 203	13.675	[illegible]
138	19 044	2 628 072	11.747	5.168	188	35 344	6 644 672	13 711	[illegible]
139	19 321	2 685 619	11.790	5.180	189	35 721	6 751 269	13 748	[illegible]
140	19 600	2 744 000	11.832	5.192	190	36 100	6 859 000	13 784	[illegible]
141	19 881	2 803 221	11.874	5.205	191	36 481	6 967 871	13 820	[illegible]
142	20 164	2 863 288	11.916	5.217	192	36 864	7 077 888	13 856	[illegible]
143	20 449	2 924 207	11.958	5.229	193	37 249	7 189 057	13 892	[illegible]
144	20 736	2 985 984	12	5.241	194	37 636	7 301 384	13 928	[illegible]
145	21 025	3 048 625	12.042	5.254	195	38 025	7 414 875	13 964	[illegible]
146	21 316	3 112 136	12.083	5.266	196	38 416	7 529 536	14	[illegible]
147	21 609	3 176 523	12.124	5.278	197	38 809	7 645 373	14 036	[illegible]
148	21 904	3 241 792	12.166	5 290	198	39 204	7 762 392	14 071	[illegible]
149	22 201	3 307 949	12.207	5.321	199	39 601	7 880 599	14 107	[illegible]

Squares, Cubes, Square and Cube Roots (*Continued*)

No	Square	Cube	Square root	Cube root	No	Square	Cube	Square root	Cube root
0	40 000	8 000 000	14.142	5.848	250	62 500	15 625 000	15.811	6.300
1	40 401	8 120 601	14.177	5.858	251	63 001	15 813 251	15.843	6.308
2	40 804	8 242 408	14.213	5.867	252	63 504	16 003 008	15.875	6.316
3	41 209	8 365 427	14.248	5.877	253	64 009	16 194 277	15.906	6.325
4	41 616	8 489 664	14.283	5.887	254	64 516	16 387 064	15.937	6.333
5	42 025	8 615 125	14.318	5.896	255	65 025	16 581 375	15.969	6.341
6	42 436	8 741 816	14.353	5.906	256	65 536	16 777 216	16	6.350
7	42 849	8 869 743	14.387	5.915	257	66 049	16 974 593	16.031	6.358
8	43 264	8 998 912	14.422	5.925	258	66 564	17 173 512	16.062	6.366
9	43 681	9 129 329	14.457	5.934	259	67 081	17 373 979	16.093	6.374
0	44 100	9 261 000	14.491	5.944	260	67 600	17 576 000	16.125	6.383
1	44 521	9 393 931	14.526	5.953	261	68 121	17 779 581	16.155	6.391
2	44 944	9 528 128	14.560	5.963	262	68 644	17 984 728	16.186	6.399
3	45 369	9 663 597	14.595	5.972	263	69 169	18 191 447	16.217	6.407
4	45 796	9 800 344	14.629	5.981	264	69 696	18 399 744	16.248	6.415
5	46 225	9 938 375	14.663	5.991	265	70 225	18 609 625	16.279	6.423
6	46 656	10 077 696	14.697	6	266	70 756	18 821 096	16.310	6.431
7	47 089	10 218 313	14.731	6.009	267	71 289	19 034 163	16.340	6.439
8	47 524	10 360 232	14.765	6.019	268	71 824	19 248 832	16.371	6.447
9	47 961	10 503 459	14.799	6.028	269	72 361	19 465 109	16.401	6.455
0	48 400	10 648 000	14.832	6.037	270	72 900	19 683 000	16.432	6.463
1	48 841	10 793 861	14.866	6.046	271	73 441	19 902 511	16.462	6.471
2	49 284	10 941 048	14.900	6.055	272	73 984	20 123 648	16.492	6.479
3	49 729	11 089 567	14.933	6.064	273	74 529	20 346 417	16.523	6.487
4	50 176	11 239 424	14.967	6.073	274	75 076	20 570 824	16.553	6.495
5	50 625	11 390 625	15	6.082	275	75 625	20 796 875	16.583	6.503
6	51 076	11 543 176	15.033	6.091	276	76 176	21 024 576	16.613	6.511
7	51 529	11 697 083	15.067	6.100	277	76 729	21 253 933	16.643	6.519
8	51 984	11 852 352	15.100	6.109	278	77 284	21 484 952	16.673	6.527
9	52 441	12 008 989	15.133	6.118	279	77 841	21 717 639	16.703	6.534
0	52 900	12 167 000	15.166	6.127	280	78 400	21 952 000	16.733	6.542
1	53 361	12 326 391	15.199	6.136	281	78 961	22 188 041	16.763	6.550
2	53 824	12 487 168	15.232	6.145	282	79 524	22 425 768	16.793	6.558
3	54 289	12 649 337	15.264	6.153	283	80 089	22 665 187	16.823	6.565
4	54 756	12 812 904	15.297	6.162	284	80 656	22 906 304	16.852	6.573
5	55 225	12 977 875	15.330	6.171	285	81 225	23 149 125	16.882	6.581
6	55 696	13 144 256	15.362	6.180	286	81 796	23 393 656	16.912	6.589
7	56 169	13 312 053	15.395	6 188	287	82 369	23 639 903	16.941	6.596
8	56 644	13 481 272	15.427	6.197	288	82 944	23 887 872	16.971	6.604
9	57 121	13 651 919	15.460	6.206	289	83 521	24 137 569	17	6 611
0	57 600	13 824 000	15.492	6 214	290	84 100	24 389 000	17.029	6.619
1	58 081	13 997 521	15.524	6.223	291	84 681	24 642 171	17.059	6.627
2	58 564	14 172 488	15.556	6.232	292	85 264	24 897 088	17.088	6 634
3	59 049	14 348 907	15.588	6.240	293	85 849	25 153 757	17.117	6 642
4	59 536	14 526 784	15.620	6.249	294	86 436	25 412 184	17.146	6.649
5	60 025	14 706 125	15.652	6.257	295	87 025	25 672 375	17.176	6 657
6	60 516	14 886 936	15.684	6 266	296	87 616	25 934 336	17.205	6.664
7	61 009	15 069 223	15.716	6 274	297	88 209	26 198 073	17.234	6.672
8	61 504	15 252 992	15.748	6 283	298	88 804	26 463 592	17.263	6 679
9	62 001	15 438 249	15.780	6 291	299	89 401	26 730 899	17.292	6 6

Squares, Cubes, Square and Cube Roots (*Continued*)

No	Square	Cube	Square root	Cube root	No	Square	Cube	Square root	Cube root
300	90 000	27 000 000	17.321	6.694	350	122 500	42 875 000	18 708	
301	90 601	27 270 901	17.349	6.702	351	123 201	43 243 551	18.735	
302	91 204	27 543 608	17.378	6.709	352	123 904	43 614 208	18.762	
303	91 809	27 818 127	17.407	6.717	353	124 609	43 986 977	18.788	
304	92 416	28 094 464	17.436	6.724	354	125 316	44 361 864	18.815	
305	93 025	28 372 625	17.462	6.731	355	126 025	44 738 875	18.841	
306	93 636	28 652 616	17.493	6.739	356	126 736	45 118 016	18.868	
307	94 249	28 934 443	17.521	6 746	357	127 449	45 499 293	18.894	
308	94 864	29 218 112	17.550	6.753	358	128 164	45 882 712	18.921	
309	95 481	29 503 629	17.578	6.761	359	128 881	46 268 279	18.947	
310	96 100	29 791 000	17.607	6.768	360	129 600	46 656 000	18.974	
311	96 721	30 080 231	17.635	6.775	361	130 321	47 045 881	19	
312	97 344	30 371 328	17.664	6.782	362	131 044	47 437 928	19.026	
313	97 969	30 664 297	17.692	6.790	363	131 769	47 832 147	19.053	
314	98 596	30 959 144	17.720	6.797	364	132 496	48 228 544	19.079	
315	99 225	31 255 875	17.748	6.804	365	133 225	48 627 125	19.105	
316	99 856	31 554 496	17.776	6.811	366	133 956	49 027 896	19.131	
317	100 489	31 855 013	17.804	6.818	367	134 689	49 430 863	19.157	
318	101 124	32 157 432	17.833	6.826	368	135 424	49 836 032	19.183	
319	101 761	32 461 759	17.861	6.833	369	136 161	50 243 409	19.209	
320	102 400	32 768 000	17.889	6.840	370	136 900	50 653 000	19 235	
321	103 041	33 076 161	17.916	6.847	371	137 641	51 064 811	19.261	
322	103 684	33 386 248	17.944	6.854	372	138 384	51 478 848	19.287	
323	104 329	33 698 267	17.972	6.861	373	139 129	51 895 117	19 313	
324	104 976	34 012 224	18	6.868	374	139 876	52 313 624	19.339	
325	105 625	34 328 125	18.028	6.875	375	140 625	52 734 375	19.365	
326	106 276	34 645 976	18.055	6.882	376	141 376	53 157 376	19 391	
327	106 929	34 965 783	18.083	6.889	377	142 129	53 582 633	19.416	
328	107 584	35 287 552	18.111	6.896	378	142 884	54 010 152	19.442	
329	108 241	35 611 289	18.138	6.903	379	143 641	54 439 939	19.468	
330	108 900	35 937 000	18.166	6.910	380	144 400	54 872 000	19.494	
331	109 561	36 264 691	18.193	6.917	381	145 161	55 306 341	19.519	
332	110 224	36 594 368	18.221	6.924	382	145 924	55 742 968	19 545	
333	110 889	36 926 037	18.248	6.931	383	146 689	56 181 887	19.570	
334	111 556	37 259 704	18.276	6.938	384	147 456	56 623 104	19.596	
335	112 225	37 595 375	18.303	6.945	385	148 225	57 066 625	19.621	
336	112 896	37 933 056	18.330	6.952	386	148 996	57 512 456	19.647	
337	113 569	38 272 753	18.358	6.959	387	149 769	57 960 603	19.672	
338	114 244	38 614 472	18.385	6.966	388	150 544	58 411 072	19.698	
339	114 921	38 958 219	18.412	6 973	389	151 321	58 863 869	19 723	
340	115 600	39 304 000	18.439	6.980	390	152 100	59 319 000	19.748	
341	116 281	39 651 821	18.466	6.986	391	152 881	59 776 471	19.774	
342	116 964	40 001 688	18.493	6.993	392	153 664	60 236 288	19 799	
343	117 649	40 353 607	18.520	7	393	154 449	60 698 457	19 821	
344	118 336	40 707 584	18.547	7.007	394	155 236	61 162 984	19 849	
345	119 025	41 063 625	18.574	7.014	395	156 025	61 629 875	19 875	
346	119 716	41 421 736	18.601	7.020	396	156 816	62 099 136	19 900	
347	120 409	41 781 923	18 628	7 027	397	157 609	62 570 773	19 925	
348	121 104	42 144 192	18 655	7.034	398	158 404	63 044 792	19.950	
		42 508 549	18.682	7 041	399	159 201	63 521 199	19 975	

Squares, Cubes, Square and Cube Roots (*Continued*)

o	Square	Cube	Square root	Cube root	No	Square	Cube	Square root	Cube root
00	160 000	64 000 000	20	7.368	450	202 500	91 125 000	21.213	7.663
01	160 801	64 481 201	20.025	7.374	451	203 401	91 733 851	21.237	7.669
02	161 604	64 964 808	20.050	7.380	452	204 304	92 345 408	21.260	7.674
03	162 409	65 450 827	20.075	7.386	453	205 209	92 959 677	21.284	7.680
04	163 216	65 939 264	20.100	7.393	454	206 116	93 576 664	21.307	7.686
05	164 025	66 430 125	20.125	7.399	455	207 025	94 196 375	21.331	7.691
06	164 836	66 923 416	20.149	7.405	456	207 936	94 818 816	21.354	7.697
07	165 649	67 419 143	20.174	7.411	457	208 849	95 443 993	21.378	7.703
08	166 464	67 917 312	20.199	7.417	458	209 764	96 071 912	21.401	7.708
09	167 281	68 417 929	20.224	7.423	459	210 681	96 702 579	21.424	7.714
10	168 100	68 921 000	20.248	7.429	460	211 600	97 336 000	21.448	7.719
11	168 921	69 426 531	20.273	7.435	461	212 521	97 972 181	21.471	7.725
12	169 744	69 934 528	20.298	7.441	462	213 444	98 611 128	21.494	7.731
13	170 569	70 444 997	20.322	7.447	463	214 369	99 252 847	21.517	7.736
14	171 396	70 957 944	20.347	7.453	464	215 296	99 897 344	21.541	7.742
15	172 225	71 473 375	20.372	7.459	465	216 225	100 544 625	21.564	7.747
16	173 056	71 991 296	20.396	7.465	466	217 156	101 194 696	21.587	7.753
17	173 889	72 511 713	20.421	7.471	467	218 089	101 847 563	21.610	7.758
18	174 724	73 034 632	20.445	7.477	468	219 024	102 503 232	21.633	7.764
19	175 561	73 560 059	20.469	7.483	469	219 961	103 161 709	21.656	7.769
10	176 400	74 088 000	20.494	7.489	470	220 900	103 823 000	21.679	7.775
11	177 241	74 618 461	20.518	7.495	471	221 841	104 487 111	21.703	7.780
12	178 084	75 151 448	20.543	7.501	472	222 784	105 154 048	21.726	7.786
13	178 929	75 686 967	20.567	7.507	473	223 729	105 823 817	21.749	7.791
14	179 776	76 225 024	20.591	7.513	474	224 676	106 496 424	21.772	7.797
15	180 625	76 765 625	20.616	7.518	475	225 625	107 171 875	21.794	7.802
16	181 476	77 308 776	20.640	7.524	476	226 576	107 850 176	21.817	7.808
17	182 329	77 854 483	20.664	7.530	477	227 529	108 531 333	21.840	7.813
18	183 184	78 402 752	20.688	7.536	478	228 484	109 215 352	21.863	7.819
19	184 041	78 953 589	20.712	7.542	479	229 441	109 902 239	21.886	7.824
20	184 900	79 507 000	20.736	7.548	480	230 400	110 592 000	21.909	7.830
21	185 761	80 062 991	20.761	7.554	481	231 361	111 284 641	21.932	7.835
22	186 624	80 621 568	20.785	7.560	482	232 324	111 980 168	21.954	7.841
23	187 489	81 182 737	20.809	7.565	483	233 289	112 678 587	21.977	7.846
24	188 356	81 746 504	20.833	7.571	484	234 256	113 379 904	22	7.851
25	189 225	82 312 875	20.857	7.577	485	235 225	114 084 125	22.023	7.857
26	190 096	82 881 856	20.881	7.583	486	236 196	114 791 256	22.045	7.862
27	190 969	83 453 453	20.905	7.589	487	237 169	115 501 303	22.068	7.868
28	191 844	84 027 672	20.928	7.594	488	238 144	116 214 272	22.091	7.873
29	192 721	84 604 519	20.952	7.600	489	239 121	116 930 169	22.113	7.878
30	193 600	85 184 000	20.976	7.606	490	240 100	117 649 000	22.136	7.884
31	194 481	85 766 121	21	7.612	491	241 081	118 370 771	22.159	7.889
32	195 364	86 350 888	21.024	7.617	492	242 064	119 095 488	22.181	7.894
33	196 249	86 938 307	21.048	7.623	493	243 049	119 823 157	22.204	7.900
34	197 136	87 528 384	21.071	7.629	494	244 036	120 553 784	22.226	7.905
35	198 025	88 121 125	21.095	7.635	495	245 025	121 287 375	22.249	7.910
36	198 916	88 716 536	21.119	7.640	496	246 016	122 023 936	22.271	7.916
37	199 809	89 314 623	21 142	7.646	497	247 009	122 763 473	22.293	7.921
38	200 704	89 915 392	21.166	7.652	498	248 004	123 505 992	22.316	7.926
39	201 601	90 518 849	21.190	7.657	499	249 001	124 251 499	22.338	7.932

Squares, Cubes, Square and Cube Roots (*Continued*)

No	Square	Cube	Square root	Cube root	No	Square	Cube	Square root	Cube root
500	250 000	125 000 000	22.361	7.937	550	302 500	166 375 000	23 452	
501	251 001	125 751 501	22.383	7.943	551	303 601	167 284 151	23.473	
502	252 004	126 506 008	22.405	7.948	552	304 704	168 196 608	23.495	
503	253 009	127 263 527	22.428	7.953	553	305 809	169 112 377	23 516	
504	254 016	128 024 064	22.450	7.958	554	306 916	170 031 464	23.537	
505	255 025	128 787 625	22.472	7.963	555	308 025	170 953 875	23 558	
506	256 036	129 554 216	22.494	7.969	556	309 136	171 879 616	23 580	
507	257 049	130 323 843	22.517	7.974	557	310 249	172 808 693	23 601	
508	258 064	131 096 512	22.539	7.979	558	311 364	173 741 112	23.622	
509	259 081	131 872 229	22.561	7.984	559	312 481	174 676 879	23.643	
510	260 100	132 651 000	22.583	7.990	560	313 600	175 616 000	23 664	
511	261 121	133 432 831	22.605	7.995	561	314 721	176 558 481	23 685	
512	262 144	134 217 728	22.627	8	562	315 844	177 504 328	23.707	
513	263 169	135 005 697	22.650	8.005	563	316 969	178 453 547	23 728	
514	264 196	135 796 744	22.672	8.010	564	318 096	179 406 144	23.749	
515	265 225	136 590 875	22.694	8.016	565	319 225	180 362 125	23.770	
516	266 256	137 388 096	22.716	8.021	566	320 356	181 321 496	23.791	
517	267 289	138 188 413	22 738	8.026	567	321 489	182 284 263	23 812	
518	268 324	138 991 832	22.760	8.031	568	322 624	183 250 432	23.833	
519	269 361	139 798 359	22.782	8.036	569	323 761	184 220 009	23 854	
520	270 400	140 608 000	22.804	8.041	570	324 900	185 193 000	23 875	
521	271 441	141 420 761	22.825	8.047	571	326 041	186 169 411	23.896	
522	272 484	142 236 648	22.847	8.052	572	327 184	187 149 248	23 917	
523	273 529	143 055 667	22.869	8.057	573	328 329	188 132 517	23.937	
524	274 576	143 877 824	22.891	8.062	574	329 476	189 119 224	23.958	
525	275 625	144 703 125	22.913	8.067	575	330 625	190 109 375	23 979	
526	276 676	145 531 576	22.935	8.072	576	331 776	191 102 976	24	
527	277 729	146 363 183	22.957	8.077	577	332 929	192 100 033	24 021	
528	278 784	147 197 952	22.978	8.082	578	334 084	193 100 552	24.042	
529	279 841	148 035 889	23	8.088	579	335 241	194 104 539	24.062	
530	280 900	148 877 000	23.022	8.093	580	336 400	195 112 000	24.083	
531	281 961	149 721 291	23.043	8.098	581	337 561	196 122 941	24.104	
532	283 024	150 568 768	23.065	8.103	582	338 724	197 137 368	24.125	
533	284 089	151 419 437	23.087	8.108	583	339 889	198 155 287	24.145	
534	285 156	152 273 304	23.108	8.113	584	341 056	199 176 704	24.166	
535	286 225	153 130 375	23.130	8.118	585	342 225	200 201 625	24.187	
536	287 296	153 990 656	23.152	8.123	586	343 396	201 230 056	24 207	
537	288 369	154 854 153	23.173	8.128	587	344 569	202 262 003	24 228	
538	289 444	155 720 872	23.195	8.133	588	345 744	203 297 472	24 249	
539	290 521	156 590 819	23.216	8.138	589	346 921	204 336 469	24.269	
540	291 600	157 464 000	23.238	8.143	590	348 100	205 379 000	24.290	
541	292 681	158 340 421	23.259	8.148	591	349 281	206 425 071	24 310	
542	293 764	159 220 088	23.281	8.153	592	350 464	207 474 688	24 331	
543	294 849	160 103 007	23.302	8.158	593	351 649	208 527 857	24.352	
544	295 936	160 989 184	23.324	8.163	594	352 836	209 584 584	24.372	
545	297 025	161 878 625	23.345	8.168	595	354 025	210 644 875	24 393	
546	298 116	162 771 336	23.367	8.173	596	358 216	211 708 736	24 413	
547	299 209	163 667 323	23.388	8.178	597	356 409	212 776 173	24.434	
548	300 304	164 566 592	23.409	8.183	598	357 604	213 847 192	24.454	
549	301 401	165 469 149	23.431	8.188	599	358 801	214 921 799	24.474	

Squares, Cubes, Square and Cube Roots (*Continued*)

o	Square	Cube	Square root	Cube root	No	Square	Cube	Square root	Cube root
0	360 000	216 000 000	24.495	8.434	650	422 500	274 625 000	25.495	8.662
1	361 201	217 081 801	24.515	8.439	651	423 801	275 894 451	25.515	8.667
2	362 404	218 167 208	24.536	8.444	652	425 104	277 167 808	25.534	8.671
3	363 609	219 256 227	24.556	8.448	653	426 409	278 445 077	25.554	8.676
4	364 816	220 348 864	24.576	8.453	654	427 716	279 726 264	25.573	8.680
5	366 025	221 445 125	24.597	8.458	655	429 025	281 011 375	25.593	8.685
6	367 236	222 545 016	24.617	8.462	656	430 336	282 300 416	25.612	8.689
7	368 449	223 648 543	24.637	8.467	657	431 649	283 593 393	25.632	8.693
8	369 664	224 755 712	24.658	8.472	658	432 964	284 890 312	25.652	8.698
9	370 881	225 866 529	24.678	8.476	659	434 281	286 191 179	25.671	8.702
0	372 100	226 981 000	24.698	8.481	660	435 600	287 496 000	25.690	8.707
1	373 321	228 099 131	24.718	8.486	661	436 921	288 804 781	25.710	8.711
2	374 544	229 220 928	24.739	8.490	662	438 244	290 117 528	25.730	8.715
3	375 769	230 346 397	24.759	8.495	663	439 569	291 434 247	25.749	8.720
4	376 996	231 475 544	24.779	8.499	664	440 896	292 754 944	25.768	8.724
5	378 225	232 608 375	24.799	8.504	665	442 225	294 079 625	25.788	8.729
6	379 456	233 744 896	24.819	8.509	666	443 556	295 408 296	25.807	8.733
7	380 689	234 885 113	24.839	8.513	667	444 889	296 740 963	25.826	8.737
8	381 924	236 029 032	24.860	8.518	668	446 224	298 077 632	25.846	8.742
9	383 161	237 176 659	24.880	8.522	669	447 561	299 418 309	25.865	8.746
0	384 400	238 328 000	24.900	8.527	670	448 900	300 763 000	25.884	8.750
1	385 641	239 483 061	24.920	8.532	671	450 241	302 111 711	25.904	8.755
2	386 884	240 641 848	24.940	8.536	672	451 584	303 464 448	25.923	8.759
3	388 129	241 804 367	24.960	8.541	673	452 929	304 821 217	25.942	8.763
4	389 376	242 970 624	24.980	8.545	674	454 276	306 182 024	25.962	8.768
5	390 625	244 140 625	25	8.550	675	455 625	307 546 875	25.981	8.772
6	391 876	245 314 376	25.020	8.554	676	456 976	308 915 776	26	8.776
7	393 129	246 491 883	25.040	8.559	677	458 329	310 288 733	26.019	8.781
8	394 384	247 673 152	25.060	8.564	678	459 684	311 665 752	26.038	8.785
9	395 641	248 858 189	25.080	8.568	679	461 041	313 046 839	26.058	8.789
0	396 900	250 047 000	25.100	8.573	680	462 400	314 432 000	26.077	8.794
1	398 161	251 239 591	25.120	8.577	681	463 761	315 821 241	26 096	8.798
2	399 424	252 435 968	25.140	8.582	682	465 124	317 214 568	26.115	8.802
3	400 689	253 636 137	25.159	8.586	683	466 489	318 611 987	26.134	8 807
4	401 956	254 840 104	25.180	8.591	684	467 856	320 013 504	26.153	8 811
5	403 225	256 047 875	25.200	8.595	685	469 225	321 419 125	26 173	8.815
6	404 496	257 259 456	25.219	8.600	686	470 596	322 828 856	26 192	8.819
7	405 769	258 474 853	25.239	8.604	687	471 969	324 242 703	26 211	8.824
8	407 044	259 694 072	25.259	8.609	688	473 344	325 660 672	26 230	8.828
9	408 321	260 917 119	25.278	8.613	689	474 721	327 082 769	26 249	8.832
0	409 600	262 144 000	25.298	8.618	690	476 100	328 509 000	26 268	8.837
1	410 881	263 374 721	25.318	8.622	691	477 481	329 939 371	26.287	8.841
2	412 164	264 609 288	25.338	8.627	692	478 864	331 373 888	26.306	8.845
3	413 449	265 847 707	25.357	8.631	693	480 249	332 812 557	26 325	8.849
4	414 736	267 089 984	25.377	8.636	694	481 636	334 255 384	26 344	8.854
5	416 025	268 336 125	25.397	8.641	695	483 025	335 702 375	26.363	8.858
6	417 316	269 586 136	25.417	8.645	696	484 416	337 153 536	26.382	8.862
7	418 609	270 840 023	25 436	8 649	697	485 809	338 608 873	26.401	8.866
8	419 904	272 097 792	25 456	8.654	698	487 204	340 068 392	26 420	8.870
9	421 201	273 359 449	25.475	8 658	699	488 601	341 532 099	26.439	

Squares, Cubes, Square and Cube Roots (*Continued*)

No	Square	Cube	Square root	Cube root	No	Square	Cube	Square root	
700	490 000	343 000 000	26.458	8.879	750	562 500	421 875 000	27.386	9
701	491 401	344 472 101	26.476	8.883	751	564 001	423 564 751	27.404	9
702	492 804	345 948 408	26.495	8.887	752	565 504	425 259 008	27.423	9
703	494 209	347 428 927	26.514	8.892	753	567 009	426 957 777	27.441	9
704	495 616	348 913 664	26.533	8.896	754	568 516	428 661 064	27.459	9
705	497 025	350 402 625	26.552	8.900	755	570 025	430 368 875	27.477	9
706	498 436	351 895 816	26.571	8.904	756	571 536	432 081 216	27.495	9
707	499 849	353 393 243	26.589	8.909	757	573 049	433 798 093	27.514	9
708	501 264	354 894 912	26.608	8.913	758	574 564	435 519 512	27 532	9
709	502 681	356 400 829	26.627	8.917	759	576 081	437 245 479	27.550	9
710	504 100	357 911 000	26.646	8.921	760	577 600	438 976 000	27.568	9
711	505 521	359 425 431	26.665	8.925	761	579 121	440 711 081	27 586	9
712	506 944	360 944 128	26.683	8.929	762	580 644	442 450 728	27.604	9
713	508 369	362 467 097	26.702	8.934	763	582 169	444 194 947	27.622	9
714	509 796	363 994 344	26.721	8.938	764	583 696	445 943 744	27 641	9
715	511 225	365 525 875	26.739	8.942	765	585 225	447 697 125	27.659	9
716	512 656	367 061 696	26.758	8.946	766	586 756	449 455 096	27.677	9
717	514 089	368 601 813	26.777	8.950	767	588 289	451 217 663	27 695	9
718	515 524	370 146 232	26.796	8.955	768	589 824	452 984 832	27 713	9
719	516 961	371 694 959	26.814	8.959	769	591 361	454 756 609	27.731	9
720	518 400	373 248 000	26.833	8.963	770	592 900	456 533 000	27.749	9
721	519 841	374 805 361	26.851	8.967	771	594 441	458 314 011	27.767	9
722	521 284	376 367 048	26.870	8.971	772	595 984	460 099 648	27.785	9
723	522 729	377 933 067	26.889	8.975	773	597 529	461 889 917	27.803	9
724	524 176	379 503 424	26.907	8.979	774	599 076	463 684 824	27 821	9
725	525 625	381 078 125	26.926	8.984	775	600 625	465 484 375	27 839	9
726	527 076	382 657 176	26.944	8.988	776	602 176	467 288 576	27 857	9
727	528 529	384 240 583	26.963	8.992	777	603 729	469 097 433	27.875	9
728	529 984	385 828 352	26.981	8.996	778	605 284	470 910 952	27.893	9
729	531 441	387 420 489	27	9	779	606 841	472 729 139	27.911	9
730	532 900	389 017 000	27.019	9.004	780	608 400	474 552 000	27.928	9
731	534 361	390 617 891	27.037	9.008	781	609 961	476 379 541	27.946	9
732	535 824	392 223 168	27.055	9.012	782	611 524	478 211 768	27 964	9
733	537 289	393 832 837	27.074	9.016	783	613 089	480 048 687	27 982	9
734	538 756	395 446 904	27.092	9.021	784	614 656	481 890 304	28	9
735	540 225	397 065 375	27.111	9.025	785	616 225	483 736 625	28.018	9
736	541 696	398 688 256	27.129	9.029	786	617 796	485 587 656	28 036	9
737	543 169	400 315 553	27.148	9.033	787	619 369	487 443 403	28 054	9
738	544 644	401 947 272	27.166	9.037	788	620 944	489 303 872	28 071	9
739	546 121	403 583 419	27.185	9.041	789	622 521	491 169 069	28.089	9
740	547 600	405 224 000	27.203	9.045	790	624 100	493 039 000	28 107	9
741	549 081	406 869 021	27.221	9.049	791	625 681	494 913 671	28 125	9
742	550 564	408 518 488	27.240	9.053	792	627 264	496 793 088	28.142	9
743	552 049	410 172 407	27.258	9.057	793	628 849	498 677 257	28 160	9
744	553 536	411 830 784	27.276	9.061	794	630 436	500 566 184	28 178	9
745	555 025	413 493 625	27.295	9.065	795	632 025	502 459 875	28.196	9
746	556 516	415 160 936	27.313	9.069	796	633 616	504 358 336	28 213	9
747	558 009	416 832 723	27.331	9.073	797	635 209	506 261 573	28.231	9
748	559 504	418 508 992	27.350	9.078	798	636 804	508 169 592	28 249	9
749	561 001	420 189 749	27.368	9.082	799	638 401	510 082 399	28.267	9

Squares, Cubes, Square and Cube Roots (*Continued*)

o	Square	Cube	Square root	Cube root	No	Square	Cube	Square root	Cube root
0	640 000	512 000 000	28.284	9.283	850	722 500	614 125 000	29.155	9.473
1	641 601	513 922 401	28.302	9.287	851	724 201	616 295 051	29.172	9.476
2	643 204	515 849 608	28.320	9.291	852	725 904	618 470 208	29.189	9.480
3	644 809	517 781 627	28.337	9.295	853	727 609	620 650 477	29.206	9.484
4	646 416	519 718 464	28.355	9.299	854	729 316	622 835 864	29.223	9.488
5	648 025	521 660 125	28.373	9.302	855	731 025	625 026 375	29.240	9.491
6	649 636	523 606 616	28.390	9.306	856	732 736	627 222 016	29.257	9.495
7	651 249	525 557 943	28.408	9.310	857	734 449	629 422 793	29.275	9.499
8	652 864	527 514 112	28.425	9.314	858	736 164	631 628 712	29.292	9.502
9	654 481	529 475 129	28.443	9.318	859	737 881	633 839 779	29.309	9.506
0	656 100	531 441 000	28.460	9.322	860	739 600	636 056 000	29.326	9.510
1	657 721	533 411 731	28.478	9.326	861	741 321	638 277 381	29.343	9.513
2	659 344	535 387 328	28.496	9.329	862	743 044	640 503 928	29.360	9.517
3	660 969	537 367 797	28.513	9.333	863	744 769	642 735 647	29.377	9.521
4	662 596	539 353 144	28.531	9.337	864	746 496	644 972 544	29.394	9.524
5	664 225	541 343 375	28.548	9.341	865	748 225	647 214 625	29.411	9.528
6	665 856	543 338 496	28.566	9.345	866	749 956	649 461 896	29.428	9.532
7	667 489	545 338 513	28.583	9.348	867	751 689	651 714 363	29.445	9.535
8	669 124	547 343 432	28.601	9.352	868	753 424	653 972 032	29.462	9.539
9	670 761	549 353 259	28.618	9.356	869	755 161	656 234 909	29.479	9.543
0	672 400	551 368 000	28.636	9.360	870	756 900	658 503 000	29.496	9.546
1	674 041	553 387 661	28.653	9.364	871	758 641	660 776 311	29.513	9.550
2	675 684	555 412 248	28.671	9.368	872	760 384	663 054 848	29.530	9.554
3	677 329	557 441 767	28.688	9.371	873	762 129	665 338 617	29.547	9.557
4	678 976	559 476 224	28.705	9.375	874	763 876	667 627 624	29.563	9.561
5	680 625	561 515 625	28.723	9.379	875	765 625	669 921 875	29.580	9.565
6	682 276	563 559 976	28.740	9.383	876	767 376	672 221 376	29.597	9.568
7	683 929	565 609 283	28.758	9.386	877	769 129	674 526 133	29.614	9.572
8	685 584	567 663 552	28.775	9.390	878	770 884	676 836 152	29.631	9.576
9	687 241	569 722 789	28.792	9.394	879	772 641	679 151 439	29.648	9.579
0	688 900	571 787 000	28.810	9.398	880	774 400	681 472 000	29.665	9.583
1	690 561	573 856 191	28.827	9.402	881	776 161	683 797 841	29.682	9.586
2	692 224	575 930 368	28.844	9.405	882	777 924	686 128 968	29.698	9.590
3	693 889	578 009 537	28.862	9.409	883	779 689	688 465 387	29.715	9.594
4	695 556	580 093 704	28.879	9.413	884	781 456	690 807 104	29.732	9.597
5	697 225	582 182 875	28.896	9.417	885	783 225	693 154 125	29.749	9.601
6	698 896	584 277 056	28.914	9.420	886	784 996	695 506 456	29.766	9.605
7	700 569	586 376 253	28.931	9.424	887	786 769	697 864 103	29.783	9.608
8	702 244	588 480 472	28.948	9.428	888	788 544	700 227 072	29.799	9.612
9	703 921	590 589 719	28.965	9.432	889	790 321	702 595 369	29.816	9.615
0	705 600	592 704 000	28.983	9.435	890	792 100	704 969 000	29.832	9.619
1	707 281	594 823 321	29	9.439	891	793 881	707 347 971	29.850	9.623
2	708 964	596 947 688	29.017	9.443	892	795 664	709 732 288	29.866	9.626
3	710 649	599 077 107	29.034	9.447	893	797 449	712 121 957	29.883	9.630
4	712 336	601 211 584	29.052	9.450	894	799 236	714 516 984	29.900	9.633
5	714 025	603 351 125	29.069	9.454	895	801 025	716 917 375	29.917	9.637
6	715 716	605 495 736	29.086	9.458	896	802 816	719 323 136	29.933	9.641
7	717 409	607 645 423	29.103	9.462	897	804 609	721 734 273	29.950	9.644
8	719 104	609 800 192	29.120	9.465	898	806 404	724 150 792	29.967	9.648
9	720 801	611 960 049	29.138	9.469	899	808 201	726 572 699	29.983	9.651

Squares, Cubes, Square and Cube Roots (*Continued*)

No	Square	Cube	Square root	Cube root	No	Square	Cube	Square root
900	810 000	729 000 000	30	9.655	950	902 500	857 375 000	30.822
901	811 801	731 432 701	30.017	9.658	951	904 401	860 085 351	30.838
902	813 604	733 870 808	30.033	9.662	952	906 304	862 801 408	30.854
903	815 409	736 314 327	30.050	9.666	953	908 209	865 523 177	30.871
904	817 216	738 763 264	30.067	9.669	954	910 116	868 250 664	30.887
905	819 025	741 217 625	30.083	9.673	955	912 025	870 983 875	30.903
906	820 836	743 677 416	30.100	9.676	956	913 936	873 722 816	30.919
907	822 649	746 142 643	30.116	9.680	957	915 849	876 467 493	30.935
908	824 464	748 613 312	30.133	9.683	958	917 764	879 217 912	30.952
909	826 281	751 089 429	30.150	9.687	959	919 681	881 974 079	30.968
910	828 100	753 571 000	30.166	9.691	960	921 600	884 736 000	30.984
911	829 921	756 058 031	30.183	9.694	961	923 521	887 503 681	31
912	831 744	758 550 528	30.199	9.698	962	925 444	890 277 128	31.016
913	833 569	761 048 497	30.216	9.701	963	927 369	893 056 347	31.032
914	835 396	763 551 944	30.232	9 705	964	929 296	895 841 344	31.048
915	837 225	766 060 875	30.249	9.708	965	931 225	898 632 125	31.064
916	839 056	768 575 296	30.265	9.712	966	933 156	901 428 696	31.081
917	840 889	771 095 213	30.282	9.715	967	935 089	904 231 063	31.097
918	842 724	773 620 632	30.299	9.719	968	937 024	907 039 232	31.113
919	844 561	776 151 559	30.315	9.722	969	938 961	909 853 209	31.129
920	846 400	778 688 000	30.331	9.726	970	940 900	912 673 000	31.145
921	848 241	781 229 961	30.348	9.729	971	942 841	915 498 611	31.161
922	850 084	783 777 448	30.364	9.733	972	944 784	918 330 048	31 177
923	851 929	786 330 467	30.381	9.736	973	946 729	921 167 317	31.193
924	853 776	788 889 024	30.397	9.740	974	948 676	924 010 424	31.209
925	855 625	791 453 125	30.414	9.743	975	950 625	926 859 375	31.225
926	857 476	794 022 776	30.430	9.747	976	952 576	929 714 176	31.241
927	859 329	796 597 983	30.447	9.750	977	954 529	932 574 833	31 257
928	861 184	799 178 752	30.463	9.754	978	956 484	935 441 352	31.273
929	863 041	801 765 089	30.480	9.758	979	958 441	938 313 739	31.289
930	864 900	804 357 000	30.496	9.761	980	960 400	941 192 000	31.305
931	866 761	806 954 491	30.512	9.764	981	962 361	944 076 141	31.321
932	868 624	809 557 568	30.529	9.768	982	964 324	946 966 168	31.337
933	870 489	812 166 237	30.545	9.771	983	966 289	949 862 087	31.353
934	872 356	814 780 504	30.561	9.775	984	968 256	952 763 904	31.369
935	874 225	817 400 375	30.578	9.778	985	970 225	955 671 625	31.385
936	876 096	820 025 856	30.594	9.782	986	972 196	958 585 256	31.401
937	877 969	822 656 953	30.610	9.785	987	974 169	961 504 803	31.417
938	879 844	825 293 672	30.627	9.789	988	976 144	964 430 272	31.433
939	881 721	827 936 019	30.643	9.792	989	978 121	967 361 669	31.448
940	883 600	830 584 000	30.659	9.796	990	980 100	970 299 000	31.464
941	885 481	833 237 621	30.676	9.799	991	982 081	973 242 271	31.480
942	887 364	835 896 888	30.692	9.803	992	984 064	976 191 488	31.496
943	889 249	838 561 807	30.708	9.806	993	986 049	979 146 657	31.512
944	891 136	841 232 384	30.725	9.810	994	988 036	982 107 784	31 528
945	893 025	843 908 625	30.741	9.813	995	990 025	985 074 875	31.544
946	894 916	846 590 536	30.757	9.817	996	992 016	988 047 936	31.559
947	896 809	849 278 123	30.773	9.820	997	994 009	991 026 973	31.575
948	898 704	851 971 392	30.790	9.824	998	996 004	994 011 992	31.591
949	900 601	854 670 349	30.806	9.827	999	998 001	997 002 999	31.607

2. Areas of Circles, Diameters in Units and Eighths

0	1/8	1/4	3/8	1/2	5/8	3/4	7/8
0.0000	0.0123	0.0491	0.1104	0.1963	0.3068	0.4418	0.6013
0.7854	0.9940	1.2272	1.4849	1.7671	2.0739	2.4053	2.7612
3.1416	3.5466	3.9761	4.4301	4.9087	5.4119	5.9396	6.4918
7.0686	7.6699	8.2958	8.9462	9.6211	10.321	11.045	11.793
12.566	13.364	14.186	15.033	15.904	16.800	17.721	18.665
19.635	20.629	21.648	22.691	23.758	24.850	25.967	27.109
28.274	29.465	30.680	31.919	33.183	34.472	35.785	37.122
38.485	39.871	41.282	42.718	44.179	45.664	47.173	48.707
50.265	51.849	53.456	55.088	56.745	58.426	60.132	61.862
63.617	65.397	67.201	69.029	70.882	72.760	74.662	76.589
78.540	80.516	82.516	84.541	86.590	88.664	90.763	92.886
95.033	97.205	99.402	101.62	103.87	106.14	108.43	110.75
113.10	115.47	117.86	120.28	122.72	125.19	127.68	130.19
132.73	135.30	137.89	140.50	143.14	145.80	148.49	151.20
153.94	156.70	159.48	162.30	165.13	167.99	170.87	173.78
176.71	179.67	182.65	185.66	188.69	191.75	194.83	197.93
201.06	204.22	207.39	210.60	213.82	217.08	220.35	223.65
226.98	230.33	233.71	237.10	240.53	243.98	247.45	250.95
254.47	258.02	261.59	265.18	268.80	272.45	276.12	279.81
283.53	287.27	291.04	294.83	298.65	302.49	306.35	310.24
314.16	318.10	322.06	326.05	330.06	334.10	338.16	342.25
346.36	350.50	354.66	358.84	363.05	367.28	371.54	375.83
380.13	384.46	388.82	393.20	397.61	402.04	406.49	410.97
415.48	420.00	424.56	429.13	433.74	438.36	443.01	447.69
452.39	457.11	461.86	466.64	471.44	476.26	481.11	485.98
490.87	495.79	500.74	505.71	510.71	515.72	520.77	525.84
530.93	536.05	541.19	546.35	551.55	556.76	562.00	567.27
572.56	577.87	583.21	588.57	593.96	599.37	604.81	610.27
615.75	621.26	626.80	632.36	637.94	643.55	649.18	654.84
660.52	666.23	671.96	677.71	683.49	689.30	695.13	700.98
706.86	712.76	718.69	724.64	730.62	736.62	742.64	748.69
754.77	760.87	766.99	773.14	779.31	785.51	791.73	797.98
804.25	810.54	816.86	823.21	829.58	835.97	842.39	848.83
855.30	861.79	868.31	874.85	881.41	888.00	894.62	901.26
907.92	914.61	921.32	928.06	934.82	941.61	948.42	955.25
962.11	969.00	975.91	982.84	989.80	996.78	1003.8	1010.8
1017.9	1025.0	1032.1	1039.2	1046.3	1053.5	1060.7	1068.0
1075.2	1082.5	1089.8	1097.1	1104.5	1111.8	1119.2	1126.7
1134.1	1141.6	1149.1	1156.6	1164.2	1171.7	1179.3	1186.9
1194.6	1202.3	1210.0	1217.7	1225.4	1233.2	1241.0	1248.8
1256.6	1264.5	1272.4	1280.3	1288.2	1296.2	1304.2	1312.2
1320.3	1328.3	1336.4	1344.5	1352.7	1360.8	1369.0	1377.2
1385.4	1393.7	1402.0	1410.3	1418.6	1427.0	1435.4	1443.8
1452.2	1460.7	1469.1	1477.6	1486.2	1494.7	1503.3	1511.9
1520.5	1529.2	1537.9	1546.6	1555.3	1564.0	1572.8	1581.6

4. Common Logarithms

n	0	1	2	3	4	5	6	7	8	9
10	00000	00432	00860	01284	01703	02119	02531	02938	03342	03743
11	04139	04532	04922	05308	05690	06070	06446	06819	07188	07555
12	07918	08279	08636	08991	09342	09691	10037	10380	10721	11059
13	11394	11727	12057	12385	12710	13033	13354	13672	13988	14301
14	14613	14922	15229	15534	15836	16137	16435	16732	17026	17319
15	17609	17898	18184	18469	18752	19033	19312	19590	19866	20140
16	20412	20683	20952	21219	21484	21748	22011	22272	22531	22789
17	23045	23300	23553	23805	24055	24304	24551	24797	25042	25285
18	25527	25768	26007	26245	26482	26717	26951	27184	27416	27646
19	27875	28103	28330	28556	28780	29003	29226	29447	29667	29885
20	30103	30320	30535	30750	30963	31175	31387	31597	31806	32015
21	32222	32428	32634	32838	33041	33244	33445	33646	33846	34044
22	34242	34439	34635	34830	35025	35218	35411	35603	35793	35984
23	36173	36361	36549	36736	36922	37107	37291	37475	37658	37840
24	38021	38202	38382	38561	38739	38917	39094	39270	39445	39620
25	39794	39967	40140	40312	40483	40654	40824	40993	41162	41330
26	41497	41664	41830	41996	42160	42325	42488	42651	42813	42975
27	43136	43297	43457	43616	43775	43933	44091	44248	44404	44560
28	44716	44871	45025	45179	45332	45484	45637	45788	45939	46090
29	46240	46389	46538	46687	46835	46982	47129	47276	47422	47567
30	47712	47857	48001	48144	48287	48430	48572	48714	48855	48996
31	49136	49276	49415	49554	49693	49831	49969	50106	50243	50379
32	50515	50651	50786	50920	51055	51188	51322	51455	51587	51720
33	51851	51983	52114	52244	52375	52504	52634	52763	52892	53020
34	53148	53275	53403	53529	53656	53782	53908	54033	54155	54282
35	54407	54531	54654	54777	54900	55023	55145	55267	55388	55509
36	55630	55751	55871	55991	56110	56229	56348	56467	56585	56703
37	56820	56937	57054	57171	57287	57403	57519	57634	57749	57864
38	57978	58092	58206	58320	58433	58546	58659	58771	58883	58995
39	59106	59218	59329	59439	59550	59660	59770	59879	59988	60097
40	60206	60314	60423	60531	60638	60746	60853	60959	61066	61172
41	61278	61384	61490	61595	61700	61805	61909	62014	62118	62221
42	62325	62428	62531	62634	62737	62839	62941	63043	63144	63246
43	63347	63448	63548	63649	63749	63849	63949	64048	64147	64246
44	64345	64444	64542	64640	64738	64836	64933	65031	65128	65225
45	65321	65418	65514	65610	65706	65801	65896	65992	66087	66181
46	66276	66370	66464	66558	66652	66745	66839	66932	67025	67117
47	67210	67302	67394	67486	67578	67669	67761	67852	67943	68034
48	68124	68215	68305	68395	68485	68574	68664	68753	68842	68931
49	69020	69108	69197	69285	69373	69461	69548	69636	69723	69810
50	69897	69984	70070	70157	70243	70329	70415	70501	70586	70672
51	70757	70842	70927	71012	71096	71181	71265	71349	71433	71517
52	71600	71684	71767	71850	71933	72016	72099	72181	72263	72346
53	72428	72509	72591	72673	72754	72835	72916	72997	73078	73159
54	73239	73320	73400	73480	73560	73640	73719	73799	73878	73957
	0	1	2	3	4	5	6	7	8	9

nbers from 000 to 999

0	1	2	3	4	5	6	7	8	9
74036	74115	74194	74273	74351	74429	74507	74586	74663	74741
74819	74896	74974	75051	75128	75205	75282	75358	75435	75511
75587	75664	75740	75815	75891	75967	76042	76118	76193	76268
76343	76418	76492	76567	76641	76716	76790	76864	76938	77012
77085	77159	77232	77305	77379	77452	77525	77597	77670	77743
77815	77887	77960	78032	78104	78176	78247	78319	78390	78462
78533	78604	78675	78746	78817	78888	78958	79029	79099	79169
79239	79309	79379	79449	79518	79588	79657	79727	79796	79865
79934	80003	80072	80140	80209	80277	80346	80414	80482	80550
80618	80686	80754	80821	80889	80956	81023	81090	81158	81224
81291	81358	81425	81491	81558	81624	81690	81757	81823	81889
81954	82020	82086	82151	82217	82282	82347	82413	82478	82543
82607	82672	82737	82802	82866	82930	82995	83059	83123	83187
83251	83315	83378	83442	83506	83569	83632	83696	83759	83822
83885	83948	84011	84073	84136	84198	84261	84323	84386	84448
84510	84572	84634	84696	84757	84819	84880	84942	85003	85065
85126	85187	85248	85309	85370	85431	85491	85552	85612	85673
85733	85794	85854	85914	85974	86034	86094	86153	86213	86273
86332	86392	86451	86510	86570	86629	86688	86747	86806	86864
86923	86982	87040	87099	87157	87216	87274	87332	87390	87448
87506	87564	87622	87679	87737	87795	87852	87910	87967	88024
88081	88138	88195	88252	88309	88366	88423	88480	88536	88593
88649	88705	88762	88818	88874	88930	88986	89042	89098	89154
89209	89265	89321	89376	89432	89487	89542	89597	89653	89708
89763	89818	89873	89927	89982	90037	90091	90146	90200	90255
90309	90363	90417	90472	90526	90580	90634	90687	90741	90795
90849	90902	90956	91009	91062	91116	91169	91222	91275	91328
91381	91434	91487	91540	91593	91645	91698	91751	91803	91855
91908	91960	92012	92065	92117	92169	92221	92273	92324	92376
92428	92480	92531	92583	92634	92686	92737	92788	92840	92891
92942	92993	93044	93095	93146	93197	93247	93298	93349	93399
93450	93500	93551	93601	93651	93702	93752	93802	93852	93902
93952	94002	94052	94101	94151	94201	94250	94300	94349	94399
94448	94498	94547	94596	94645	94694	94743	94792	94841	94890
94939	94988	95036	95085	95134	95182	95231	95279	95328	95376
95424	95472	95521	95569	95617	95665	95713	95761	95809	95856
95904	95952	95999	96047	96095	96142	96190	96237	96284	96332
96379	96426	96473	96520	96567	96614	96661	96708	96755	96802
96848	96895	96942	96988	97035	97081	97128	97174	97220	97267
97313	97359	97405	97451	97497	97543	97589	97635	97681	97727
97772	97818	97864	97909	97955	98000	98046	98091	98137	98182
98227	98272	98318	98363	98408	98453	98498	98543	98588	98632
98677	98722	98767	98811	98856	98900	98945	98989	99034	99078
99123	99167	99211	99255	99300	99344	99388	99432	99476	99520
99564	99607	99651	99695	99739	99782	99826	99870	99913	99957

0	1	2	3	4	5	6	7	8

5. Common Logarithms

LOG SINE

Angle	0′	10′	20′	30′	40′	50′	60′	
0°	. — ∞	$\bar{3}$.46373	$\bar{3}$.76475	$\bar{3}$.94084	$\bar{2}$.06578	$\bar{2}$.16268	$\bar{2}$.24186	89
1	$\bar{2}$.24186	$\bar{2}$.30879	$\bar{2}$.36678	$\bar{2}$.41792	$\bar{2}$.46366	$\bar{2}$.50504	$\bar{2}$.54282	88
2	$\bar{2}$.54282	$\bar{2}$.57757	$\bar{2}$.60973	$\bar{2}$.63968	$\bar{2}$.66769	$\bar{2}$.69400	$\bar{2}$.71880	87
3	$\bar{2}$.71880	$\bar{2}$.74226	$\bar{2}$.76451	$\bar{2}$.78568	$\bar{2}$.80585	$\bar{2}$.82513	$\bar{2}$.84358	86
4	$\bar{2}$.84358	$\bar{2}$.86128	$\bar{2}$.87829	$\bar{2}$.89464	$\bar{2}$.91040	$\bar{2}$.92561	$\bar{2}$.94030	85
5°	$\bar{2}$.94030	$\bar{2}$.95450	$\bar{2}$.96825	$\bar{2}$.98157	$\bar{2}$.99450	$\bar{1}$.00704	$\bar{1}$.01923	84
6	$\bar{1}$.01923	$\bar{1}$.03109	$\bar{1}$.04262	$\bar{1}$.05386	$\bar{1}$.06481	$\bar{1}$.07548	$\bar{1}$.08589	83
7	$\bar{1}$.08589	$\bar{1}$.09606	$\bar{1}$.10599	$\bar{1}$.11570	$\bar{1}$.12519	$\bar{1}$.13447	$\bar{1}$.14356	82
8	$\bar{1}$.14356	$\bar{1}$.15245	$\bar{1}$.16116	$\bar{1}$.16970	$\bar{1}$.17807	$\bar{1}$.18628	$\bar{1}$.19433	81
9	$\bar{1}$.19433	$\bar{1}$.20223	$\bar{1}$.20999	$\bar{1}$.21761	$\bar{1}$.22509	$\bar{1}$.23244	$\bar{1}$.23967	80
10°	$\bar{1}$.23967	$\bar{1}$.24677	$\bar{1}$.25376	$\bar{1}$.26063	$\bar{1}$.26739	$\bar{1}$.27405	$\bar{1}$.28060	79
11	$\bar{1}$.28060	$\bar{1}$.28705	$\bar{1}$.29340	$\bar{1}$.29966	$\bar{1}$.30582	$\bar{1}$.31189	$\bar{1}$.31788	78
12	$\bar{1}$.31788	$\bar{1}$.32378	$\bar{1}$.32960	$\bar{1}$.33534	$\bar{1}$.34100	$\bar{1}$.34658	$\bar{1}$.35209	77
13	$\bar{1}$.35209	$\bar{1}$.35752	$\bar{1}$.36289	$\bar{1}$.36819	$\bar{1}$.37341	$\bar{1}$.37858	$\bar{1}$.38368	76
14	$\bar{1}$.38368	$\bar{1}$.38871	$\bar{1}$.39369	$\bar{1}$.39860	$\bar{1}$.40346	$\bar{1}$.40825	$\bar{1}$.41300	75
15°	$\bar{1}$.41300	$\bar{1}$.41768	$\bar{1}$.42232	$\bar{1}$.42690	$\bar{1}$.43143	$\bar{1}$.43591	$\bar{1}$.44034	74
16	$\bar{1}$.44034	$\bar{1}$.44472	$\bar{1}$.44905	$\bar{1}$.45334	$\bar{1}$.45758	$\bar{1}$.46178	$\bar{1}$.46594	73
17	$\bar{1}$.46594	$\bar{1}$.47005	$\bar{1}$.47411	$\bar{1}$.47814	$\bar{1}$.48213	$\bar{1}$.48607	$\bar{1}$.48998	72
18	$\bar{1}$.48998	$\bar{1}$.49385	$\bar{1}$.49768	$\bar{1}$.50148	$\bar{1}$.50523	$\bar{1}$.50896	$\bar{1}$.51264	71
19	$\bar{1}$.51264	$\bar{1}$.51629	$\bar{1}$.51991	$\bar{1}$.52350	$\bar{1}$.52705	$\bar{1}$.53056	$\bar{1}$.53405	70
20°	$\bar{1}$.53405	$\bar{1}$.53751	$\bar{1}$.54093	$\bar{1}$.54433	$\bar{1}$.54769	$\bar{1}$.55102	$\bar{1}$.55433	69
21	$\bar{1}$.55433	$\bar{1}$.55761	$\bar{1}$.56085	$\bar{1}$.56408	$\bar{1}$.56727	$\bar{1}$.57044	$\bar{1}$.57358	68
22	$\bar{1}$.57358	$\bar{1}$.57669	$\bar{1}$.57978	$\bar{1}$.58284	$\bar{1}$.58588	$\bar{1}$.58889	$\bar{1}$.59188	67
23	$\bar{1}$.59188	$\bar{1}$.59484	$\bar{1}$.59778	$\bar{1}$.60070	$\bar{1}$.60359	$\bar{1}$.60646	$\bar{1}$.60931	66
24	$\bar{1}$.60931	$\bar{1}$.61214	$\bar{1}$.61494	$\bar{1}$.61773	$\bar{1}$.62049	$\bar{1}$.62323	$\bar{1}$.62595	65
25°	$\bar{1}$.62595	$\bar{1}$.62865	$\bar{1}$.63133	$\bar{1}$.63398	$\bar{1}$.63662	$\bar{1}$.63924	$\bar{1}$.64184	64
26	$\bar{1}$.64184	$\bar{1}$.64442	$\bar{1}$.64698	$\bar{1}$.64953	$\bar{1}$.65205	$\bar{1}$.65456	$\bar{1}$.65705	63
27	$\bar{1}$.65705	$\bar{1}$.65952	$\bar{1}$.66197	$\bar{1}$.66441	$\bar{1}$.66682	$\bar{1}$.66922	$\bar{1}$.67161	62
28	$\bar{1}$.67161	$\bar{1}$.67398	$\bar{1}$.67633	$\bar{1}$.67866	$\bar{1}$.68098	$\bar{1}$.68328	$\bar{1}$.68557	61
29	$\bar{1}$.68557	$\bar{1}$.68784	$\bar{1}$.69010	$\bar{1}$.69234	$\bar{1}$.69456	$\bar{1}$.69677	$\bar{1}$.69897	60
30°	$\bar{1}$.69897	$\bar{1}$.70115	$\bar{1}$.70332	$\bar{1}$.70547	$\bar{1}$.70761	$\bar{1}$.70973	$\bar{1}$.71184	59
31	$\bar{1}$.71184	$\bar{1}$.71393	$\bar{1}$.71602	$\bar{1}$.71809	$\bar{1}$.72014	$\bar{1}$.72218	$\bar{1}$.72421	58
32	$\bar{1}$.72421	$\bar{1}$.72622	$\bar{1}$.72823	$\bar{1}$.73022	$\bar{1}$.73219	$\bar{1}$.73416	$\bar{1}$.73611	57
33	$\bar{1}$.73611	$\bar{1}$.73805	$\bar{1}$.73997	$\bar{1}$.74189	$\bar{1}$.74379	$\bar{1}$.74568	$\bar{1}$.74756	56
34	$\bar{1}$.74756	$\bar{1}$.74943	$\bar{1}$.75128	$\bar{1}$.75313	$\bar{1}$.75496	$\bar{1}$.75678	$\bar{1}$.75859	55
35°	$\bar{1}$.75859	$\bar{1}$.76039	$\bar{1}$.76218	$\bar{1}$.76395	$\bar{1}$.76572	$\bar{1}$.76747	$\bar{1}$.76922	54
36	$\bar{1}$.76922	$\bar{1}$.77095	$\bar{1}$.77268	$\bar{1}$.77439	$\bar{1}$.77609	$\bar{1}$.77778	$\bar{1}$.77946	53
37	$\bar{1}$.77946	$\bar{1}$.78113	$\bar{1}$.78280	$\bar{1}$.78445	$\bar{1}$.78609	$\bar{1}$.78772	$\bar{1}$.78934	52
38	$\bar{1}$.78934	$\bar{1}$.79095	$\bar{1}$.79256	$\bar{1}$.79415	$\bar{1}$.79573	$\bar{1}$.79731	$\bar{1}$.79887	51
39	$\bar{1}$.79887	$\bar{1}$.80043	$\bar{1}$.80197	$\bar{1}$.80351	$\bar{1}$.80504	$\bar{1}$.80656	$\bar{1}$.80807	50°
40°	$\bar{1}$.80807	$\bar{1}$.80957	$\bar{1}$.81106	$\bar{1}$.81254	$\bar{1}$.81402	$\bar{1}$.81549	$\bar{1}$.81694	49
41	$\bar{1}$.81694	$\bar{1}$.81839	$\bar{1}$.81983	$\bar{1}$.82126	$\bar{1}$.82269	$\bar{1}$.82410	$\bar{1}$.82551	48
42	$\bar{1}$.82551	$\bar{1}$.82691	$\bar{1}$.82830	$\bar{1}$.82968	$\bar{1}$.83106	$\bar{1}$.83242	$\bar{1}$.83378	47
43	$\bar{1}$.83378	$\bar{1}$.83513	$\bar{1}$.83648	$\bar{1}$.83781	$\bar{1}$.83914	$\bar{1}$.84046	$\bar{1}$.84177	46
44	$\bar{1}$.84177	$\bar{1}$.84308	$\bar{1}$.84437	$\bar{1}$.84566	$\bar{1}$.84694	$\bar{1}$.84822	$\bar{1}$.84949	45°
	60′	50′	40′	30′	20′	10′	0′	Angle

LOG COSINE

Sines and Cosines

LOG SINE

ngle	0′	10′	20′	30′	40′	50′	60′	
15°	T.84949	T.85074	T.85200	T.85324	T.85448	T.85571	T.85693	44
16	T.85693	T.85815	T.85936	T.86056	T.86176	T.86295	T.86413	43
17	T.86413	T.86530	T.86647	T.86763	T.86879	T.86993	T.87107	42
18	T.87107	T.87221	T.87334	T.87446	T.87557	T.87668	T.87778	41
19	T.87778	T.87887	T.87996	T.88105	T.88212	T.88319	T.88425	40°
20°	T.88425	T.88531	T.88636	T.88741	T.88844	T.88948	T.89050	39
21	T.89050	T.89152	T.89254	T.89354	T.89455	T.89554	T.89653	38
22	T.89653	T.89752	T.89849	T.89947	T.90043	T.90139	T.90235	37
23	T.90235	T.90330	T.90424	T.90518	T.90611	T.90704	T.90796	36
24	T.90796	T.90887	T.90978	T.91069	T.91158	T.91248	T.91336	35°
25°	T.91336	T.91425	T.91512	T.91599	T.91686	T.91772	T.91857	34
26	T.91857	T.91942	T.92027	T.92111	T.92194	T.92277	T.92359	33
27	T.92359	T.92441	T.92522	T.92603	T.92683	T.92763	T.92842	32
28	T.92842	T.92921	T.92999	T.93077	T.93154	T.93230	T.93307	31
29	T.93307	T.93382	T.93457	T.93532	T.93606	T.93680	T.93753	30°
30°	T.93753	T.93826	T.93898	T.93970	T.94041	T.94112	T.94182	29
31	T.94182	T.94252	T.94321	T.94390	T.94458	T.94526	T.94593	28
32	T.94593	T.94660	T.94727	T.94793	T.94858	T.94923	T.94988	27
33	T.94988	T.95052	T.95116	T.95179	T.95242	T.95304	T.95366	26
34	T.95366	T.95427	T.95488	T.95549	T.95609	T.95668	T.95728	25°
35°	T.95728	T.95786	T.95844	T.95902	T.95960	T.96017	T.96073	24
36	T.96073	T.96129	T.96185	T.96240	T.96294	T.96349	T.96403	23
37	T.96403	T.96456	T.96509	T.96562	T.96614	T.96665	T.96717	22
38	T.96717	T.96767	T.96818	T.96868	T.96917	T.96966	T.97015	21
39	T.97015	T.97063	T.97111	T.97159	T.97206	T.97252	T.97299	20′
40°	T.97299	T.97344	T.97390	T.97435	T.97479	T.97523	T.97567	19
41	T.97567	T.97610	T.97653	T.97696	T.97738	T.97779	T.97821	18
42	T.97821	T.97861	T.97902	T.97942	T.97982	T.98021	T.98060	17
43	T.98060	T.98098	T.98136	T.98174	T.98211	T.98248	T.98284	16
44	T.98284	T.98320	T.98356	T.98391	T.98426	T.98460	T.98494	15°
45°	T.98494	T.98528	T.98561	T.98594	T.98627	T.98659	T.98690	14
46	T.98690	T.98722	T.98753	T.98783	T.98813	T.98843	T.98872	13
47	T.98872	T.98901	T.98930	T.98958	T.98986	T.99013	T.99040	12
48	T.99040	T.99067	T.99093	T.99119	T.99145	T.99170	T.99195	11
49	T.99195	T.99219	T.99243	T.99267	T.99290	T.99313	T.99335	10°
50°	T.99335	T.99357	T.99379	T.99400	T.99421	T.99442	T.99462	9
51	T.99462	T.99482	T.99501	T.99520	T.99539	T.99557	T.99575	8
52	T.99575	T.99593	T.99610	T.99627	T.99643	T.99659	T.99675	7
53	T.99675	T.99690	T.99705	T.99720	T.99734	T.99748	T.99761	6
54	T.99761	T.99775	T.99787	1.99800	T.99812	T.99823	T.99834	5°
55°	T.99834	T.99845	T.99856	T.99866	T.99876	T.99885	T.99894	4
56	T.99894	T.99903	T.99911	T.99919	T.99926	T.99934	T.99940	3
57	T.99940	T.99947	T.99953	T.99959	T.99964	T.99969	T.99974	2
58	T.99974	T.99978	T.99982	T.99985	T.99988	T.99991	T.99993	1
59	T.99993	T.99995	T.99997	T.99998	T.99999	0.00000	0.00000	0°
	60′	50′	40′	30′	20′	10′	0′	

LOG COSINE

LOG TANGENT

Angle	0′	10′	20′	30′	40′	50′	60′	
0°	— ∞	$\bar{3}$.46373	$\bar{3}$.76476	$\bar{3}$.94086	$\bar{2}$.06581	$\bar{2}$.16273	$\bar{2}$.24192	89
1	$\bar{2}$.24192	$\bar{2}$.30888	$\bar{2}$.36689	$\bar{2}$.41807	$\bar{2}$.46385	$\bar{2}$.50527	$\bar{2}$.54308	88
2	$\bar{2}$.54308	$\bar{2}$.57788	$\bar{2}$.61009	$\bar{2}$.64009	$\bar{2}$.66816	$\bar{2}$.69453	$\bar{2}$.71940	87
3	$\bar{2}$.71940	$\bar{2}$.74292	$\bar{2}$.76525	$\bar{2}$.78649	$\bar{2}$.80674	$\bar{2}$.82610	$\bar{2}$.84464	86
4	$\bar{2}$.84464	$\bar{2}$.86243	$\bar{2}$.87953	$\bar{2}$.89598	$\bar{2}$.91185	$\bar{2}$.92716	$\bar{2}$.94195	85
5°	$\bar{2}$.94195	$\bar{2}$.95627	$\bar{2}$.97013	$\bar{2}$.98358	$\bar{2}$.99662	$\bar{1}$.00930	$\bar{1}$.02162	84
6	$\bar{1}$.02162	$\bar{1}$.03361	$\bar{1}$.04528	$\bar{1}$.05666	$\bar{1}$.06775	$\bar{1}$.07858	$\bar{1}$.08914	83
7	$\bar{1}$.08914	$\bar{1}$.09947	$\bar{1}$.10956	$\bar{1}$.11943	$\bar{1}$.12909	$\bar{1}$.13854	$\bar{1}$.14780	82
8	$\bar{1}$.14780	$\bar{1}$.15688	$\bar{1}$.16577	$\bar{1}$.17450	$\bar{1}$.18306	$\bar{1}$.19146	$\bar{1}$.19971	81
9	$\bar{1}$.19971	$\bar{1}$.20782	$\bar{1}$.21578	$\bar{1}$.22361	$\bar{1}$.23130	$\bar{1}$.23887	$\bar{1}$.24632	80
10°	$\bar{1}$.24632	$\bar{1}$.25365	$\bar{1}$.26086	$\bar{1}$.26797	$\bar{1}$.27496	$\bar{1}$.28186	$\bar{1}$.28865	79
11	$\bar{1}$.28865	$\bar{1}$.29535	$\bar{1}$.30195	$\bar{1}$.30846	$\bar{1}$.31489	$\bar{1}$.32122	$\bar{1}$.32747	78
12	$\bar{1}$.32747	$\bar{1}$.33365	$\bar{1}$.33974	$\bar{1}$.34576	$\bar{1}$.35170	$\bar{1}$.35757	$\bar{1}$.36336	77
13	$\bar{1}$.36336	$\bar{1}$.36909	$\bar{1}$.37476	$\bar{1}$.38035	$\bar{1}$.38589	$\bar{1}$.39136	$\bar{1}$.39677	76
14	$\bar{1}$.39677	$\bar{1}$.40212	$\bar{1}$.40742	$\bar{1}$.41266	$\bar{1}$.41784	$\bar{1}$.42297	$\bar{1}$.42805	75
15°	$\bar{1}$.42805	$\bar{1}$.43308	$\bar{1}$.43806	$\bar{1}$.44299	$\bar{1}$.44787	$\bar{1}$.45271	$\bar{1}$.45750	74
16	$\bar{1}$.45750	$\bar{1}$.46224	$\bar{1}$.46694	$\bar{1}$.47160	$\bar{1}$.47622	$\bar{1}$.48080	$\bar{1}$.48534	73
17	$\bar{1}$.48534	$\bar{1}$.48984	$\bar{1}$.49430	$\bar{1}$.49872	$\bar{1}$.50311	$\bar{1}$.50746	$\bar{1}$.51178	72
18	$\bar{1}$.51178	$\bar{1}$.51606	$\bar{1}$.52031	$\bar{1}$.52452	$\bar{1}$.52870	$\bar{1}$.53285	$\bar{1}$.53697	71
19	$\bar{1}$.53697	$\bar{1}$.54106	$\bar{1}$.54512	$\bar{1}$.54915	$\bar{1}$.55315	$\bar{1}$.55712	$\bar{1}$.56107	70
20°	$\bar{1}$.56107	$\bar{1}$.56498	$\bar{1}$.56887	$\bar{1}$.57274	$\bar{1}$.57658	$\bar{1}$.58039	$\bar{1}$.58418	69
21	$\bar{1}$.58418	$\bar{1}$.58794	$\bar{1}$.59168	$\bar{1}$.59540	$\bar{1}$.59909	$\bar{1}$.60276	$\bar{1}$.60641	68
22	$\bar{1}$.60641	$\bar{1}$.61004	$\bar{1}$.61364	$\bar{1}$.61722	$\bar{1}$.62079	$\bar{1}$.62433	$\bar{1}$.62785	67
23	$\bar{1}$.62785	$\bar{1}$.63135	$\bar{1}$.63484	$\bar{1}$.63830	$\bar{1}$.64175	$\bar{1}$.64517	$\bar{1}$.64858	66
24	$\bar{1}$.64858	$\bar{1}$.65197	$\bar{1}$.65535	$\bar{1}$.65870	$\bar{1}$.66204	$\bar{1}$.66537	$\bar{1}$.66867	65
25°	$\bar{1}$.66867	$\bar{1}$.67196	$\bar{1}$.67524	$\bar{1}$.67850	$\bar{1}$.68174	$\bar{1}$.68497	$\bar{1}$.68818	64
26	$\bar{1}$.68818	$\bar{1}$.69138	$\bar{1}$.69457	$\bar{1}$.69774	$\bar{1}$.70089	$\bar{1}$.70404	$\bar{1}$.70717	63
27	$\bar{1}$.70717	$\bar{1}$.71028	$\bar{1}$.71339	$\bar{1}$.71648	$\bar{1}$.71955	$\bar{1}$.72262	$\bar{1}$.72567	62
28	$\bar{1}$.72567	$\bar{1}$.72872	$\bar{1}$.73175	$\bar{1}$.73476	$\bar{1}$.73777	$\bar{1}$.74077	$\bar{1}$.74375	61
29	$\bar{1}$.74375	$\bar{1}$.74673	$\bar{1}$.74969	$\bar{1}$.75264	$\bar{1}$.75558	$\bar{1}$.75852	$\bar{1}$.76144	60
30°	$\bar{1}$.76144	$\bar{1}$.76435	$\bar{1}$.76725	$\bar{1}$.77015	$\bar{1}$.77303	$\bar{1}$.77591	$\bar{1}$.77877	59
31	$\bar{1}$.77877	$\bar{1}$.78163	$\bar{1}$.78448	$\bar{1}$.78732	$\bar{1}$.79015	$\bar{1}$.79297	$\bar{1}$.79579	58
32	$\bar{1}$.79579	$\bar{1}$.79860	$\bar{1}$.80140	$\bar{1}$.80419	$\bar{1}$.80697	$\bar{1}$.80975	$\bar{1}$.81252	57
33	$\bar{1}$.81252	$\bar{1}$.81528	$\bar{1}$.81803	$\bar{1}$.82078	$\bar{1}$.82352	$\bar{1}$.82626	$\bar{1}$.82899	56
34	$\bar{1}$.82899	$\bar{1}$.83171	$\bar{1}$.83442	$\bar{1}$.83713	$\bar{1}$.83984	$\bar{1}$.84254	$\bar{1}$.84523	55
35°	$\bar{1}$.84523	$\bar{1}$.84791	$\bar{1}$.85059	$\bar{1}$.85327	$\bar{1}$.85594	$\bar{1}$.85860	$\bar{1}$.86126	54
36	$\bar{1}$.86126	$\bar{1}$.86392	$\bar{1}$.86656	$\bar{1}$.86921	$\bar{1}$.87185	$\bar{1}$.87448	$\bar{1}$.87711	53
37	$\bar{1}$.87711	$\bar{1}$.87974	$\bar{1}$.88236	$\bar{1}$.88498	$\bar{1}$.88759	$\bar{1}$.89020	$\bar{1}$.89281	52
38	$\bar{1}$.89281	$\bar{1}$.89541	$\bar{1}$.89801	$\bar{1}$.90061	$\bar{1}$.90320	$\bar{1}$.90578	$\bar{1}$.90837	51
39	$\bar{1}$.90837	$\bar{1}$.91095	$\bar{1}$.91353	$\bar{1}$.91610	$\bar{1}$.91868	$\bar{1}$.92125	$\bar{1}$.92381	50
40°	$\bar{1}$.92381	$\bar{1}$.92638	$\bar{1}$.92894	$\bar{1}$.93150	$\bar{1}$.93406	$\bar{1}$.93661	$\bar{1}$.93916	49
41	$\bar{1}$.93916	$\bar{1}$.94171	$\bar{1}$.94426	$\bar{1}$.94681	$\bar{1}$.94935	$\bar{1}$.95190	$\bar{1}$.95444	48
42	$\bar{1}$.95444	$\bar{1}$.95698	$\bar{1}$.95952	$\bar{1}$.96205	$\bar{1}$.96459	$\bar{1}$.96712	$\bar{1}$.96966	47
43	$\bar{1}$.96966	$\bar{1}$.97219	$\bar{1}$.97472	$\bar{1}$.97725	$\bar{1}$.97978	$\bar{1}$.98231	$\bar{1}$.98484	46
44	$\bar{1}$.98484	$\bar{1}$.98737	$\bar{1}$.98989	$\bar{1}$.99242	$\bar{1}$.99495	$\bar{1}$.99747	0.00000	45
	60′	50′	40′	30′	20′	10′	0′	Angle

LOG COTANGENT

ngents and Cotangents

LOG TANGENT

°	0′	10′	20′	30′	40′	50′	60′	
	0.00000	0.00253	0.00505	0.00758	0.01011	0.01263	0.01516	44
	0.01516	0.01769	0.02022	0.02275	0.02528	0.02781	0.03034	43
	0.03034	0.03288	0.03541	0.03795	0.04048	0.04302	0.04556	42
	0.04556	0.04810	0.05065	0.05319	0.05574	0.05829	0.06084	41
	0.06084	0.06339	0.06594	0.06850	0.07106	0.07362	0.07619	40°
	0.07619	0.07875	0.08132	0.08390	0.08647	0.08905	0.09163	39
	0.09163	0.09422	0.09680	0.09939	0.10199	0.10459	0.10719	38
	0.10719	0.10980	0.11241	0.11502	0.11764	0.12026	0.12289	37
	0.12289	0.12552	0.12815	0.13079	0.13344	0.13608	0.13874	36
	0.13874	0.14140	0.14406	0.14673	0.14941	0.15209	0.15477	35°
	0.15477	0.15746	0.16016	0.16287	0.16558	0.16829	0.17101	34
	0.17101	0.17374	0.17648	0.17922	0.18197	0.18472	0.18748	33
	0.18748	0.19025	0.19303	0.19581	0.19860	0.20140	0.20421	32
	0.20421	0.20703	2.20985	0.21268	0.21552	0.21837	0.22123	31
	0.22123	0.22409	0.22697	0.22985	0.23275	0.23565	0.23856	30°
	0.23856	0.24148	0.24442	0.24736	0.25031	0.25327	0.25625	29
	0.25625	0.25923	0.26223	0.26524	0.26825	0.27128	0.27433	28
	0.27433	0.27738	0.28045	0.28352	0.28661	0.28972	0.29283	27
	0.29283	0.29596	0.29911	0.30226	0.30543	0.30862	0.31182	26
	0.31182	0.31503	0.31826	0.32150	0.32476	0.32804	0.33133	25°
	0.33133	0.33463	0.33796	0.34130	0.34465	0.34803	0.35142	24
	0.35142	0.35483	0.35825	0.36170	0.36516	0.36865	0.37215	23
	0.37215	0.37567	0.37921	0.38278	0.38636	0.38996	0.39359	22
	0.39359	0.39724	0.40091	0.40460	0.40832	0.41206	0.41582	21
	0.41582	0.41961	0.42342	0.42726	0.43113	0.43502	0.43893	20°
	0.43893	0.44288	9.44685	0.45085	0.45488	0.45894	0.46303	19
	0.46303	0.46715	0.47130	0.47548	0.47969	0.48394	0.48822	18
	0.48822	0.49254	0.49689	0.50128	0.50570	0.51016	0.51466	17
	0.51466	0.51920	0.52378	0.52840	0.53306	0.53776	0.54250	16
	0.54250	0.54729	0.55213	0.55701	0.56194	0.56692	0.57195	15°
	0.57195	0.57703	0.58216	0.58734	0.59258	0.59788	0.60323	14
	0.60323	0.60864	0.61411	0.61965	0.62524	0.63091	0.63664	13
	0.63664	0.64243	0.64830	0.65424	0.66026	0.66635	0.67253	12
	0.67253	0.67878	0.68511	0.69154	0.69805	0.70465	0.71135	11
	0.71135	0.71814	0.72504	0.73203	0.73914	0.74635	0.75368	10°
	0.75368	0.76113	0.76870	0.77639	0.78422	0.79218	0.80029	9
	0.80029	0.80854	0.81694	0.82550	0.83423	0.84312	0.85220	8
	0.85220	0.86146	0.87091	0.88057	0.89044	0.90053	0.91086	7
	0.91086	0.92142	0.93225	0.94334	0.95472	0.96639	0.97838	6
	0.97838	0.99070	1.00338	1.01642	1.02987	1.04373	1.05805	5°
	1.05805	1.07284	1.08815	1.10402	1.12047	1.13757	1.15536	4
	1.15536	1.17390	1.19326	1.21351	1.23475	1.25708	1.28060	3
	1.28060	1.30547	1.33184	1.35991	1.38991	1.42212	1.45692	2
	1.45692	1.49473	1.53615	1.58193	1.63311	1.69112	1.75808	1
	1.75808	1.83727	1.93419	2.05914	2.23524	2.53627	∞	0°

7. Natural Sines

SINE

Angle	0′	10′	20′	30′	40′	50′	60′	
0°	0.00000	0.00291	0.00582	0.00873	0.01164	0.01454	0.01745	89
1	0.01745	0.02036	0.02327	0.02618	0.02908	0.03199	0.03490	88
2	0.03490	0.03781	0.04071	0.04362	0.04653	0.04943	0.05234	87
3	0.05234	0.05524	0.05814	0.06105	0.06395	0.06685	0.06976	86
4	0.06976	0.07266	0.07556	0.07846	0.08136	0.08426	0.08716	85
5°	0.08716	0.09005	0.09295	0.09585	0.09874	0.10164	0.10453	84
6	0.10453	0.10742	0.11031	0.11320	0.11609	0.11898	0.12187	83
7	0.12187	0.12476	0.12764	0.13053	0.13341	0.13629	0.13917	82
8	0.13917	0.14205	0.14493	0.14781	0.15069	0.15356	0.15643	81
9	0.15643	0.15931	0.16218	0.16505	0.16792	0.17078	0.17365	80
10°	0.17365	0.17651	0.17937	0.18224	0.18509	0.18795	0.19081	79
11	0.19081	0.19366	0.19652	0.19937	0.20222	0.20507	0.20791	78
12	0.20791	0.21076	0.21360	0.21644	0.21928	0.22212	0.22495	77
13	0.22495	0.22778	0.23062	0.23345	0.23627	0.23910	0.24192	76
14	0.24192	0.24474	0.24756	0.25038	0.25320	0.25601	0.25882	75
15°	0.25882	0.26163	0.26443	0.26724	0.27004	0.27284	0.27564	74
16	0.27564	0.27843	0.28123	0.28402	0.28680	0.28959	0.29237	73
17	0.29237	0.29515	0.29793	0.30071	0.30348	0.30625	0.30902	72
18	0.30902	0.31178	0.31454	0.31730	0.32006	0.32282	0.32557	71
19	0.32557	0.32832	0.33106	0.33381	0.33655	0.33929	0.34202	70
20°	0.34202	0.34475	0.34748	0.35021	0.35293	0.35565	0.35837	69
21	0.35837	0.36108	0.36379	0.36650	0.36921	0.37191	0.37461	68
22	0.37461	0.37730	0.37999	0.38268	0.38537	0.38805	0.39073	67
23	0.39073	0.39341	0.39608	0.39875	0.40142	0.40408	0.40674	66
24	0.40674	0.40939	0.41204	0.41469	0.41734	0.41998	0.42262	65
25°	0.42262	0.42525	0.42788	0.43051	0.43313	0.43575	0.43837	64
26	0.43837	0.44098	0.44359	0.44620	0.44880	0.45140	0.45399	63
27	0.45399	0.45658	0.45917	0.46175	0.46433	0.46690	0.46947	62
28	0.46947	0.47204	0.47460	0.47716	0.47971	0.48226	0.48481	61
29	0.48481	0.48735	0.48989	0.49242	0.49495	0.49748	0.50000	60°
30°	0.50000	0.50252	0.50503	0.50754	0.51004	0.51254	0.51504	59
31	0.51504	0.51753	0.52002	0.52250	0.52498	0.52745	0.52992	58
32	0.52992	0.53238	0.53484	0.53730	0.53975	0.54220	0.54464	57
33	0.54464	0.54708	0.54951	0.55194	0.55436	0.55678	0.55919	56
34	0.55919	0.56160	0.56401	0.56641	0.56880	0.57119	0.57358	55
35°	0.57358	0.57596	0.57833	0.58070	0.58307	0.58543	0.58779	54
36	0.58779	0.59014	0.59248	0.59482	0.59716	0.59949	0.60182	53
37	0.60182	0.60414	0.60645	0.60876	0.61107	0.61337	0.61566	52
38	0.61566	0.61795	0.62024	0.62251	0.62479	0.62706	0.62932	51
39	0.62932	0.63158	0.63383	0.63608	0.63832	0.64056	0.64279	50
40°	0.64279	0.64501	0.64723	0.64945	0.65166	0.65386	0.65606	49
41	0.65606	0.65825	0.66044	0.66262	0.66480	0.66697	0.66913	48
42	0.66913	0.67129	0.67344	0.67559	0.67773	0.67987	0.68200	47
43	0.68200	0.68412	0.68624	0.68835	0.69046	0.69256	0.69466	46
44	0.69466	0.69675	0.69883	0.70091	0.70298	0.70505	0.70711	45
	60′	50′	40′	30′	20′	10′	0′	Angle

COSINE.

Cosines

SINE

′	0′	10′	20′	30′	40′	50′	60′	
	0.70711	0.70916	0.71121	0.71325	0.71529	0.71732	0.71934	44
	0.71934	0.72136	0.72337	0.72537	0.72737	0.72937	0.73135	43
	0.73135	0.73333	0.73531	0.73728	0.73924	0.74120	0.74314	42
	0.74314	0.74509	0.74703	0.74896	0.75088	0.75280	0.75471	41
	0.75471	0.75661	0.75851	0.76041	0.76229	0.76417	0.76604	40°
	0.76604	0.76791	0.76977	0.77162	0.77347	0.77531	0.77715	39
	0.77715	0.77897	0.78079	0.78261	0.78442	0.78622	0.78801	38
	0.78801	0.78980	0.79158	0.79335	0.79512	0.79688	0.79864	37
	0.79864	0.80038	0.80212	0.80386	0.80558	0.80730	0.80902	36
	0.80902	0.81072	0.81242	0.81412	0.81580	0.81748	0.81915	35°
	0.81915	0.82082	0.82248	0.82413	0.82577	0.82741	0.82904	34
	0.82904	0.83066	0.83228	0.83389	0.83549	0.83708	0.83867	33
	0.83867	0.84025	0.84182	0.84339	0.84495	0.84650	0.84805	32
	0.84805	0.84959	0.85112	0.85264	0.85416	0.85567	0.85717	31
	0.85717	0.85866	0.86015	0.86163	0.86310	0.86457	0.86603	30°
	0.86603	0.86748	0.86892	0.87036	0.87178	0.87321	0.87462	29
	0.87462	0.87603	0.87743	0.87882	0.88020	0.88158	0.88295	28
	0.88295	0.88431	0.88566	0.88701	0.88835	0.88968	0.89101	27
	0.89101	0.89232	0.89363	0.89493	0.89623	0.89752	0.89879	26
	0.89879	0.90007	0.90133	0.90259	0.90383	0.90507	0.90631	25°
	0.90631	0.90753	0.90875	0.90996	0.91116	0.91236	0.91355	24
	0.91355	0.91472	0.91590	0.91706	0.91822	0.91936	0.92050	23
	0.92050	0.92164	0.92276	0.92388	0.92499	0.92609	0.92718	22
	0.92718	0.92827	0.92935	0.93042	0.93148	0.93253	0.93358	21
	0.93358	0.93462	0.93565	0.93667	0.93769	0.93869	0.93969	20°
	0.93969	0.94068	0.94167	0.94264	0.94361	0.94457	0.94552	19
	0.94552	0.94646	0.94740	0.94832	0.94924	0.95015	0.95106	18
	0.95106	0.95195	0.95284	0.95372	0.95459	0.95545	0.95630	17
	0.95630	0.95715	0.95799	0.95882	0.95964	0.96046	0.96126	16
	0.96126	0.96206	0.96285	0.96363	0.96440	0.96517	0.96593	15°
	0.96593	0.96667	0.96742	0.96815	0.96887	0.96959	0.97030	14
	0.97030	0.97100	0.97169	0.97237	0.97304	0.97371	0.97437	13
	0.97437	0.97502	0.97566	0.97630	0.97692	0.97754	0.97815	12
	0.97815	0.97875	0.97934	0.97992	0.98050	0.98107	0.98163	11
	0.98163	0.98218	0.98272	0.98325	0.98378	0.98430	0.98481	10°
	0.98481	0.98531	0.98580	0.98629	0.98676	0.98723	0.98769	9
	0.98769	0.98814	0.98858	0.98902	0.98944	0.98986	0.99027	8
	0.99027	0.99067	0.99106	0.99144	0.99182	0.99219	0.99255	7
	0.99255	0.99290	0.99324	0.99357	0.99390	0.99421	0.99452	6
	0.99452	0.99482	0.99511	0.99540	0.99567	0.99594	0.99619	5°
	0.99619	0.99644	0.99668	0.99692	0.99714	0.99736	0.99756	4
	0.99756	0.99776	0.99795	0.99813	0.99831	0.99847	0.99863	3
	0.99863	0.99878	0.99892	0.99905	0.99917	0.99929	0.99939	2
	0.99939	0.99949	0.99958	0.99966	0.99973	0.99979	0.99985	1
	0.99985	0.99989	0.99993	0.99996	0.99998	1.00000	1.00000	0°
′	60′	50′	40′	30′	20′	10′	0′	Ar

COSINE

TANGENT

Degrees	0′	10′	20′	30′	40′	50′
1°	[illegible]	[illegible]	[illegible]	[illegible]	0.011664	0.014[illegible]
2	[illegible]	[illegible]	[illegible]	[illegible]	0.023210	0.0[illegible]
3	[illegible]	[illegible]	[illegible]	[illegible]	0.046558	0.04[illegible]
4	[illegible]	[illegible]	[illegible]	[illegible]	0.064208	0.06[illegible]
5°	[illegible]	[illegible]	[illegible]	[illegible]	0.081663	0.08[illegible]
6	[illegible]	[illegible]	[illegible]	0.098629	0.099923	0.10[illegible]
7	[illegible]	[illegible]	[illegible]	0.11394	0.116858	0.11[illegible]
8	[illegible]	[illegible]	[illegible]	0.13165	0.134611	0.13[illegible]
9	[illegible]	[illegible]	[illegible]	0.14945	0.15243	0.15[illegible]
10°	[illegible]	[illegible]	[illegible]	0.16734	0.17033	0.17[illegible]
11	[illegible]	[illegible]	0.18233	0.18534	0.18835	0.19[illegible]
12	[illegible]	[illegible]	0.20042	0.20345	0.20648	0.20[illegible]
13	[illegible]	[illegible]	0.21864	0.22169	0.22475	0.22[illegible]
14	[illegible]	[illegible]	0.23700	0.24008	0.24316	0.24624
15°	[illegible]	[illegible]	0.25552	0.25862	0.26172	0.26483
16	[illegible]	0.27127	0.27419	0.27732	0.28046	0.28360
17	[illegible]	0.29090	0.29305	0.29621	0.29938	0.30255
18	[illegible]	0.30991	0.31210	0.31530	0.31850	0.32171
19	[illegible]	0.32924	0.33136	0.33460	0.33783	0.34108
20°	[illegible]	0.34755	0.35085	0.35412	0.35740	0.36069
21	[illegible]	[illegible]	0.37057	0.37388	0.37720	0.38053
22	[illegible]	[illegible]	[illegible]	0.39391	0.39727	0.40065
23	[illegible]	0.40741	0.41081	0.41421	0.41763	0.42105
24	[illegible]	0.42791	0.43136	0.43481	0.43828	0.44175
25°	0.44523	0.44872	0.45222	0.45573	0.45924	0.46277
26	0.46631	0.46985	0.47341	0.47698	0.48055	0.48414
27	0.48773	0.49134	0.49495	0.49858	0.50222	0.50587
28	0.50953	0.51320	0.51688	0.52057	0.52427	0.52798
29	0.53171	0.53545	0.53920	0.54296	0.54673	0.55051
30°	0.55431	0.55812	0.56194	0.56577	0.56962	0.57348
31	0.57735	0.58124	0.58513	0.58905	0.59297	0.59691
32	0.60086	0.60483	0.60881	0.61280	0.61681	0.62083
33	0.62487	0.62892	0.63299	0.63707	0.64117	0.64528
34	0.64941	0.65355	0.65771	0.66189	0.66608	0.67028
35°	0.67451	0.67875	0.68301	0.68728	0.69157	0.69588
36	0.70021	0.70455	0.70891	0.71329	0.71769	0.72211
37	0.72654	0.73100	0.73547	0.73996	0.74447	0.74900
38	0.75355	0.75812	0.76272	0.76733	0.77196	0.77661
39	0.78129	0.78598	0.79070	0.79544	0.80020	0.80498
40°	0.80978	0.81461	0.81946	0.82434	0.82923	0.83415
41	0.83910	0.84407	0.84906	0.85408	0.85912	0.86419
42	0.86929	0.87441	0.87955	0.88473	0.88992	0.89515
43	0.90040	0.90569	0.91099	0.91633	0.92170	0.92709
44	0.93252	0.93797	0.94345	0.94896	0.95451	0.96008
	0.96569	0.97133	0.97700	0.98270	0.98843	0.99420
	60′	50′	40′	30′	20′	10′

COTANGENT

′	10′	20′	30′	40′	50′	60′	
	TANGENT						
0	1.00583	1.01170	1.01761	1.02355	1.02952	1.03553	44
3	1.04158	1.04766	1.05378	1.05994	1.06613	1.07237	43
7	1.07864	1.08496	1.09131	1.09770	1.10414	1.11061	42
11	1.11713	1.12369	1.13029	1.13694	1.14363	1.15037	41
17	1.15715	1.16398	1.17085	1.17777	1.18474	1.19175	40°
75	1.19882	1.20593	1.21310	1.22031	1.22758	1.23490	39
90	1.24227	1.24969	1.25717	1.26471	1.27230	1.27994	38
94	1.28764	1.29541	1.30323	1.31110	1.31904	1.32704	37
04	1.33511	1.34323	1.35142	1.35968	1.36800	1.37638	36
38	1.38484	1.39336	1.40195	1.41061	1.41934	1.42815	35°
15	1.43703	1.44598	1.45501	1.46411	1.47330	1.48256	34
56	1.49190	1.50133	1.51084	1.52043	1.53010	1.53987	33
87	1.54972	1.55966	1.56969	1.57981	1.59002	1.60033	32
33	1.61074	1.62125	1.63185	1.64256	1.65337	1.66428	31
28	1.67530	1.68643	1.69766	1.70901	1.72047	1.73205	30°
05	1.74375	1.75556	1.76749	1.77955	1.79174	1.80405	29
05	1.81649	1.82906	1.84177	1.85462	1.86760	1.88073	28
73	1.89400	1.90741	1.92098	1.93470	1.94858	1.96261	27
61	1.97680	1.99116	2.00569	2.02039	2.03526	2.05030	26
30	2.06553	2.08094	2.09654	2.11233	2.12832	2.14451	25°
51	2.16090	2.17749	2.19430	2.21132	2.22857	2.24604	24
04	2.26374	2.28167	2.29984	2.31826	2.33693	2.35585	23
85	2.37504	2.39449	2.41421	2.43422	2.45451	2.47509	22
09	2.49597	2.51715	2.53865	2.56046	2.58261	2.60509	21
09	2.62791	2.65109	2.67462	2.69853	2.72281	2.74748	20°
48	2.77254	2.79802	2.82391	2.85023	2.87700	2.90421	19
21	2.93189	2.96004	2.98869	3.01783	3.04749	3.07768	18
68	3.10842	3.13972	3.17159	3.20406	3.23714	3.27085	17
85	3.30521	3.34023	3.37594	3.41236	3.44951	3.48741	16
41	3.52609	3.56557	3.60588	3.64705	3.68909	3.73205	15°
05	3.77595	3.82083	3.86671	3.91364	3.96165	4.01078	14
78	4.06107	4.11256	4.16530	4.21933	4.27471	4.33148	13
48	4.38969	4.44942	4.51071	4.57363	4.63825	4.70463	12
63	4.77286	4.84300	4.91516	4.98940	5.06584	5.14455	11
55	5.22566	5.30928	5.39552	5.48451	5.57638	5.67128	10°
28	5.76937	5.87080	5.97576	6.08444	6.19703	6.31375	9
75	6.43484	6.56055	6.69116	6.82694	6.96823	7.11537	8
37	7.26873	7.42871	7.59575	7.77035	7.95302	8.14435	7
35	8.34496	8.55555	8.77689	9.00983	9.25530	9.51436	6
36	9.78817	10.0780	10.3854	10.7119	11.0594	11.4301	5°
01	11.8262	12.2505	12.7062	13.1969	13.7267	14.3007	4
07	14.9244	15.6048	16.3499	17.1693	18.0750	19.0811	3
11	20.2056	21.4704	22.9038	24.5418	26.4316	28.6363	2
63	31.2416	34.3678	38.1885	42.9641	49.1039	57.2900	1
00	68.7501	85.9398	114.589	171.885	343.774	∞	0°
60′	50′	40′	30′	20′	10′	0′	Angle
	COTANGENT						

9. Naperian Logarithms of Numbers from 1 to 119

#	0.	1.	2.	3.	4.	5.	6.	7.	8.	9.
0	$-\infty$	0.0000	0.6931	1.0986	1.3863	1.6094	1.7918	1.9459	2.0794	2.1
1	2.3026	2.3979	2.4849	2.5649	2.6391	2.7081	2.7726	2.8332	2.8904	2.9
2	2.9957	3.0445	3.0910	3.1355	3.1781	3.2189	3.2581	3.2958	3.3322	3.
3	3.4012	3.4340	3.4657	3.4965	3.5264	3.5553	3.5835	3.6109	3.6376	3.
4	3.6889	3.7136	3.7377	3.7612	3.7842	3.8067	3.8286	3.8501	3.8712	3.
5	3.9120	3.9318	3.9512	3.9703	3.9890	4.0073	4.0254	4.0430	4.0604	4.
6	4.0943	4.1109	4.1271	4.1431	4.1589	4.1744	4.1897	4.2047	4.2195	4.
7	4.2485	4.2627	4.2767	4.2905	4.3041	4.3175	4.3307	4.3438	4.3567	4.
8	4.3820	4.3944	4.4067	4.4188	4.4308	4.4427	4.4543	4.4659	4.4773	4.
9	4.4998	4.5109	4.5218	4.5326	4.5433	4.5539	4.5643	4.5747	4.5850	4.
10	4.6052	4.6151	4.6250	4.6347	4.6444	4.6540	4.6634	4.6728	4.6821	4.
11	4.7005	4.7095	4.7185	4.7274	4.7362	4.7449	4.7536	4.7622	4.7707	4.

10. Multipliers for Transferring Logarithms

	Common to Naperian			Naperian to Common	
1	2.302585093	Example.	1	0.434294482	Example.
2	4.605170186	Find Nap log of 105	2	0.868588964	Find number corresponding to Nap log 1.6
3	6.907755279		3	1.302883446	.6 0.2
		Com log 105 = 2.02119			07
4	9.210340372	2 4.605170	4	1.737177928	8
5	11.512925465	.02 46052	5	2.171472410	+0
6	13.815510558	1 2303	6	2.605766891	−0.4
		1 230			1
7	16.118095651	9 207	7	3.040061373	
8	18.420680744	Nap log 105 = 4.65396	8	3.474355855	Com log =
9	20.723265837		9	3.908650337	Number = 0.6

11. Multipliers for Finding Lengths of Circular Arcs

	Degrees	Minutes	Seconds	
1	0.017453293	0.000290888	0.000004848	Example.
2	0.034906585	0.000581776	0.000009696	Find length of arc for a central angle of 48° 45′ in circle of 12 ft. radius.
3	0.052359878	0.000872665	0.000014544	
4	0.069813170	0.001163553	0.000019393	40° 0.69813 2
5	0.087266463	0.001454441	0.000024241	8° .139626
6	0.104719755	0.001745329	0.000029089	40′ .011636
7	0.122173048	0.002036217	0.000033937	5′ .001454
8	0.139626340	0.002327106	0.000038785	0.85085
9	0.157079633	0.002617994	0.000043633	12
				Length = 10.210 ft.

12. Circumferences of Circles

Diameters in Units and Tenths

.0	.1	.2	.3	.4	.5	.6	.7	.8	.9
0.000	0.314	0.628	0.942	1.257	1.571	1.885	2.199	2.513	2.827
3.142	3.456	3.770	4.084	4.398	4.712	5.027	5.341	5.655	5.969
6.283	6.597	6.912	7.226	7.540	7.854	8.168	8.482	8.796	9.111
9.425	9.739	10.05	10.37	10.68	11.00	11.31	11.62	11.94	12.25
12.57	12.88	13.19	13.51	13.82	14.14	14.45	14.77	15.08	15.39
15.71	16.02	16.34	16.65	16.96	17.28	17.59	17.91	18.22	18.54
18.85	19.16	19.48	19.79	20.11	20.42	20.73	21.05	21.36	21.68
21.99	22.31	22.62	22.93	23.25	23.56	23.88	24.19	24.50	24.82
25.13	25.45	25.76	26.08	26.39	26.70	27.02	27.33	27.65	27.96
28.27	28.59	28.90	29.22	29.53	29.85	30.16	30.47	30.79	31.10
31.42	31.73	32.04	32.36	32.67	32.99	33.30	33.62	33.93	34.24
34.56	34.87	35.19	35.50	35.81	36.13	36.44	36.76	37.07	37.38
37.70	38.01	38.33	38.64	38.96	39.27	39.58	39.90	40.21	40.53
40.84	41.15	41.47	41.78	42.10	42.41	42.73	43.04	43.35	43.67
43.98	44.30	44.61	44.92	45.24	45.55	45.87	46.18	46.50	46.81
47.12	47.44	47.75	48.07	48.38	48.69	49.01	49.32	49.64	49.95
50.27	50.58	50.89	51.21	51.52	51.84	52.15	52.46	52.78	53.09
53.41	53.72	54.04	54.35	54.66	54.98	55.29	55.61	55.92	56.23
56.55	56.86	57.18	57.49	57.81	58.12	58.43	58.75	59.06	59.38
59.69	60.00	60.32	60.63	60.95	61.26	61.58	61.89	62.20	62.52

13. Circumferences of Circles

Diameters in Units and Eighths

0	1/8	1/4	3/8	1/2	5/8	3/4	7/8
0.0000	0.3927	0.7854	1.1781	1.5708	1.9635	2.3562	2.7489
3.1416	3.5343	3.9270	4.3197	4.7124	5.1051	5.4978	5.8905
6.2832	6.6759	7.0686	7.4613	7.8540	8.2467	8.6394	9.0321
9.4248	9.8175	10.210	10.603	10.996	11.388	11.781	12.174
12.566	12.959	13.352	13.744	14.137	14.530	14.923	15.315
15.708	16.101	16.493	16.886	17.279	17.671	18.064	18.457
18.850	19.242	19.635	20.028	20.420	20.813	21.206	21.598
21.991	22.384	22.777	23.169	23.562	23.955	24.347	24.740
25.133	25.525	25.918	26.311	26.704	27.096	27.489	27.882
28.274	28.667	29.060	29.452	29.845	30.238	30.631	31.023
31.416	31.809	32.201	32.594	32.987	33.379	33.772	34.165
34.558	34.950	35.343	35.736	36.128	36.521	36.914	37.306
37.699	38.092	38.485	38.877	39.270	39.663	40.055	40.448
40.841	41.233	41.626	42.019	42.412	42.804	43.197	43.590
43.982	44.375	44.768	45.160	45.553	45.946	46.338	46.731
47.124	47.517	47.909	48.302	48.695	49.087	49.480	49.873
50.265	50.658	51.051	51.444	51.836	52.229	52.622	53.014
53.407	53.800	54.192	54.585	54.978	55.371	55.763	56.156
56.549	56.941	57.334	57.727	58.119	58.512	58.905	59.298
59.690	60.083	60.476	60.868	61.261	61.654	62.046	62.439

14. Decimal Equivalents for Common Fractions

The given decimals are the parts of inches corresponding to fraction of inches in first column; also, the parts of feet for the fraction of inches in third column.

	0.0052	1/16		0.2552	3 1/16		0.5052	6 1/16		0.7552	9
	0.0104	1/8		0.2604	3 1/8		0.5104	6 1/8		0.7604	9
1/64	0.015625	3/16	17/64	0.265625	3 3/16	33/64	0.515625	6 3/16	49/64	0.765625	9
	0.0208	1/4		0.2708	3 1/4		0.5208	6 1/4		0.7708	9
	0.0260	5/16		0.2760	3 5/16		0.5260	6 5/16		0.7760	9
1/32	0.03125	3/8	9/32	0.28125	3 3/8	17/32	0.53125	6 3/8	25/32	0.78125	9
	0.0364	7/16		0.2865	3 7/16		0.5364	6 7/16		0.7865	9
	0.0417	1/2		0.2917	3 1/2		0.5417	6 1/2		0.7917	9
3/64	0.046875	9/16	19/64	0.296875	3 9/16	35/64	0.546875	6 9/16	51/64	0.796875	9
	0.0521	5/8		0.3021	3 5/8		0.5521	6 5/8		0.8021	9
	0.0573	11/16		0.3073	3 11/16		0.5573	6 11/16		0.8073	9
1/16	0.0625	3/4	5/16	0.3125	3 3/4	9/16	0.5625	6 3/4	13/16	0.8125	9
	0.0677	13/16		0.3177	3 13/16		0.5677	6 13/16		0.8177	9
	0.0729	7/8		0.3229	3 7/8		0.5729	6 7/8		0.8229	9
5/64	0.078125	15/16	21/64	0.328125	3 15/16	37/64	0.578125	6 15/16	53/64	0.828125	9
	0.0833	1		0.3333	4		0.5833	7		0.8333	10
	0.0885	1 1/16		0.3385	4 1/16		0.5885	7 1/16		0.8385	10
3/32	0.09375	1 1/8	11/32	0.34375	4 1/8	19/32	0.59375	7 1/8	27/32	0.84375	10
	0.0990	1 3/16		0.3490	4 3/16		0.5990	7 3/16		0.8490	10
	0.1042	1 1/4		0.3542	4 1/4		0.6042	7 1/4		0.8542	10
7/64	0.109375	1 5/16	23/64	0.359375	4 5/16	39/64	0.609375	7 5/16	55/64	0.859375	10
	0.1146	1 3/8		0.3646	4 3/8		0.6146	7 3/8		0.8646	10
	0.1198	1 7/16		0.3698	4 7/16		0.6198	7 7/16		0.8698	10
1/8	0.1250	1 1/2	3/8	0.3750	4 1/2	5/8	0.6250	7 1/2	7/8	0.8750	10
	0.1302	1 9/16		0.3802	4 9/16		0.6302	7 9/16		0.8802	10
	0.1354	1 5/8		0.3854	4 5/8		0.6354	7 5/8		0.8854	10
9/64	0.140625	1 11/16	25/64	0.390625	4 11/16	41/64	0.640625	7 11/16	57/64	0.890625	10
	0.1458	1 3/4		0.3958	4 3/4		0.6458	7 3/4		0.8958	10
	0.1510	1 13/16		0.4010	4 13/16		0.6510	7 13/16		0.9010	10
5/32	0.15625	1 7/8	13/32	0.40625	4 7/8	21/32	0.65625	7 7/8	29/32	0.90625	10
	0.1615	1 15/16		0.4114	4 15/16		0.6615	7 15/16		0.9115	10
	0.1667	2		0.4167	5		0.6667	8		0.9167	11
11/64	0.171875	2 1/16	27/64	0.421875	5 1/16	43/64	0.671875	8 1/16	59/64	0.921875	11
	0.1771	2 1/8		0.4271	5 1/8		0.6771	8 1/8		0.9271	11
	0.1823	2 3/16		0.4323	5 3/16		0.6823	8 3/16		0.9323	11
3/16	0.1875	2 1/4	7/16	0.4375	5 1/4	11/16	0.6875	8 1/4	15/16	0.9375	11
	0.1927	2 5/16		0.4427	5 5/16		0.6927	8 5/16		0.9427	11
	0.1979	2 3/8		0.4479	5 3/8		0.6979	8 3/8		0.9479	11
13/64	0.203125	2 7/16	29/64	0.453125	5 7/16	45/64	0.703125	8 7/16	61/64	0.953125	11
	0.2083	2 1/2		0.4583	5 1/2		0.7083	8 1/2		0.9583	11
	0.2135	2 9/16		0.4635	5 9/16		0.7135	8 9/16		0.9635	11
7/32	0.21875	2 5/8	15/32	0.46875	5 5/8	23/32	0.71875	8 5/8	31/32	0.96875	11
	0.2240	2 11/16		0.4740	5 11/16		0.7240	8 11/16		0.9740	11
	0.2292	2 3/4		0.4792	5 3/4		0.7292	8 3/4		0.9792	11
15/64	0.234375	2 13/16	31/64	0.484375	5 13/16	47/64	0.734375	8 13/16	63/64	0.984375	11
	0.2395	2 7/8		0.4896	5 7/8		0.7396	8 7/8		0.9896	11
	0.2448	2 15/16		0.4948	5 15/16		0.7448	8 15/16		0.9948	11
1/4	0.2500	3	1/2	0.5000	6	3/4	0.7500	9	1	1.0000	12

15. Product of Fractions Expressed in Decimals

	1/16	1/8	3/16	1/4	5/16	3/8	7/16	1/2	9/16	5/8	11/16	3/4	13/16	7/8	15/16
1/16	.0039														
1/8	.0078	.0156													
3/16	.0117	.0234	.0351												
1/4	.0156	.0313	.0469	.0625											
5/16	.0195	.0391	.0587	.0781	.0977										
3/8	.0234	.0469	.0705	.0937	.1172	.1406									
7/16	.0273	.0547	.0823	.1093	.1367	.1641	.1914								
1/2	.0313	.0625	.0938	.1250	.1562	.1875	.2188	.2500							
9/16	.0352	.0703	.1056	.1406	.1757	.2110	.2462	.2813	.3164						
5/8	.0391	.0781	.1172	.1562	.1952	.2343	.2734	.3125	.3516	.3906					
11/16	.0430	.0859	.1289	.1718	.2148	.2578	.3007	.3438	.3867	.4297	.4727				
3/4	.0469	.0938	.1406	.1875	.2344	.2813	.3281	.3750	.4219	.4686	.5156	.5625			
13/16	.0508	.1016	.1523	.2031	.2539	.3047	.3555	.4063	.4570	.5078	.5586	.6094	.6601		
7/8	.0547	.1094	.1640	.2187	.2734	.3281	.3828	.4375	.4922	.5469	.6016	.6563	.7109	.7656	
15/16	.0586	.1172	.1757	.2343	.2929	.3515	.4102	.4688	.5274	.5859	.6445	.7031	.7617	.8203	.8789
1000	.0625	.1250	.1875	.2500	.3125	.3750	.4375	.5000	.5625	.6250	.6875	.7500	.8125	.8750	.9375

16. Weights and Measures

(See also Circular No 47 of U S Bureau of Standards, on Units of Weights and Measures, issued July 1, 1914)

Avoirdupois Weight

ains	Drams	Ounces	Pounds	Hund'd-wt	Net tons	Gross tons
1	0.03657	0.002286	0.000143	0.00000127	0.00000007135	0.00000006371
34375	1	0.0625	0.003906	0.00003488	0.000001953	0.000001744
5	16	1	0.0625	0.00055804	0.0003125	0.00002790
000	256	16	1	0.0089286	0.0005	0.0004464
1 000	28 672	1 792	112	1	0.056	0 05
00 000	512 000	32 000	2 000	17.857	1	0.89286
80 000	573 440	35 840	2 240	20	1.12	1

lb avoir = 1.215278 lb troy. 1 net ton = 2 000 lb = 0.892857 gross ton.

Troy Weight

Grains	Pennyweight	Ounces	Pounds
1	0.041667	0.0020833	0.0001736
24	1	0.05	0.0041667
480	20	1	0.0833333
5 760	240	12	1

1 lb troy = 0.822857 lb avoir. 175 oz troy = 192 oz avoir.

Apothecaries' Weight

Grains	Scruples	Drams	Ounces	Pounds
1	0.05	0.016667	0.0020833	0.00017361
20	1	0.333333	0.0416667	0.0034722
60	3	1	0.125	0 0104167
480	24	8	1	0.0833333
5 760	288	96	12	1

The lb, oz, and gr are same as in troy wt. Avoir gr = troy gr = apothecaries' gr.

Density

Lb per cu in	Lb per cu ft	Net tons per cu yd	Lb per U S gallon	Gm per cu
1	1 728	23.328	230.999	27.6826
0.0005787	1	0.0135	0.13368	0.01601
0.042867	74.074	1	9.9023	1.18653
0.004329	7.4805	0.10099	1	0.11983
0.036127	62.4281	0.84278	8.3454	1

Linear Measure

Inches	Feet	Yards	Rods	Furlongs	Statute miles	Meters
1	0.08333	0.02778	0.0050505	0.00012626	0.00001578	0.0254
12	1	0.33333	0.0606061	0.00151515	0.00018939	0.3048
36	3	1	0.1818182	0.00454545	0.00056818	0.9144
198	16.5	5.5	1	0.025	0.003125	5.029
7 920	660	220	40	1	0.125	201.16
63 360	5 280	1 760	320	8	1	1609.3
39.37	3.2808	1.0936	0.1988	0.0000777	0.0006214	1

Rope and Cable Measure

Inches	Spans	Feet	Fathoms	Cable lengths	Statute miles	Meters
1	0.1111	0.08333	0.013889	0.0001157	0.00001578	0.0254
9	1	0.75	0.16667	0.001042	0.000142	0.2286
12	1.3333	1	0.125	0.001389	0.0001894	0.3048
72	8	6	1	0.008333	0.001364	1.8288
8 640	960	720	120	1	0.136368	219.456
63 360	7 040	5 280	880	7.3333	1	1 609.3
39.37	4.377	3.2808	0.547	0.0455	0.0006214	1

Nautical Measure

1 nautical mile, as adopted by the U S Coast and Geodetic Survey, equals the length of one minute of arc of a great circle of a sphere the surface of which equals that of the earth = 6 080.204 ft = 1.1516 statute miles. 1 league = 3 nautical miles = 18 239.612 ft.

Gunter's Chain

Inches	Links	Feet	Rods	Chains	Statute miles	Meters
1	0.1263	0.08333	0.0050505	0.001263	0.00001578	0.0254
7.92	1	0.66	0.04	0.01	0.000125	0.2012
12	1.7156	1	0.060606	0.01515	0.0001894	0.3048
198	25	16.5	1	0.25	0.003125	5.029
792	100	66	4	1	0.0125	20.12
63 360	8 000	5 280	320	80	1	1609.3
39.37	4.971	3.2808	0.1988	0.04971	0.0006214	1

Square or Land Measure

Sq inches	Sq feet	Sq yards	Sq rods	Acres	Sq miles	Sq meters
1	0.006944	0.0007716	.....	.....	.....	0.0006452
144	1	0.111111	.....	.....	.....	0.0929
1 296	9.0	1	0.03306	0.0002066	.....	0.8361
39 204	272.25	30.25	1	0.00625	0.00000977	25.30
6 272 640	43 560	4 840	160	1	0.0015625	4 047
.....	27 878 400	3 097 600	102 400	640	1	2 590 000
1 550	10.764	1.196	0.03954	0.0002471	.....	1

1 sq rood = 40 sq rods. 1 acre = 4 sq roods. 1 sq acre = 208.71 ft sq.

Cubic or Solid Measure

Cu in	Cu ft	Cu yd	Cu meters
1	0.0005787	0.000021433	0.000016387
1 728	1	0.037037	0.02832
46 656	27	1	0.7646
61 020	35.31	1.3079	1

ard of wood = 128 cu ft = 4 by 4 by 8 ft. 1 perch of masonry = 24.75 cu ft = y 1.5 by 1 ft; usually taken as 25 cu ft.

Dry Measure

ls	Quarts	Gallons	Pecks	Bushels	Cu in	Cu ft	Cu meters
	0.50	0.125	0.0625	0.015625	33.6003	0.01944	0.0005506
	1	0.25	0.125	0.03125	67.2006	0.03889	0.001101
	4	1	0.50	0.125	268.8025	0.1556	0.004405
	8	2	1	0.25	537.605	0.3111	0.00881
	32	8	4	1	2 150.42	1.2444	0.03524
782	0.014881	0.0037202	0.0018601	0.00046503	1	0.0005787	0.000016387
31	25.7141	6.42851	3.21426	0.80356	1 728	1	0.02832
773	908.086	227.022	113.511	28.3777	61 020	35.31	1

ped bushel = 1.25 struck bushel, and the cone must be not less than 6 in high.

Liquid Measure

	Pints	Quarts	Gallons	Barrels	Cu in	Cu ft	Cu meters
	0.25	0.125	0.03125	0.000992	7.21875	0.004177	0.0001183
	1	0.5	0.125	0.003968	28.875	0.01671	0.0004732
	2	1	0.25	0.007937	57.75	0.03342	0.0009464
	8	4	1	0.031746	231	0.1337	0.003785
	252	126	31.5	1	7 276.5	4.2109	0.1192
	0.034632	0.017316	0.004329	0.00013743	1	0.0005787	0.000016387
	59.8443	29.9221	7.48053	0.23748	1 728	1	0.02832
2	2 113.385	1 056.693	264.1732	8.38645	61 020	35.31	1

al. British imperial gal = 277.410 cu in, or 10 lb avoir of pure water at 62° F n at 30 in. British imperial gal = 1.20091 U S gal. 1 fluid dram = 60 minims luid oz = 0.0078125 pint. 1 fluid oz = 480 minims = 8 drams = 0.0625 pint.

Metric System

Measures of Length, Capacity, and Weight

Kilo-meter	Hecto-meter	Deca-meter	Meter	Deci-meter	Centi-meter	Milli-meter
Kiloliter or Stere	Hecto-liter or decistere	Decaliter or centi-stere	Liter or millistere	Deciliter	Centiliter	Milli-liter
Kilo-gramme	Hecto-gramme	Deca-gramme	Gramme	Deci-gramme	Centi-gramme	Milli-gramme
1	10	100	1 000	10 000	100 000	1 000 000
	1	10	100	1 000	10 000	100 000
		1	10	100	1 000	10 000
			1	10	100	1 000
			0.1	1	10	100
			0.01	0.1	1	10
			0.001	0.01	0.1	1

eter = 10 kilometers = 10 000 meters. 1 tonne = 1 000 kilogrammes = 100 10 myriagrammes. 1 gramme = 1 cu centimeter of distilled water at its max ea level in latitude of Paris and barom at 760 millimeters. 1 liter =

Metric System (*Continued*)

Square or Surface Measure

Square kilometer	Square hectometer or hectare	Square decameter or are	Square meter or centiare	Square decimeter	Square centimeter	Square millimeter
	100	10 000	1 000 000			
	1	100	10 000	1 000 000		
	0.01	1	100	10 000	1 000 000	
	0.0001	0.01	1	100	10 000	1 000 000
	0.000001	0.0001	0.01	1	100	10 000
		0.000001	0.0001	0.01	1	100
			0.000001	0.0001	0.01	1

1 sq myriameter = 100 sq kilometers = 100 000 000 sq meters.

Cubic Measure

Cubic decameter	Cubic meter or stere	Cubic decimeter or liter	Cubic centimeter	Cubic millimeter
1	1 000	1 000 000	1 000 000 000	
0.001	1	1 000	1 000 000	1 000 000 000
0.000001	0.001	1	1 000	1 000 000
0.000000001	0.000001	0.001	1	1 000
	0.000000001	0.000001	0.001	1

1 cu meter = 1 kiloliter = 1 stere.

17. Fractions of an Inch to Millimeters

16ths	32nds	64ths	Millimeter	16ths	32nds	64ths	Millimeter	16ths	32nds	64ths	Millimeter
		1	0.397			33	9.128			45	17.859
	1	2	0.794	6	12	24	9.525		23	46	18.256
		3	1.191			25	9.922			47	18.653
1	2	4	1.588		13	26	10.319	12	24	48	19.050
		5	1.984			27	10.716			49	19.447
	3	6	2.381	7	14	28	11.113		25	50	19.844
		7	2.778			29	11.509			51	20.241
2	4	8	3.175		15	30	11.906	13	26	52	20.638
		9	3.572			31	12.303			53	21.034
	5	10	3.969	8	16	32	12.700		27	54	21.431
		11	4.366			33	13.097			55	21.828
3	6	12	4.763		17	34	13.494	14	28	56	22.225
		13	5.159			35	13.891			57	22.622
	7	14	5.556	9	18	36	14.288		29	58	23.019
		15	5.953			37	14.684			59	23.416
4	8	16	6.350		19	38	15.081	15	30	60	23.812
		17	6.747			39	15.478			61	24.209
	9	18	7.144	10	20	40	15.875		31	62	24.606
		19	7.541			41	16.272			63	25.003
5	10	20	7.938		21	42	16.669	16	32	64	25.400
		21	8.334			43	17.066				
	11	22	8.731	11	22	44	17.463				

18. Conversion Table of Measures

	1	2	3	4	5	6	7	8	9
ı to centimeters (cm)...	2.5400	5.0800	7.6200	10.160	12.700	15.240	17.780	20.320	22.860
t to meters.....	0.3048	0.6096	0.9144	1.2192	1.5240	1.8288	2.1336	2.4384	2.7432
ards to meters.....	0.9144	1.8288	2.7432	3.6576	4.5720	5.4864	6.4008	7.3152	8.2296
athoms to meters.....	1.8288	3.6576	5.4864	7.3152	9.1440	10.973	12.802	14.630	16.459
tatute miles to kilom....	1.6093	3.2187	4.8280	6.4374	8.0467	9.6561	11.265	12.875	14.484
lautical miles to kilom..	1.8533	3.7066	5.5600	7.4133	9.2666	11.120	12.973	14.826	16.679
m to in.....	0.3937	0.7874	1.1811	1.5748	1.9685	2.3622	2.7559	3.1496	3.5433
leters (m) to ft.....	3.2808	6.5617	9.8425	13.123	16.404	19.685	22.966	26.247	29.527
leters to yards.....	1.0936	2.1872	3.2808	4.3744	5.4681	6.5617	7.6553	8.7489	9.8425
leters to fathoms.....	0.5468	1.0936	1.6404	2.1872	2.7340	3.2808	3.8276	4.3744	4.9212
:m to statute miles.....	0.6214	1.2427	1.8641	2.4855	3.1069	3.7282	4.3496	4.9710	5.5923
:m to nautical miles.....	0.5396	1.0791	1.6187	2.1583	2.6979	3.2374	3.7770	4.3166	4.8561
ıq in to sq cm.....	6.4516	12.903	19.355	25.807	32.258	38.710	45.161	51.613	58.065
ıq ft to sq m.....	0.0929	0.1858	0.2787	0.3716	0.4645	0.5574	0.6503	0.7432	0.8361
ıq yards to sq m.....	0.8361	1.6723	2.5084	3.3445	4.1807	5.0168	5.8529	6.6890	7.5252
ıcres to hectares.....	0.4047	0.8094	1.2141	1.6187	2.0234	2.4281	2.8328	3.2375	3.6422
ıq stat miles to sq km...	2.5900	5.1800	7.7700	10.360	12.950	15.540	18.130	20.720	23.310
ıq cm to sq in.....	0.1550	0.3100	0.4650	0.6200	0.7750	0.9300	1.0850	1.2400	1.3950
ıq m to sq ft.....	10.764	21.528	32.292	43.055	53.819	64.583	75.347	86.111	96.875
ıq m to sq yd.....	1.1960	2.3920	3.5880	4.7839	5.9799	7.1759	8.3719	9.5679	10.764
Hectares to acres.....	2.4710	4.9421	7.4131	9.8842	12.355	14.826	17.297	19.768	22.239
ıq km to sq stat miles...	0.3861	0.7722	1.1583	1.5444	1.9305	2.3166	2.7027	3.0888	3.4749
?u in to cu cm.....	16.387	32.774	49.162	65.549	81.936	98.323	114.71	131.10	147.48
?u ft to cu m.....	0.0283	0.0566	0.0850	0.1133	0.1416	0.1699	0.1982	0.2265	0.2549
?u yd to cu m.....	0.7646	1.5291	2.2937	3.0582	3.8228	4.5874	5.3519	6.1165	6.8810
?u cm to cu in.....	0.0610	0.1220	0.1831	0.2441	0.3051	0.3661	0.4272	0.4882	0.5492
?u m to cu ft.....	35.314	70.629	105.94	141.26	176.57	211.89	247.20	282.51	317.83
?u m to cu yd.....	1.3079	2.6159	3.9238	5.2318	6.5397	7.8477	9.1556	10.464	11.772
Fluid drams to milliliters	3.6966	7.3932	11.090	14.786	18.483	22.180	25.876	29.573	33.269
Fluid oz to liters.....	0.0296	0.0591	0.0887	0.1183	0.1479	0.1774	0.2070	0.2366	0.2662
Liq quarts to liters.....	0.9463	1.8927	2.8390	3.7853	4.7317	5.6780	6.6243	7.5707	8.5170
Dry quarts to liters.....	1.1012	2.2024	3.3036	4.4048	5.5060	6.6072	7.7084	8.8096	9.9108
U S bush to hectoliters...	0.3524	0.7048	1.0572	1.4095	1.7619	2.1143	2.4667	2.8191	3.1715
U S liq barrels to kiloliters	0.1192	0.2385	0.3577	0.4770	0.5962	0.7154	0.8347	0.9539	1.0732
Cu ft to hectoliters.....	0.2832	0.5663	0.8495	1.1327	1.4158	1.6990	1 9821	2.2653	2.5484
Milliliters to fluid drams.	0.2705	0.5410	0.8116	1.0821	1.3526	1.6231	1.8936	2.1641	2.4347
Liters to fluid oz.....	33.815	67.629	101.44	135.26	169.07	202.89	236.70	270.52	304.33
Liters to liquid quarts...	1.0567	2.1134	3.1701	4.2268	5.2836	6.3403	7.3970	8.4537	9.5104
Liters to dry quarts.....	0.9081	1.8162	2.7243	3.6324	4.5405	5.4486	6.3567	7.2648	8.1729
Hectoliters to U S bush..	2.8378	5.6756	8.5135	11.351	14.189	17.027	19.865	22.703	25.540
Kiloliters to U S liq bar'ls	8.3864	16.773	25.159	33.546	41.932	50.319	58.705	67.092	75.478
Hectoliters to cu ft.....	3.5315	7.0631	10.595	14.126	17.658	21.189	24.721	28.252	31.784
Grains to grams.....	0.0648	0.1296	0.1944	0.2592	0.3240	0.3888	0.4536	0.5184	0.5832
Oz [avoir] to gm.....	28.350	56.699	85.049	113.40	141.75	170.10	198.45	226.80	255.15
Oz [troy] to gm.....	31.103	62.207	93.310	124.41	155.52	186.62	217.72	248.83	279.93
Lb [avoir] to kg.....	0.4536	0.9072	1.3608	1.8144	2.2680	2.7215	3.1751	3.6287	4.0823
Net tons to tonnes [met'c]	0.9072	1.8144	2.7215	3.6287	4.5359	5.4431	6.3503	7.2575	8.1647
Gross tons to tonnes [met]	1.0160	2.0321	3.0481	4.0642	5.0802	6.0963	7.1123	8.1284	9.1444
Gm to grains.....	15.432	30.865	46.297	61.729	77.162	92.594	108.03	123.46	138.89
Gm to oz [avoir].....	0.0353	0.0705	0.1058	0.1411	0.1764	0.2116	0.2469	0.2822	0.3175
Gm to oz [troy].....	0.0322	0.0643	0.0965	0.1286	0.1608	0.1929	0.2251	0.2572	0.2894
Kg to lb [avoir].....	2.2046	4.4092	6.6139	8.8185	11.023	13.228	15.432	17.637	19.842
Tonnes [metric] to net ton	1.1023	2.2046	3.3069	4.4092	5.5116	6.6139	7.7162	8.8185	9.9208
Tonnes [met'c] to gross t'n	0.9842	1.9684	2.9526	3.9368	4.9210	5.9052	6.8894	7.8737	8.8579

Conversion Table of Measures (*Continued*)

		1	2	3	4	5	6	7	8
Speed	Ft per sec to cm per sec	30.480	60.960	91.440	121.92	152.40	182.88	213.36	243.84
	Ft per sec to m per min	18.288	36.576	54.864	73.152	91.440	109.73	128.02	146.30
	Ft per min to cm per sec	0.5080	1.0160	1.5240	2.0320	2.5400	3.0480	3.5560	4.0640
	Knots to km per hr	1.8533	3.7066	5.5600	7.4133	9.2666	11.120	12.973	14.826
	Cm per sec to ft per sec	0.0328	0.0656	0.0984	0.1312	0.1640	0.1968	0.2297	0.2625
	M per min to ft per sec	0.0547	0.1094	0.1640	0.2187	0.2734	0.3281	0.3828	0.4374
	Cm per sec to ft per min	1.9685	3.9370	5.9055	7.8740	9.8425	11.811	13.780	15.748
	Kilom per hr to knots	0.5396	1.0792	1.6188	2.1584	2.6980	3.2376	3.7772	4.3168
Force per Length, Area, and Volume	Lb per lin ft to kg per m	1.4881	2.9763	4.4644	5.9526	7.4407	8.9289	10.417	11.905
	Lb/sq in to kg/sq cm	0.0703	0.1406	0.2109	0.2812	0.3515	0.4218	0.4921	0.5625
	Lb per sq ft to kg per sq m	4.8825	9.7650	14.647	19.530	24.412	29.294	34.177	39.059
	Net tons per sq in to tonnes per sq cm	0.1406	0.2812	0.4218	0.5625	0.7031	0.8437	0.9843	1.1249
	Net tons per sq ft to tonnes per sq m	9.7651	19.530	29.295	39.060	48.826	58.591	68.356	78.121
	Lb/cu in to gm/cu cm	27.680	55.359	83.039	110.72	138.40	166.08	193.76	221.44
	Lb per cu ft to kg per cu m	16.018	32.037	48.055	64.073	80.092	96.110	112.13	128.15
	Kg per m to lb per lin ft	0.6720	1.3439	2.0159	2.6879	3.3599	4.0318	4.7038	5.3758
	Kg per sq cm to lb per sq in	14.223	28.446	42.669	56.892	71.115	85.338	99.561	113.78
	Kg per sq m to lb per sq ft	0.2048	0.4096	0.6145	0.8193	1.0241	1.2289	1.4337	1.6385
	Tonnes per sq m to net tons per sq ft	0.1024	0.2048	0.3072	0.4096	0.5120	0.6144	0.7168	0.8192
	Gm/cu cm to lb/cu in	0.0361	0.0723	0.1084	0.1445	0.1806	0.2168	0.2529	0.2890
	Kg per cu m to lb per cu ft	0.0624	0.1249	0.1873	0.2497	0.3121	0.3746	0.4370	0.4995
	Tonnes per sq cm to net tons per sq in	7.1117	14.223	21.335	28.447	35.558	42.670	49.782	56.894
Work	Ft-lb to meter-kg	0.1382	0.2765	0.4148	0.5530	0.6913	0.8295	0.9678	1.1060
	Ft-lb to joules	1.3563	2.7126	4.0689	5.4251	6.7814	8.1377	9.4940	10.850
	Meter-kg to ft-lb	7.2330	14.466	21.699	28.932	36.165	43.398	50.631	57.864
	Joules to ft-lb	0.7373	1.4746	2.2119	2.9492	3.6866	4.4239	5.1612	5.8985
Power	Hp to kilowatts (kw)	0.7457	1.4914	2.2371	2.9828	3.7285	4.4742	5.2199	5.9656
	Hp to cheval-vapeur	1.0139	2.0277	3.0416	4.0555	5.0694	6.0832	7.0971	8.1110
	Ft-lb per sec to watts	1.3563	2.7126	4.0689	5.4251	6.7814	8.1377	9.4940	10.850
	Lb per hp to kg per cheval-vapeur	0.4474	0.8947	1.3421	1.7895	2.2369	2.6842	3.1316	3.5790
	Kw to hp	1.3410	2.6820	4.0231	5.3641	6.7051	8.0461	9.3871	10.728
	Cheval-vapeur to hp	0.9863	1.9726	2.9590	3.9453	4.9316	5.9179	6.9042	7.8905
	Watts to ft-lb per sec	0.7373	1.4746	2.2119	2.9492	3.6866	4.4239	5.1612	5.8985
	Kg/cheval-vapeur to lb/hp	2.2353	4.4706	6.7058	8.9411	11.176	13.412	15.647	17.882
Heat	B t u to calories	0.252	0.504	0.756	1.008	1.260	1.512	1.764	2.016
	B t u to ft-lb [mean]	777.52	1555.0	2332.6	3110.1	3887.6	4665.1	5442.6	6220.2
	B t u/lb to calories/kg	0.5556	1.1111	1.6667	2.2223	2.7778	3.3334	3.8890	4.4445
	B t u per sq ft to calories per sq m	2.7126	5.4252	8.1378	10.850	13.563	16.276	18.988	21.701
	B t u per cu ft to calories per cu m	8.8993	17.799	26.698	35.597	44.496	53.396	62.295	71.194
	Calories to B t u	3.9683	7.9366	11.905	15.873	19.842	23.810	27.778	31.746
	Ft-lb to B t u [mean]	1.286*	2.572*	3.858*	5.145*	6.431*	7.717*	9.003*	1.029†
	Calories/kg to B t u/lb	1.8000	3.5999	5.3999	7.1998	8.9998	10.800	12.600	14.400
	Calories per sq m to B t u per sq ft	0.3687	0.7373	1.1060	1.4746	1.8433	2.2119	2.5806	2.9492
	Calories per cu m to B t u per cu ft	0.1124	0.2247	0.3371	0.4495	0.5618	0.6742	0.7866	0.8990

* (10^{-3}) † (10^{-2})

Conversion Table of Measures (*Concluded*)

	1	2	3	4	5	6	7	8	9
r ft to francs per m....	17.000	34.001	51.001	68.001	85.002	102.00	119.00	136.00	153.00
r ft to marks per m....	13.785	27.570	41.355	55.140	68.925	82.710	96.495	110.28	124.06
ı ft to francs/sq m....	55.773	111.55	167.32	223.09	278.87	334.64	390.41	446.19	501.96
ı ft to marks/sq m....	45.228	90.456	135.68	180.91	226.14	271.37	316.60	361.82	407.05
ı ft to francs/cu m....	182.98	365.95	548.93	731.91	914.89	1097.9	1280.8	1463.8	1646.8
ı ft to marks/cu m....	148.38	296.76	445.14	593.52	741.90	890.28	1038.7	1187.0	1335.4
et ton to francs/tonne.	5.7115	11.423	17.134	22.846	28.557	34.269	39.980	45.692	51.403
et ton to marks/tonne.	4.6316	9.2632	13.895	18.526	23.158	27.789	32.421	37.053	41.684
r hp to francs per che-al-vapeur.....	5.1103	10.221	15.331	20.441	25.552	30.662	35.772	40.882	45.993
r hp to marks per che-al-vapeur.....	4.1441	8.2882	12.432	16.576	20.720	24.864	29.009	33.153	37.297
t u to francs/calorie .	20.561	41.122	61.683	82.244	102.80	123.37	143.93	164.49	185.05
t u to marks/calorie.	16.673	33.347	50.020	66.693	83.367	100.04	116.71	133.39	150.06

e = 10 million ergs = 10 million dyne-centimeters
t = 1 nautical mile = 6 080 ft per hour
se power = 550 ft-lb per second
watt = 1 000 watts = 1 000 joules per sec
val-vapeur = 75 kilogram-meters per sec
u = heat necessary to raise 1 lb of water 1° F
orie = heat necessary to raise 1 kg of water 1° C
nc = 0.193 dollar
rk = 0.238 dollar

19. Chinese Measures

lowing is the new system of weights and measures adopted by China in pursuance imperial rescript of August 28, 1908.

MEASURES OF LENGTH

se			Metric
o	equals	0.0032	Centimeter
	"	0.032	"
n	"	0.32	"
'un	"	3.2	Centimeters
'ih	"	0.32	Meter
	"	1.6	Meters
ang	"	3.2	"
	"	576.0	"

MEASURES OF SURFACE

Chinese		Metric	
1 Hao	equals	0.006144	Are
1 Li	"	0.06144	"
1 Fen	"	0.6144	"
1 Fang pu	"	0.0256	"
1 Mu	"	6.144	Ares
1 Ch'ing	"	614.4	"

MEASURES OF CAPACITY

se		Metric	
ao	equals	0.0104	Liter
	"	0.1035	"
eng	"	1.0355	Liters
u	"	10.355	"
	"	51.7734	"
n or Picul	"	103.5469	"

MEASURES OF WEIGHT

Chinese		Metric	
1 Hao	equals	0.0037301	Gram
1 Li	"	0.037301	"
1 Fen	"	0.37301	"
1 Ch'ien	"	3.7301	Grams
1 Liang	"	37.301	"
1 Chin (catty)	"	596.816	"

20. Japanese Measures

van, = 3.75 kg or 8.2673336 lb avoir, is the unit of mass. It is divided 1 000 mommes, the momme into 10 funs, the fun into 10 rins, the rin into os, and the mo into 10 shis.

aku is the unit of length, equal to 10 ÷ 33 (0.30303) meter, or 0.99419 ft. It is decimally divided into sun, bu, rin, mo, and shi. Multiples

the shaku are: the ken = 6 shaku; the cho = 60 ken, and the ri = 36 cho = 12 960 shaku.

The shaku (land measure) = 0.00033 are (metric), or 0.9884217534 sq ft. 10 shaku = 1 go, 10 gos = 1 bu or tsubo, 30 bu = 1 sé, 10 sé = 1 tan, and 10 tan = 1 cho.

The shaku (capacity measure) = 0.01804 liter, or 0.0180616 U S liquid quarts. 10 shakus = 1 go, 10 gos = 1 sho, 10 shos = 1 to, and 10 tos = 1 koku (4.96 bushels).

Metric weights and measures are recognized as legal in accordance with the above equivalents.

21. Russian Measures

Funt, = 0.40951241 kg, or 0.90282028 lb avoir, is the basis of Russian weights. The funt is divided into 96 zolotniks or 32 loths, and the zolotnik is divided into 96 dolis; 40 funts = 1 pood.

Archine, = 28 in or 0.71120 meter, is the basis of length measure. The archine contains 16 verskops; 3 archines = 1 sagene and 500 sagenes = 1 verst.

Dessiatine, = 2 400 square saganes or 2.6997245 acres, is the unit for land measure.

Tchetvert, = 2.09908 hectoliters, or 5.95666 U S bushels, is the unit for dry measure. The tchetvert is divided into 8 tchetveriks, and the tchetverik into 8 garnets. 12 tchetverts = 1 last.

Vedro, = 12.2993285 liters, or 3.249119 U S gal, is used for liquids.

22. Spanish-American Measures

Arroba = 25.4 lb av = ¼ quintál = 11.42 kg (also, in Spain, 1 arroba = cántara = 4.2 U S gal approx).

Tonelada = 20 quintales = 2 032.2 lb av = 921.8 kg.

Vara = 2.75 ft (Peru), 2.841 ft (Argentina), 2.782 ft (Spain).

Estaca (Peru) = 167 meters sq = 27 889 sq meters.

Fanega (Peru) = 1.65 acre (varying locally); also = 141.43 lb av = 64.14 kg; also = 1.599 bushel (in Mex = 2.577 bushels).

Pertenencia, or mining claim (Mexico) = 167.6 m horiz on strike of vein, width of 94.28 to 167.6 m, according to dip.

Weights used for silver and gold in Peru: 3 granos = 1 tomin, 12 tomines = 1 adarme; 16 adarmes = 1 onza; 8 onzas = 1 marco; 2 marcos = 1 libra (= 1.014 lb Spain and Mexico).

1 marco = 50 castellanos = 230.07 gm = 3550.54 gr = 0.507 lb av. 1 onza = 1.23 oz Troy. 1 oz Troy = 6.755 castellanos.

In Mexico, the silver contents of an ore is expressed in MARCOS per CARGA. 1 marco per carga = 6.571 oz Troy per ton.

22. Values of Foreign Coins

ollowing equivalents, in terms of the U S gold dollar, were established by Secretary of the Treasury on Jan 1, 1911, for estimating the value of foreign handise exported to the U S and expressed in such metallic currencies.

Country	Monetary unit	U. S. equivalent
entine Republic	Peso	$0.965
tria-Hungary	Crown	.203
ium	Franc	.193
via	Boliviano	.389
il	Milreis	.546
th America (except Newf'nd). tral Amer. States	Dollar	1.000
osta Rica	Colon	.465
ritish Honduras	Dollar	1.000
uatemala, onduras, icaragua, lvador	Peso *	.403
e	Peso	.365
	Amoy	.661
	Canton	.659
	Cheefoo	.632
	Chin Kiang	.646
	Fuchau	.612
	Haikwan	.673
	Hankow	.619
	Kiaochow	.641
	Tael* Nankin	.655
a	Niuchwang	.620
	Ningpo	.636
	Peking	.645
	Shanghai	.604
	Swatow	.611
	Takau	.665
	Tientsin	.641
	Hongkong	.435
	Dollar* British	.435
	Mexican	.438

Country	Monetary unit	U. S. equivalent
Colombia	Dollar	$1.000
Denmark	Crown	.268
Ecuador	Sucre	.487
Egypt	Pound (100 piasters)	4.943
Finland	Mark	.193
France	Franc	.193
Ger'an Empire	Mark	.238
Great Britain	Pound	4.866½
Greece	Drachma	.193
Haiti	Gourde	.965
India (British)	Rupee	.324½
Italy	Lira	.193
Japan	Yen	.498
Liberia	Dollar	1.000
Mexico	Peso	.498
Netherlands	Florin	.402
Newfoundland	Dollar	1.014
Norway	Crown	.268
Panama	Balboa	1.000
Persia	Kran	.1704
Peru	Libra†	4.866½
Philippine I'ds	Peso	.500
Portugal	Milreis	1.080
Russia	Ruble	.515
Spain	Peseta	.193
Straits Settle't	Dollar	.421
Sweden	Crown	.268
Switzerland	Franc	.193
Turkey	Piaster	.044
Uruguay	Peso	1.034
Venezuela	Bolivar	.193

* Silver standard; other countries have the gold standard.

† Silver peso (sol) is also used in Peru.

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